Proceedings of the 1998 Coal Operators' Conference

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Proceedings of the 1st Australasian Coal Operators Conference COAL98

E Y Baafi
K Cram
G A Gibson
P Hanna
Editors

University of Wollongong, NSW

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The Australasian Institute of Mining and Metallurgy, University of Wollongong and NSW Coal Mines Managers Association are not responsible for the facts and opinions expressed in this volume.

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Proceedings of 1st Australasian Coal Operators Conference

COAL98

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The success of the Conference would not have been possible without cooperation of the conference participants, authors, presenters, reviewers of technical papers, session chairs, sponsors, exhibitors, and the dedication of the workshop presenters. Bob Kininmonth, Mining Consultant deserves recognition and gratitude for his valuable editorial comments. Sincere thanks to Ryan Baafi, undergraduate student, University of Wollongong for the time and effort in formatting the entire Conference volume. The organising committee appreciates the assistance given by the audio-visual support person Barry Robson and University of Wollongong mining students Majid Ataee-Pour, Ruibao Feng, Ben Smith and Dave McKinnon. The committee is very grateful to management of Port Kembla Coal Terminal, BHP Blast Furnace, Appin EDL Gas Utilisation Plant and Tower Colliery for making the field visits possible.

A conference of this magnitude will not be possible without a good manager. James Cook, Conference Manager, Wollongong Unicentre worked tirelessly within and outside social working hours to ensure the best outcome. The organising committee is indebted to James for his managerial and organisational skills which facilitated the planning and running of COAL98.

*****

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COAL98 is the first industry-oriented Australasian operators conference devoted to coal mining. The theme *Improving Fundamental Practices* was adopted because the Organising Committee believes that such improvement is critical to the industry’s future and viability particularly here in the Illawarra region in NSW where the coal industry is a major contributor in providing employment and revenue. Since the mid-1800’s the coal industry has been a major contributor to the Australia conomy. The coal industry is currently under extreme pressure as easier won resources diminish and global forces continue to contain real prices. What operating practices will enable us to ensure the future viability of the industry throughout Australasia? COAL98 will make significant contribution to the development of the industry over the next few years as it will provide participants from the industry with valuable information to keep abreast with current developments. The information will also challenge participants to review work practices and encourage implementation of the best operation practices. Hopefully COAL98 will be a vehicle through which your firm can develop and expand.

The technical papers which are presented in this volume are expected to cover the ambit of coal mining; from exploration to operations and through to shipping. The papers have been prepared by many of the industry’s leading and dynamic operators and practitioners and there is no doubt their input will induce critical thinking and encourage creativity. While some ideas will be easily embraced, some thought-provoking ones will lead to energetic debate and discussions which in itself could be a source of knowledge.

COAL98 will be opened by Senator Warwick Parer, Federal Minister for Resources followed by two Keynote Addresses from Kevin Crutchfield (incoming President of Cyprus Australia Coal Company) and John Maitland (National General President of the United Mine Workers Division, CFMEU). The speakers will outline the issues facing the industry from the perspectives of government, operators/corporate, and employees/unions. It is expected that the Keynote Addresses will deal with projected global marketing trends and developments, with particular mention of the current Asian economic crisis. An assessment of the potential impact of the crisis on Australian coal operations will be addressed. The speakers will seek to reinforce the need to further improve productivity at mines and reduce costs throughout the entire coal chain.

Another Keynote Address on Mine Safety by Bob Martin (NSW Minister of Mineral Resources and Fisheries) on Day 3 of the Conference, will address all the relevant issues that impact on and contribute to the future success of the industry.

The Conference activities include three one-day workshops covering the topics *Underground Coal Mine Planning* (offered by ECS International Pty Ltd), *Geotechnical Design of Coal Mines* (offered by Strata Control Technology Pty Ltd) and *Coal Gas Measurement and Control* (offered by GeoGAS Sytems Pty Ltd). Field trips to Port Kembla Coal Terminal, Blast Furnace, local coal mines and the Appin EDL Gas Utilisation Plant have also been arranged.

To keep the industry abreast with the ever-changing technology, it is planned for subsequent conferences in this series to be held every two to three years at an Australian major coal mining locality, the future organisers will definitely need your valuable support.

Ernest Y Baafi  
*COAL98 Conference Convenor*  
University of Wollongong  
February, 1998
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Enterprise Bargaining and Agreements under the Workplace Relation Act, 1996 and their Appreciation to the NSW Coal Mining Industry

J Whale

ABSTRACT

The Australian industrial relations system is in a state of constant change. We are undergoing a transformation from a compulsory arbitration and an award based system to one which is fundamentally rooted in the concepts of enterprise bargaining. This is true both at the State and Federal level. The structural changes which facilitate the movement away from the predominance of Awards to enterprise agreements has been fostered by both sides of the political spectrum and is a cornerstone of their respective industrial relations platforms.

In determining what form or combination of forms of industrial instrument(s) should apply to the individual enterprise, employers must have regard to the viability and sustainability of each form in the changing industrial climate. In particular employers should have regard to the needs and agendas of their employees and unions which may purport to represent them.

This paper traces the Federal legislative changes and fleshes out the industrial instrument options available to parties under the Workplace Relations Act, 1996 and gives an industrial relations practitioner's perspective upon the viability of those options and how they match contemporary human resource practice.

INTRODUCTION

Whilst the Workplace Relations Act, 1996 ('the Workplace Relations Act') is the first Liberal/National Party coalition Government attempt at addressing the balance between organised labour and employers its' fundamental structure is rooted in the concept of decentralised industrial relations and collective bargaining at the enterprise level which were embraced by the Hawke and Keating Labor Governments.

The Workplace Relations Act adopts the framework of collective bargaining in the form of certified agreements (CA) and provides alternatives to collectivism in the form of individual employment agreements in the form of Australian Workplace agreements and gives recognition to other forms of agreement, vis common law agreements.

The fabric of decentralised industrial relations and collective bargaining was encapsulated in the Industrial Relations Reform Act, 1993 ('the Reform Act'). The key structural components of the Reform Act in relation to forms of industrial instruments were:

- Facilitation of bargaining and agreements to provide for two types of agreement, namely certified agreements (Section 170 MA) and enterprise flexibility agreements (Section 170 NA).

- Immunity from certain civil liabilities, a right to strike and a right by employers to lock out employees (Division 4 Section 170 PA) in the course of certain bargaining arrangements.

- Provision for boycott conduct sanctioned by the Act and not subject to proceedings under the Trade Practices Act. Restructuring the role of the Commission and the Court. This entails the establishment of a bargaining division under the Commission.

1 J Whale and Associates , Sydney

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The adoption of the no disadvantage test against which agreements were assessed to ensure that on balance employees were no worse off than they would have been by reference to the appropriate award. The establishment of awards as "safety net" conditions to protect the industrially weak.

Under the Reform Act the Australian Industrial Relations Commission was required to facilitate the conclusion of enterprise agreements and not, as had been the legislative framework prior to the Reform Act, to establish the terms and conditions of employment by arbitration. The Reform Act severely limited the ability of the Commission to intercede in the negotiation of agreements or indeed the tactics employed by the negotiating parties. Inclusion within the Reform Act of limited immunity from civil liability and the operation of the Trade Practices Act and the concept of protected industrial action had the effect of limiting both the ability of the effected employer and the Commission from instituting action which ameliorated the consequences on industrial action.

The Reform Act provided for a limited the employer response to industrial action in the form of a lock out having denied to it previous common law and statutory rights. The sceptics amongst us may be of the view that the Reform Act was written by the union movement for the union movement. The net result in relation to the negotiation of agreements in those industries where unions had an established role was that the protection afforded under the Reform Act to unions acting as a negotiating party were used to full and I would suggest damaging effect. The Workplace Relations Act variously came into effect from 31 December, 1996.

The key provisions of the Workplace Relations Act in relation to the negotiation of agreements are directed at:-

1. Providing employers with choices as to the form of industrial instrument to apply at the enterprise level.
2. Enhancing the employers ability to negotiate by pegging back the scope and content of Awards to twenty allowable award matters, matters incidental thereto and exceptional matters thereby expanding the scope of matters that may be negotiated.
3. Retaining the ability of negotiating parties to engage in protected industrial action but in so doing limiting the scope of industrial action.
4. Removing the rights of unions to participate "by right" in the negotiation of and being a party to certain forms of agreement, for example an Australian Workplace Agreement, and providing an ability to represent members where sought by the member but not otherwise.
5. Restoring civil liability and the application of the Trade Practices Act other than in relation to certain protected industrial action.
6. Limiting the ability of parties to Agreements to engage in industrial action during the term of an agreement to which they are a party.

The Workplace Relations Act recognises the following forms of agreement:

1. Enterprise agreements with registered trade unions in accordance with Sections 5A, 170LH and 170 LJ which must be certified by the Australian Industrial Relations Commission (the Commission). Employers need not be bound by an Award, State or Federal, as a precondition to entering into one or more of these forms of agreement as the power relied upon emanates from the corporations power of the Constitution.
2. Certified agreements directly with employees under Sections 5A, 170LH and 170LK. As in the case of 1 above such agreements must be certified by the Commission and need not be based upon the precondition of an applicable award.
3. Australian Workplace Agreements ('AWA's') directly with their employees under Sections 170VC, 170VF and 170VG. AWA's must be filed with and approved by the employment advocate.
4. Agreements made pursuant to Section 113A, that is, an agreement made pursuant to an enterprise flexibility provision of an Award. This includes Clause 20 agreements of the Coal Mining Industry (Production and Engineering) Interim Consent Award, 1990. This form of agreement will cease to be available with the
5. Common law employment contracts directly with employees. Viable common law agreements will need to contain provisions which are no less favourable than are provided by the relevant industry award.

ANALYSIS OF THE WORKPLACE RELATIONS ACT AND ITS PRACTICAL APPLICATION

The Reform Act structurally altered the Federal Industrial Relations System by diminishing the role of the Commission in the setting of employment conditions and in lieu thereof establishing a safety net of conditions. It marks the turning point in terms of the promotion of collective bargaining at the enterprise level and supports that system by enabling parties to the negotiation of a collective certified agreement to engage in protected industrial action.

In practical terms few enterprise flexibility agreements were concluded in the period from the enactment of the Reform Act in December, 1993 to its repeal in December, 1996.

Certified agreements on the other hand were entrenched across industry generally and covered the majority of unionised employees.

A significant feature of the Reform Act was that it promoted the role of unions.

In highly unionised industries, such as the coal mining industry, unions engaged in pattern bargaining and adopted a minimalist position to change.

That is, employers seeking to negotiate a certified agreement experienced intransigence to new ideas and resistance to change.

In the coal mining industry the content of certified agreements largely reflected the award but with lip service being given to concepts of continuous improvement, benchmarking, best practice, etc.

In practically all cases agreements were negotiated through union offices and reflected a minimalist approach to collective bargaining. That is, the outcome of collective bargaining negotiations as reflected in the text of certified agreements is substantially restricted by union policies and are not what I would describe as enabling agreements ".

In particular through the negotiation process coal mining industry unions evidenced an aversion to the following concepts in the context of negotiating certified agreements:-

- performance based remuneration systems;
- individual employee performance assessment;
- a reduction in total earnings which more reflects the economics of the individual employer;
- direct relationships between the employer and the individual employee;
- flexibility in roster systems remunerated as applied in other industries (e.g. metalliferous mines);
- methods of selecting which employees would be made redundant in the event of a need to do so;
- the role and relevance of seniority;
- trading off award entitlements (inclusive of reducing existing entitlements in lieu of other benefits);
- salary schemes and in particular salary packaging;
• changing the role of trade unions at the enterprise level;
• the abandonment of pre-existing work practices and manning of certain tasks;
• cross streams or indeed one class of employee paid differentially according to skills, competency, productivity;
• recruitment on the basis of merit and other than on the basis of union membership;

On the other side of the equation unions were particularly keen and successful in securing for their members at the workplace improved wages via increases in base rates, increases in coal bonus payments which over time have lost their true relationship to performance, increases in overtime rates, improved sick leave and annual leave payments.

These improved terms and conditions were generally negotiated for some improved flexibility and relaxation of pre-existing demarcations rather than fundamental change.

Some agreements purported to change the culture of the workplace and to adopt contemporary human resource principles.

My observation is that whilst the words of the Agreement may have supported those sentiments delivery of them at the coal face was far from satisfactory if not illusory.

The coal mining industry also experienced significant industrial action in the course of negotiating certified agreements.

Instead of protected industrial action being a tool of last resort, a number of coal mining unions regarded it as the virtual first port of call to soften up the employer.

As a consequence of the repeal of sections of the Trade Practices Act and the limitations placed on the Courts in relation to industrial torts, employers suffering industrial action were unable to access the Court system or indeed obtain orders from the Commission relieving them from the effects of the industrial action.

They were left with three options, sit it out, negotiate or lock out.

This predisposition to industrial action is no doubt a product of the confrontationalist industrial history of the industry and an exhibition of the lack of maturity of the negotiating parties operating under the new system.

As a consequence certified agreements were negotiated under threat or indeed actual industrial action which was not conducive to changing the culture of either employers or employees in the long term or indeed to coming to grips with the issues which affect the viability of the business.

The certified agreement option

This analysis leads me to ask; what is different under the Workplace Relations Act that may or could result in a different outcome to that previously experienced by employers?

What has changed under the Workplace Relations Act which makes a certified agreement an attractive option to employers in the NSW coal mining industry?

Apart from providing employers with the option of negotiating directly with employees rather than via a union and perhaps the winding back of union preference provisions, I see that in a practical sense little has changed.

If an employer intends to negotiate a certified agreement with union involvement I venture to suggest that the outcome will be substantially the same as under the Reform Act.

Whilst certified agreements can be negotiated directly with employees unions have certain rights or representation. In a highly unionised industry it is unlikely that employees would negotiate without direct union involvement.

The negotiation of a certified agreement under the Workplace Relations Act is subject to the preparedness of employees in a collective sense and the Union movement to adopt change.
Substantially employee attitudes are moulded by Union policy which in New South Wales has been an impediment to achieving change.

A pre-condition to achieving change is therefore a preparedness by Unions to moderate their claims and facilitate structural change.

Where companies can point to change it comes at a price which has not worked to the best advantage of those Companies.

The Certified agreement option is however the least line of resistance option. It results in the employer "playing the game" and leaves open to unions the ability to set the agenda and participate in a industrial action, protected industrial action of course.

In the longer term Certified agreements are likely to be around for many years to come and encapsulated in industrial statutes, irrespective of the government of the day.

Should a Company select this type of industrial instrument there is every chance that they will achieve progressive change as changes occur within the union movement in relation to the role of unions in contemporary society.

Unions have changed and will continue to do so.

Consequentially the scope of Certified agreements will change and will be an attractive option to some employers.

The Workplace Relations Act does however provide other alternatives.

**Common law employment agreement option**

Recognition under the Workplace Relations Act is given to common law agreements between an employer and the individual employee or perhaps groups of employees.

Individual common law employment contracts have been in existence well before centralised industrial relations systems and cover many employment relationships both in unionised and non unionised areas, most particularly in relation to technical, professional supervisory and clerical personnel.

Common law contracts have many attributes which include promoting direct dealing between the employer and the individual and flexibility.

A significant benefit of common law employment contract is that they are not subject to technical or complex legal procedures and may be varied be consent and/or terminated in response to changing circumstances.

Under a Common Law employment contract the parties may tailor a package which meets the needs of both the individual employee and the Company in relation to terms and conditions, pay structures, salary packaging, performance assessment, training, career planning, etc.

In practical terms common law employment contracts need to meet on balance the conditions that would otherwise apply to the individual.

A number of employers have succeeded in encapsulating within the text of the common law employment agreement provisions which address employment security, litigation both in relation to the employment relationship and in relation to common law/criminal law proceedings and discriminatory provisions which exist in the relevant Award.

A key to the success of the common law employment agreement is therefore that it not only meets the award as a minimum but contains mechanisms by which the employee may, with some confidence, have grievances resolved at no cost to the employee.

The difficulty with common law employment agreements is that where an award applies to the employee, award provisions cannot be technically off set and their efficacy in the face of employees or a union(s) seeking to negotiate a collective agreement has not been tested.
That is, the existence of common law employment agreements does not on the face of it provide a defence or protection from employees or unions seeking to negotiate a certified agreement or Australian Workplace Agreements and initiating bargaining a period and engaging in protected industrial action.

An employer that intends to rely upon common law employment agreements therefore runs the risk of having them overturned by subsequent collective action by employees and/or unions.

Ultimately the overall level of benefits contained in the common law agreement, the ability of the support structure to meet employee needs and the ability of union(s) to organise and raise the level of expectation for a better deal determines the viability of common law agreements.

One of the significant limiting factors in expanding the coverage of common law agreements is the lack of exposure to them by non technical, professional et al personnel to them and the existence of other alternatives, notably certified agreements.

For many common law employment contracts whilst attractive will not be achievable in the short to mid term unless other cultural and/or structural changes occur which instil them as an effective alternative to collective action or other industrial instrument options.

Agreements under the NSW Industrial Relations Act, 1996

Whilst provision exists under the Workplace Relations Act for recognition of agreements under the appropriate State Act, there appears little attraction in pursuing this option in lieu of others available under the Workplace Relations Act.

Provision is made for enterprise agreements under the Industrial Relations Act, 1996 NSW.

The form of such agreements differs marginally from that under the Federal system and the same tests apply

Australian Workplace Agreement

Australian Workplace Agreements (AWA's) are a new form of agreement.

In substance AWA's are individual employment agreements between the employer and the employee.

A union cannot be a party to an AWA but may act as a bargaining agent for members and indeed non members. AWA's represent "the brave new world".

They rely both upon the corporations and industrial relations powers of the Constitution and under the Workplace Relations Act are subject to the same form of tests as apply to certified agreements.

An AWA is predominantly and employer generated document.

The negotiation of an AWA is a matter of choice. Neither the employer nor the employee is bound to negotiate on the document.

Parties to the negotiation of an AWA may engage in protected industrial action and are afforded the same immunity from prosecution as applies to parties negotiating a certified agreement.

The scope of an AWA is not limited other than it must address matters which relate to the employment relationship.

AWA's have a number of structural draw backs.

Firstly, experience would suggest that the turn around time between submitting the AWA to the employment advocate and receipt of the approval notice may amount to several months.

This time may be lessened by the employer presenting to the employment advocate a draft or pro forma agreement and
engaging in dialogue with the employment advocate as to difficulties with the content or procedure intended to be applied.

Secondly the negotiation of an AWA follows a strict course and is subject to legislative and bureaucratic procedures and tests.

In the absence of the employee being covered by an award (State or Federal) the employer must apply to the employment Advocate to nominate an appropriate Award.

Where the AWA is not being offered to all employees and on the same terms justification for this decision is required.

AWA’s have a finite term and cannot be terminated prior to their nominal expiry other than by either party following an exhaustive and protracted procedure.

From a union perspective AWA’s are "anti-union" as they do not institutionalise the role of unions in their negotiation or ongoing administration.

One of the features of an AWA is that an employee cannot be compelled to accept the terms offered.

From a positive perspective AWA’s enable the employer to deal directly with individual employees and negotiate face to face rather than through a third party.

In addition the employer is able to encapsulate within the AWA contemporary human resource principles and concepts in very much the same way as is available to it under the common law employment agreement option.

AWA’s are therefore likely to be attractive where the employer has developed a rapport with employees and has a mature industrial environment.

Where suspicion and mistrust abound there is unlikely to be the climate for individual agreements.

**THE FUTURE OF THE VARIOUS FORMS OF AGREEMENT**

From the analysis of the legislative framework it is clear that both Labor and Conservative governments have adopted a market orientation where the establishment of employment conditions at the enterprise level is a key and irreversible element.

Australia it would appear has shed the centralist approach to industrial regulation in lieu of enterprise regulation supported by statutorily set minimum safety net terms and conditions.

Trade union membership particularly in relation to staff has been on the decline for many years and increasingly employment conditions are being set at the enterprise or workplace level. The decline in union membership has been offset by an increase in the number of employees covered by common law employment agreements.

This trend is almost exclusively restricted to Staff but there have isolated instances in New South Wales where other mineworkers have elected to resign from their union.

I have suggested that the extension of common law employment contracts to non technical, professional, supervisory and clerical personnel requires a number of preconditions. Those preconditions are not evident at this time and therefore one can hardly expect a rush to adopt common law employment agreements.

One can reasonably expect that the incidence of common law employment agreements will continue to expand in that sector of the industry in which they are established and that conservative Governments may progressively amend legislation to give them greater recognition.

A future Federal Labor Government is unlikely to enact legislation which provides that common law employment agreements prevail over other forms of collective agreements but is also unlikely to legislate to remove their ability to exist.
The future of collective agreements must be assured either under future Labor or Conservative Governments as both parties have similar political platforms in relation to this form of industrial instrument.

What may change is the relative role of unions in the negotiation of collective agreements and the legislative protection afforded to negotiating parties.

The future of AWA’s I would suggest is substantially in the lap of employers.

Should employers not adopt and effectively implement AWA’s with the net result that their incidence is minimal, a future Labor government would have little difficulty or compunction in repealing that part of the Workplace Relations Act which provides for them.

If on the other hand they are an established form of industrial instrument at the time Labor next comes to power, the ability of a Labor Government to bring their existence to an end is far more difficult.

CONCLUDING REMARK

The future of industrial regulation is substantially in your hands.
APPENDIX 1

LEGISLATIVE FRAMEWORK

The Workplace Relations Act, 1996 (the 'Workplace Relations Act') is the first attempt by a Liberal/National Party coalition Government in more than twenty years to tackle the Federal industrial relations framework and to address the balance between the stakeholders subject to the Federal system.

From 1988 and prior to the enactment of the Workplace Relations Act changes to the Federal industrial relations system and regulation had occurred under the auspices of successive Labor Governments.

The Industrial Relations Act, 1988 and successive amending legislation was directed at shifting the role of the Australian Industrial Relations Commission away from arbitration to one of promoting direct dealings between parties, resolving disputes through conciliation and establishing minimum safety net employment conditions.

To appreciate the opportunities which present themselves today under the Workplace Relations Act we need to comprehend the structural changes which have occurred to the legislative framework, the thrust of those changes and their continuation and modification under the Workplace Relations Act.

We need also to assess the future legislative framework so that whatever form of industrial instrument we implement today is sustainable into the future.

Industrial Relations Reform Act, 1993

The Industrial Relations Reform Act, 1993 ('the Reform Act') was a key piece of legislation in this move.

The Reform Act sought to superimpose onto the Australian experience a form of collective bargaining which, until that time, was primarily associated with Continental America, and European countries and Governments whose history was influenced by those countries.

The major changes arising under the Reform Act were:

1. Establishment of statutory minimum employment conditions
2. Amendments to the Award system with the result that Awards were to provide a safety net which underpin direct bargaining
3. Facilitation of bargaining and Agreements to provide for two types of Agreement, namely Certified agreements (Section 170 MA) and Enterprise Flexibility Agreements (Section 170 NA).
4. Immunity from certain civil liabilities, a right to strike and a right by employers to lock out employees (Division 4 Section 170 PA) in the course of certain bargaining arrangements Provision for boycott conduct sanctioned by the Act and not subject to proceedings under the Trade Practices Act
5. Restructuring the role of the Commission and the Court. This entails the establishment of a bargaining division under the Commission.

For the purposes of this paper we do not develop point one above, vis minimum employment conditions, but in relation to this aspect the Reform Act generally codified principles relating to the dismissal of employees and encapsulated within the legislative framework International Labour Organisation Conventions.

Amendments to the Award system

In relation to Awards the Reform Act provided a conceptual basis whereby Awards would provide a safety net underpinning a system of collective bargaining wherein wages and other employment conditions would be set at the enterprise level.
To give effect to this intent the Reform Act establishes a separate Division of the Commission to manage and facilitate bargaining and the negotiation of workplace agreements.

The objects of this part of the Reform Act are to ensure as follows:

1. Employees are protected by Awards that set fair and enforceable minimum wages and conditions of employment
2. Awards act as a safety net underpinning direct bargaining
3. Awards are suited to the efficient performance of work according to the needs of the enterprise or industry while employees interests are also properly taken into account
4. Regard is had, in conjunction with making, reviewing and varying awards, to stable and appropriate relativities based on skill, responsibility and the conditions under which work is performed, and on the need for skilled based career paths
5. The Commissions functions and powers give employees prompt access to fair and enforceable minimum wages and conditions and encourages the prevention and settlement of industrial disputes by the making of enterprise agreements.

Importantly the content and scope of Award provisions was not limited or prescribed by statute.

In the exercise of its powers and role the Commission was required to rely as far as practicable on conciliation and to consider, in dealing with matters, the interests of the parties and the Australian community as a whole.

The Reform Act required in effect that the Commission promote enterprise flexibility and enterprise efficiency and in so doing guard against provisions which reduce employee conditions, or where so affected is not against the public interest.

Promotion of bargaining and facilitation of agreements

The Reform Act provided for two types of Enterprise Agreement being Certified agreements (Part VIB Division 2) and Enterprise Flexibility Agreements (Part VIB Division 3).

Certified agreements may have been made in response to an industrial dispute or an industrial situation.

Enterprise Flexibility Agreements were Agreements made directly between the employer and its employees. Such Agreements did not require union involvement however an eligible union had rights to represent the interest of members and, should it so choose, be a party to the Agreement.

The ability of the Reform Act to cover enterprise Flexibility Agreements arose from reliance on the corporations power of the Constitution and required that the employer must be a constitutional Corporation to enter into such an Agreement.

The Reform Act establishes certain tests to ensure that parties covered by either Certified agreements or Enterprise Flexibility Agreements are not disadvantaged in respect their terms and conditions of employment.

The no disadvantage test is an "on balance" test taking the Agreement as a whole, not term by term and is applied having regard to the Award applicable to the employees.

Immunity from Civil Liability

Division 4 of the Reform Act gives effect to what are stated to be international obligations to provide for a right to strike and legislative protection for the exercise of that right subject to certain limitations.
In relation to parties negotiating a Certified agreement, Section 170 of the Reform Act provided for the right to strike where:

1. there exists an industrial dispute involving an employer and one or more organisations members of which are:
   - employed by the employer to perform work in a single business, part of a single business or a single place of work; and
   - are covered by an Award; and

2. the employer and one or more of those organisations are negotiating an Agreement under Division 2 Part VIB

Procedurally an industrial dispute must exist and one of the negotiating parties had initiated a bargaining period (minimum of 7 days) and given the requisite notice (72 hours) of the intention to engage in protected industrial action (Section 170 PG) as contained in a notice to the other party on a specified date(s).

Providing all the requisite steps have been followed and the parties have, before engaging in industrial action, negotiated in good faith, either party could engage in industrial action which was protected under the Reform Act. The action however must be directed at advancing claims made against the other party pursuant to concluding a Certified agreement.

The termination of industrial action other than by action of the parties may be brought about by action of the Commission, that is to terminate or suspend the bargaining period (Section 170PO). Where the Commission terminates the bargaining period it is obliged to arbitrate and make an Award for a fixed period.

The ability of negotiating parties to a Certified agreement to engage in protected industrial action and enjoy immunity from civil proceedings is supported by general legislative changes which repeal Section 45E and alter Section 45D of the Trade Practices Act.

The Reform Act in relation to Section 45D of the Trade Practices Act provided that in determining whether Section 45 D(1) of that Act had been breached certain conduct, termed 'boycott conduct' would be disregarded.

Included in that category were:
- a boycott contravention
- attempting to commit a boycott contravention
- aiding, abetting, counselling or procuring a person to commit a boycott contravention
- inducing, or attempting to induce a person (whether by threats, promises or otherwise) to commit a boycott contravention
- conspiring to commit a boycott contravention.

Conduct that was protected under the Reform Act extended to persons/organisations (unions and employees of the employer) who engaged in such action where the ultimate purpose of the conduct was substantially related to:
- the remuneration, conditions of employment, hours of work or working conditions of the employee(s) working for that employer; or
- relates to the termination or action directed to the termination of a person employed by that employer.
In addition to the protection otherwise afforded, Section 164 provided immunity from prosecution under a State or Territory law and Section 166A provided that Tort action did not lie against an officer of a union or employee engaged in action in furtherance of claims that were the subject of an industrial dispute relating to the negotiation of a Certified agreement.

**Restructuring the commission and the court**

The Reform Act restructured the Commission and the Industrial Court system.

In relation to the Commission the Reform Act created a Bargaining Division whose functions as previously discussed relate to the negotiation of Agreements.

In relation to the Court structure the Reform Act established a new Federal Court, the Industrial Relations Court Australia. This Court with supportive Judicial Registrars was given certain powers previously exercisable by the High Court and original jurisdiction in relation to such matters as boycott conduct, industrial action, lockouts, etc.

A major role of the Judicial Registrars and the Court related to employment termination proceedings.

**The Workplace Relations Act, 1996**

The Workplace Relations Act, 1996 ('the Workplace Relations Act') was assented to on 25 November, 1996 and variously came into operation from 31 December, 1996.

In many respects it builds upon the concepts developed in the Reform Act but in so doing attempts to redefine the power of Unions and employers in the bargaining process and the protection afforded to negotiating parties.

The Workplace Relations Act has been called many things.

One view is that the legislation is "pro choice" adopting a legislative framework which seeks to facilitate the negotiation of Agreements and enhance flexibility at the enterprise level.

The explanatory note accompanying the Workplace Relations and Other Legislation Amendment Bill states as follows:-

- "legislative reforms are directed at supporting a more direct, co-operative relationship between employers and employees and greater labour market flexibility." And
- "the Act is framed to give primary responsibility for industrial relations and agreement making to employers and employees at the enterprise and workplace levels." and
- "the industrial relations system needs to provide them with effective choices about arrangements which suit their particular circumstances. The Act provides such choice as well as a fair go for all."

The objects of the Workplace Relations Act state that the principal object of the Act is to provide a framework for co-operative workplace relations which promotes the economic prosperity and welfare of the people of Australia by:

1. encouraging the pursuit of high employment, improved living standards, low inflation and international competitiveness through higher productivity and a flexible and fair labour market; and
2. ensuring that the primary responsibility for determining matters affecting the relationship between employers and employees rests with the employer and employees at the workplace or enterprise level; and
3. enabling employers and employees to choose the most appropriate form of agreement for their particular circumstances, whether or not that form is provided for by the Act;
4. providing the means:
   - for wages and conditions of employment to be determined as far as possible by the agreement of employers and employees at the workplace or enterprise level, upon a foundation of minimum standards;
   - to ensure the maintenance of an effective award safety net of fair and enforceable minimum wages and conditions of employment;

5. providing a framework of rights and responsibilities for employers and employees, and their organisations, which supports fair and effective agreement making and ensures that they abide by Awards and Agreements applying to them;

6. ensuring freedom of association, including the rights of employees and employers to join an organisation or association of their choice, or not to join an organisation or association;

7.

8. enabling the Commission to prevent and settle industrial disputes as far as possible by conciliation and, where appropriate and within specified limits, by arbitration;

9. assisting employees to balance their work and family responsibilities effectively through the development of mutually beneficial work practices with employers;

10. respecting and valuing the diversity of the work force by helping to prevent and eliminate discrimination on the basis of race, colour, sex, etc; and

   assisting in giving effect to Australia's international obligations in relation to labour standards”.

From these objects it is clear that the Workplace Relations Act whilst adopting many concepts from the Reform Act places greater emphasis upon the direct relationship between the employer and employees rather than promoting the role of third parties, viz trade unions and employer organisations.

The Workplace Relations Act continues the process of redefining the role of the Commission and the Court commenced under the Reform Act.

In relation to Award making, the Workplace Relations Act restricts the subject matter that the Commission can arbitrate following the finding of an industrial dispute.

The scope on an industrial dispute and the matters that the Commission may arbitrate, other than in relation to an arbitration following the termination of a bargaining period under Part VIB Division 8 Section 170 MW(3) or (7), are defined by Section 89A and limited to twenty allowable matters, matters incidental thereto and to exceptional matters.

In relation to the matters subject to arbitration under Section 170 MX following the Commission terminating a bargaining period, the Commission is not so restricted or limited to matters defined in 89A but may deal with all matters (Section 170MY) the subject of dispute between the parties.

The Award making function of the Commission is restricted to making minimum rates Awards and to establishing safety net wages and conditions.

It is a clear intention of the Act to prune back Award provisions so as to provide greater scope of provisions that may be negotiated at the enterprise level. To that end one of the roles of the Commission is to review Awards for the purpose of removing non allowable matters.

This is important in the context of agreements contemplated under the Workplace Relations Act in that these struck out Award provisions are available to be incorporated in forms of agreements envisaged under the Workplace Relations Act. The proviso being that they do not offend Part XA of the that Act.
The Workplace Relations Act is primarily directed towards promoting the implementation of workplace or enterprise agreements.

The use of different legislative powers pervades the various forms of agreement Constitutional Corporations ('employer') may enter into. Other than State Awards (or employment agreements made under the appropriate State industrial relations legislation) or Federal Awards, an employer may enter into and regulate employment conditions by one or more of the following forms of agreement:

1. Enterprise agreements with registered trade unions in accordance with Sections 5A, 170LH and 170 LJ which must be certified by the Australian Industrial Relations Commission ('the Commission'). Employers need not be bound by an Award, State or Federal, as a precondition to entering into one or more of these forms of agreement as the power relied upon emanates from the Corporations power of the Constitution.

2. Certified agreements directly with employees under Sections 5A, 170LH and 170LK. As in the case of 1 above such agreements must be certified by the Commission and need not be based upon the pre-condition of an applicable award.

3. Australian Workplace Agreements (AWA's) directly with their employees under Sections 170VC, 170VF and 170VG. AWA's must be filed with and approved by the Employment Advocate.

4. Agreements made pursuant to Section 113A, that is, an agreement made pursuant to an enterprise flexibility provision of an Award. This includes Clause 20 agreements of the Coal Mining Industry (Production and Engineering) Interim Consent Award, 1990. This form of agreement will cease to be available with the imminent review of the Act.

5. Common Law employment contracts directly with employees. Viable Common law agreements will need to contain provisions which are no less favourable than are provided by the relevant industry award.

These forms of agreement are the basis upon which the Workplace Relations Act seeks to regulate workplace relations with Awards assuming the role of setting safety net minimum wages and conditions rather than being all encompassing and prescribing, supposedly, the maximum terms and conditions of employment.

By comparison with the former Industrial Relations Act, 1988 the new Act permits:

1. Companies to enter into collective Certified agreements directly with employees rather than through the auspices of the Union. Whilst this provision existed under the previous Act it has been developed so as to extend the choices available;

2. Companies to enter into individual employment agreements directly with an employee or groups of employees without union intervention. Note this relates to parties to agreements not the ability of a union to act as a bargaining agent for a member;

3. Companies to enter into Common Law employment contracts. Previously whilst available to employers such agreements were not recognised under the industrial law. The Act now gives limited recognition to them.

4. State agreements or awards prevailing over Federal agreements or awards

Each form of agreement is subject to certain tests, notably the no disadvantage test, Part VIE, Section 170XA and meeting minimum statutory requirements.

The changes to types of Agreement available and recognition of Common Law Agreements was accompanied under the Workplace Relations Act by a restoration of the industrial Torts, common law action and the full application of the Trade Practices Act other than in relation to parties negotiating a Certified agreement or an Australian Workplace Agreement where qualified protection exists.
The Workplace Relations Act retained the concept of protected industrial action and extended it to parties negotiating both Certified agreement and Australian Workplace Agreements. Other industrial action is, for the purposes of the Workplace Relations Act, illegal and unprotected.

The primary thrust of the Act in relation to the form of agreement to be implemented at the particular enterprise is directed at enabling the employer to be the activator of choice, not a union and not employees.

The Act achieves this by relying upon the Corporations powers under the Australian Constitution (Section 51 (xx)) in addition to the powers to the labour powers (Section 52 (xxxv)).

The use of the Corporations power is not novel in the industrial relations context and indeed the availability of Enterprise Flexibility Agreements under the Reform Act relied upon the this Constitutional power.

By utilising the joint labour and corporations powers the Act provides a vehicle to implement change at the enterprise level from that previously available under the Industrial Relations Act, 1988 and the Reform Act.

The Workplace Relations Act places an employer under no compulsion or obligation to agree to negotiate any agreement or indeed any particular form of agreement.
Merit Based Selection and Performance Assessment for Mineworkers

L Edmonds-Ward¹ and B Trendell²

ABSTRACT

While objective selection and assessments are an accepted part of employing managers and other staff, they have had only a limited place when selecting mineworkers in Australia. Wambo Mining Corporation has used occupational testing as part of its recruitment process since 1994. For managers and staff in particular, it is considered this has contributed significantly to a 95% fit of those appointees. In 1997 when Wambo undertook development of a new "on site" subsidiary underground mine called Wollemi Services, they wanted to select the most appropriate people in terms of skill and on the job performance. To achieve this they reviewed and improved their recruitment processes to facilitate selection and transfer of an initial intake of almost 50 staff and mineworkers.

One of the issues for Human Resource Management was to provide an environment where employees could let go of previously held (but not necessarily individually believed) entrenched views about individual performance and assessment. It needed to be emphasised that performance could be objectively and fairly assessed. More importantly, the performance being assessed was the application of skills and that new skills could be learnt and individuals could choose to change behaviour.

A process was agreed between management, employees and their local representatives to select and transfer people on merit from within shift groups. In the first intake, three supervisors and thirty-nine production workers were selected from an existing workforce of over two hundred.

Part of the process to ensure validity and to help people feel comfortable was an objective job analysis for positions. From a computer-based analysis, person specifications were developed and appropriate test batteries identified to facilitate selection. A combination of a self-report occupational personality or work styles questionnaire and several ability tests were used. In addition, each employee and two supervisors completed an assessment of the employee’s current work performance.

Candidates were provided with individual feedback about their self-assessments, performance feedback from supervisors and asked to respond to a number of questions about their interest in and potential contribution to the new operation. When selecting employees, assessments of skills and additional competencies were also considered.

The validity of the self-report assessments has since been confirmed in a correlation analysis of the results with supervisor feedback on performance. In addition the results have been analysed to identify development needs for all candidates.

It was essential that the overall process was confidential so that people would be prepared to participate and the vast majority of people took up the competitive challenge. In the four months since the process, there has been a significant breakdown of restrictive practices. As expected, there was a productivity improvement at the new site. In addition, at the existing mine there has been a significant realignment of individual performance with many individuals being dynamic, progressive and showing real responsibility in their work.

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² Director, Business Improvement Australia Pty Ltd
INTRODUCTION

When the management of Wambo was restructured it provided an opportunity to review and improve a number of strategies and systems. One strategy selected to improve business performance was through diligently implementing core values. These values were seen as a means of collectively creating the organisation's culture and helping to influence financial, strategic, safety and technological decisions, as well as managerial and leadership styles.

The values were integrity, people involvement, performance and safety. Integrity implied earning trust, treating employees with respect and being consistent and fair. People involvement included valuing skills, knowledge and ideas of employees as well as assisting them to contribute to optimal performance.

A recruitment and selection policy was developed in 1994 that was consistent with these values. The aim was to remove negative discriminators such as the industry norm of white, Anglo-Saxon male mineworkers as well as reliance on perceptions of people during an interview. Instead the policy provided for positive discriminators and fair access to the system.

The gender bias was removed by encouraging males to apply for traditional female roles in administration and support services as well as supporting females to apply for traditional male roles. Access was not dependent on current union membership, nor age or literacy. People known to staff, families of existing employees as well as those employees seeking career change were offered access to the process, but with no guarantee of an interview or position.

Aptitude and personality assessments were included in the recruitment and selection process. Tests were selected that were considered valid for the positions and soon after coal industry applicant norms were developed. Applications and assessment results were used to short list candidates for interviews. In addition, the panel of Wambo managers and employees were provided with specific information about candidate styles that could be validated during the interview.

In this paper, the design of a merit based selection programme and subsequent validation of assessment instruments are described. In addition, perceived benefits in performance are highlighted.

WOLLEMI SERVICES

When it was decided to develop a new mine, the preferred strategy was to employ for the Wollemi Mine from the existing Homestead Mine. The final numbers would be 187 from an existing workforce of 250 and with an initial intake of 40.

While position descriptions had been developed, the same rigour had not been applied to preparing person specifications. The process needed to consistent with the values of respect, fairness and integrity, but it was also important that a scientifically based approach was used that could stand up to any legitimate challenge.

The critical steps in the process were:

- Job analysis to identify personal attributes, abilities and relevant assessment methods;
- Appraisal of current performance and assessment of abilities and work style;
- Interview and feedback from appraisal and assessment; and
- Selection based on a matrix of skills and attributes.
JOB ANALYSIS

The job analysis tool used was the computer based Saville and Holdsworth Work Profiling System (WPS). Two managers/supervisors completed the process for each position. They identified job objectives, selected task categories, rated importance and time spent on tasks and indicated context of tasks. The method by which an attribute profile and listing of relevant assessment methods are produced from a WPS analysis is based on job component validity. The required attributes are linked to task statements on the basis of rigorously scrutinised judgements made by a panel of experienced occupational psychologists.

PERSON SPECIFICATION

The most important tasks for production were identified as:

- Watching for and noting safety hazards and other dangerous situations;
- Achieving team cooperation;
- Encouraging cooperation from others;
- Conducting operations; and
- Achieving standards.

Similarly for trades they were:

- Ensuring safety
- Undertaking tasks to examine, check and repair machinery;
- Watching and listening for dangerous situations;
- Making decisions; and
- Encouraging and gaining willing cooperation.

From these the most important personal attributes for individuals were described as:

**Production**

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<td>Caring (democratic)</td>
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**Trades**

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<tr>
<td>Persuasive</td>
</tr>
<tr>
<td>Socially confident</td>
</tr>
<tr>
<td>Caring (democratic)</td>
</tr>
<tr>
<td>Forward planning</td>
</tr>
</tbody>
</table>
ASSESSMENT METHODS

Assessment methods were classified and compared for relevance to the tasks (see Table 1.).

Table 1 - Work Profiling System Candidate Assessment Regime for Most Critical Attributes

<table>
<thead>
<tr>
<th>Class</th>
<th>Production Relevance Index</th>
<th>Trades Relevance Index</th>
</tr>
</thead>
<tbody>
<tr>
<td>Personality</td>
<td>324</td>
<td>211</td>
</tr>
<tr>
<td>Structured Interview</td>
<td>100</td>
<td></td>
</tr>
<tr>
<td>Creative thought</td>
<td>94</td>
<td>286</td>
</tr>
<tr>
<td>Diagrammatic skills</td>
<td>81</td>
<td>61</td>
</tr>
<tr>
<td>Manual dexterity</td>
<td>49</td>
<td>51</td>
</tr>
<tr>
<td>Mechanical skills</td>
<td></td>
<td>44</td>
</tr>
<tr>
<td>Verbal skills</td>
<td></td>
<td>32</td>
</tr>
<tr>
<td>Numerical skills</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

APPRAISAL AND ASSESSMENT

In order to rationalise the assessment process, creative thought, diagrammatic skills and manual dexterity were assessed using questionnaires that assess "co-ordinative abilities". All of the tests used were from the Saville and Holdsworth test batteries and were:

- Work Styles Questionnaire;
- Understanding Written Instructions;
- Working with Numbers;
- Matching Shapes;
- Mechanical Comprehension; and
- Visual Checking.

Appraisal of current performance was made using a questionnaire that was answered by the applicant, a direct supervisor and a shift supervisor. The questions addressed the person's willingness to work safely, work effectively, work in a team,
work with leaders, learn, apply skills and change. Each question consisted of five behavioural anchors that described specific behaviour or action for each score such as, “waits to be told to do a task” and “carries out all tasks that need to be done”.

At the time only one candidate refused to complete the test battery. Many others did however want to discuss or debate the validity of the tests. In particular there were negative perceptions of testing at other mines associated with the belief that the tests were the sole criteria for selection. Most comments were anecdotal.

INTERVIEW AND FEEDBACK

During interviews each candidate was given feedback showing a comparison of their own and supervisors ratings of current performance as well as a summary of their abilities (normed against their own stream and shift) and work style.

A few candidates became defensive about negative performance assessments and some had difficulty accepting below average ability results. Where possible, links between personality and abilities were explained to them. An example was to explain the significance of deliberate rather than rapid decision making for a person with low abilities.

As this was the first time that mineworkers had received structured feedback about perceptions of their individual performance, it was not unexpected that some people wanted to challenge the ability of supervisors to make fair judgements. It was emphasised to them that the appraisals showed both positive and negative responses and were often consistent between supervisors.

Overall, people seemed to welcome the opportunity to know where they stood, what their match was with the requirements for the new operation and being able to have managers listen to their personal views about what was required for successful performance.

SELECTION

A matrix was made up based on consensus among the selection panel. The items in the matrix included skills as a miner driver and with other machinery, interview, attendance, performance appraisal, abilities, work style, first aid and other skills. Applicants were then ranked overall. Final selection included additional criteria such as having a minimum of one competent miner driver capable of coaching others. In addition it was important to take into consideration the impact on the continuing short term operation of the Homestead mine.

ONGOING DEVELOPMENT

Following selection each shift participated in an intensive two-week induction programme that included skills training as well as communication and team development awareness. Several months after commencing, each team participated in an experiential team development activity.

Future activities will include ongoing team and individual performance feedback.

VALIDATION

The general validity of the assessment instruments has been based on past studies conducted by Saville and Holdsworth for a range of industries. The major barriers to any validation study are the number of job incumbents and suitable appraisal of on the job performance, but in this case a detailed appraisal of willingness to perform was available for over one hundred candidates.
The objective of validation was to determine if the scales of the self-report work styles questionnaire or abilities were correlated with appraisal of current performance by supervisors.

Several scales were significantly correlated with overall performance appraisal (Table 2). The results were generally consistent for the two supervisors within each stream. As the performance appraisal focused on "willingness" rather than skill, it was more likely that personality scales would be correlated rather than abilities. The ability tests that were correlated for trades, matching shapes and visual checking, are measures of co-ordination and flexibility.

The results show that several personality scales were valid indicators of willingness to perform.

Table 2 - Correlation between overall performance appraisal, personality and abilities (Correlation co-efficient significance ** probability = .01 * probability = .05)

<table>
<thead>
<tr>
<th>Personality Scales</th>
<th>Abilities</th>
</tr>
</thead>
<tbody>
<tr>
<td>Production Section Supervisor</td>
<td>Production Shift Supervisor</td>
</tr>
<tr>
<td>Controlling *</td>
<td>Low Gregarious *</td>
</tr>
<tr>
<td>Low Gregarious *</td>
<td>Low Traditional *</td>
</tr>
<tr>
<td>Low Traditional *</td>
<td>Emotional Control **</td>
</tr>
<tr>
<td>Achieving *</td>
<td>Matching Shapes **</td>
</tr>
</tbody>
</table>

CONCLUSION

The level of productivity at Wollemi was expected to be high as a result of work practices agreed in the Certified Agreement, selection, induction and enthusiasm associated with a new enterprise. The results to date have exceeded those expectations.

While improvements were also expected at Homestead through the Certified Agreement, there was some uncertainty about the impact of the appraisal and assessment process at an individual level for mineworkers not selected for the first intake. In the four months since the process, there has been a significant breakdown of restrictive practices. In addition, there has been a significant realignment of individual performance with many individuals being dynamic, progressive and showing real responsibility in their work.

The results from the validation study provide direct support for the validity of the self-report Work Styles Questionnaire as part of a selection battery. In circumstances where there is insufficient work history, less experienced employees can be compared with those with more experience. In internal situations, there is less resistance to self-assessment than for supervisor appraisals.
The fairness and objectivity of the overall process is reflected in an observation by one of the selected mineworkers during induction. The comment was “why did we waste all that time and money when I was right in picking eight of the thirteen because I knew they were good performers and liked by the supervisors”. He did not answer the response, “does that mean that five people were selected on merit whom you thought had no chance because they were not liked by the supervisors?”
Improving Colliery Performance Through One Big Team, Many Teams or.......???

C J Seaborn

INTRODUCTION

A cry that is often heard from front-line employees, particularly as a mine grows or circumstances change is "We used to be one big team but now they have set up teams for different functions - we don't know what's going on and we don't share resources any more. We used to just get in and do what had to be done, now we can't.". It usually occurs after management decides that, with the increasing mine complexity, functions and accountabilities of a manageable size need to be identified and defined for individuals and teams to facilitate optimum performance. However, as the statement indicates, these changes have often not delivered all the intended benefits. This paper will explore the reasons why the full benefits have not been realised and will identify actions which can be taken to improve the likelihood of optimal outcomes.

ORGANISING THE MINE

One big team

Historically, until recent years, many Australian coal mines, particularly those in the underground sector, were organised along the "one big team" concept. This concept, which is graphically illustrated in Fig. 1, which illustrates, loosely defined accountabilities within large work groups involved the following:

- loosely defined accountabilities for individuals and crews within large work groups;
  membership of crews could change quite regularly as a result of factors such as roster arrangements, absentee replacement requirements and overtime needs;

- employees, whilst being notionally in one work area/function, could turn up for a shift and find themselves allocated to another role and area of the mine;

- leaders had wide spans of accountability (eg undermanager for a shift, mine manager for both long and short term decisions).

An illustration of the impact of the "one big mine" concept was the industry's response to the 1988 Coal Industry Tribunal (CIT) decision. In this decision, the companies were given the right to carry out various production activities on weekends. While from a cost point of view it would have made sense to employ only a small number of people on critical activities on a weekend related roster, the prevailing industrial climate in 1988 required everyone to be treated equally and as a result most employees were put on expensive six or seven day rosters. Often, these rosters involved splitting up small close-knit operating crews for part of the roster and having operating management personnel on different rosters altogether. Hence, the teamwork of the previous five day crews was broken down and line management accountability for issues such as operational planning and communication was often lost.

1 Principal SOS Initiatives Pty Ltd
The fact that, when the industrial climate changed, many mines have put most of their employees back on to rosters which keep crews together (e.g. five day rosters with specific weekend crews) indicates the unsatisfactory nature of many initial "one big team" type responses to the 1988 decision.

It should be understood that a "one team" concept may have validity for mines employing a limited number of people (e.g. 50) and is often the approach used successfully by smaller operators. If, however, the complexity and size of a mine dictates high capital cost equipment, interrelated key processes and a relatively large workforce (e.g. > 150) then many companies have found it necessary to develop alternative organisational approaches.

**Many independent teams**

One approach which has been taken to overcome the limitations of the "one big team" concept is to create many smaller teams, thereby trying to emulate the benefits of smaller operations. This development reflected the changes in organisations generally to flatter structures (Jaques, 1989) and greater emphasis on teams (Katzenbach and Smith, 1993). This approach involved the following concepts:

- functional leaders and teams, together with their accountabilities, are defined in terms of functional requirements;
- there are minimal changes in team membership in the short term.

This "many independent teams" model is illustrated in Fig. 2 which illustrates teams working largely independently to optimise own performance sometimes at expense of overall colliery performance.

While this approach overcame some of the defects in the "one big team" model it also raised some difficulties of its own. These deficiencies became evident when many teams began to work independently to optimise their own performance at the expense of overall colliery or organisation performance.
Many collaborative teams

An approach which tries to combine the best aspects of the "one big team" and "many independent teams" models could be described as a "many collaborative teams" model. Its key characteristics are as follows:

- functional leaders and teams identified but accountabilities defined in terms of how function contributes to overall colliery performance;
- the teams work collaboratively to optimise colliery or overall business performance;
- the colliery (or business) leader promotes teamwork between functions by own behaviour and organisational systems put in place (e.g., measures, recognition systems, decision-making processes).

A brief outline of this model is shown in Fig. 3 in which many functional leaders and teams working collaboratively to optimise colliery performance. This model is consistent with concepts such as systems thinking (Senge 1990 and Senge et al. 1994), network organisations (Miles and Snow, 1994) and ternary rather than binary modes of operation (Mant, 1997).

**IMPACTS OF HOW MINE IS ORGANISED**

Examples of the deficiencies in both the "one big team" and "many independent teams" concepts, together with how these might be overcome in a "collaborative teams" model, are given below.

**Accountability for performance**

- *One big team* - there are no clear accountabilities and individuals can be involved in too many "crisis" decisions thereby not allowing enough time to do anything else e.g., planning, communications;
• Many independent teams - setting up functional leaders and their teams has led to a clearer accountability for that function’s performance, but sometimes circumstances require cross-functional support eg a conveyor belt is buried, which is delaying the whole mine and there is a need to re-allocate resources quickly to fix the problem;

• Collaborative teams - define accountability to cover not only individual or team’s own output but also how they impact on others’ output eg does individual or team behaviour adversely affect internal suppliers or customers to the detriment of the colliery as a whole?; ensure performance measures and recognition systems are consistent with this wider accountability.

Authority to make decisions

• One big team - wide span of control can lead to poorly defined or inconsistent authorities eg front-line leader has no authority to spend resources (time, money) on a cheap improvement idea but can make production decisions with major impacts on bottom line eg continuing production when geological conditions change and a roof collapse results;

• Many independent teams - leaders can have decision making power but this can be used to the detriment of other functions; also can claim that extra line functions such as communications, safety, responding to ideas etc have them overworked;

• Collaborative teams - clarify authorities (ie the boundaries of discretion) and also give opportunities for employees at each level in the organisation to increase authority eg short term planning, spending authority etc which can reduce workload on leaders who can focus on issues where they can have greatest impact.

Effectiveness of communication

• One big team - results in key players being spread thinly and so busy that time for communication processes is low on the priority list, often relies on the grapevine;

• Many independent teams - can result in teams not sharing with or understanding what’s happening in other teams;
• **Collaborative teams** - set up processes to ensure an individual team receives all relevant information to do its role effectively, including internal supplier and customer requirements, contextual information on colliery in terms appropriate to team, feedback on performance plus discussion of ideas and concerns.

**Long term versus short term planning**

• **One big team** - where leaders are accountable for the short term operating performance as well as the long term planning, immediate challenges prevent enough consideration of longer term threats and opportunities eg the focus is on today's production at expense of roadway development or overburden removal;

• **Many independent teams** - where long term planning is off-line there can be inadequate communication and decisionmaking systems to ensure that short term decisions do not adversely affect long term viability eg those negotiating purchase of an adjacent lease delay settlement to get a slightly cheaper price but this delay adversely affects mining options in the longer term at significant cost;

• **Collaborative teams** - set up a structure to enable long term planning to be done off-line with the appropriate level of skill/experience; set up communication/problem solving processes to ensure effective interaction between planning and line functions.

**Risk management**

• **One big team** - everyone thinks they know what everyone else is doing and so little or no procedures are developed; employees can work in areas where they have limited training or experience eg on overtime, absentee replacement;

• **Many independent teams** - procedures are developed in functional areas but cross-functional interactions not defined eg face personnel were following the designated "normal" procedures but a serious incident still occurred because pre-mining conditions had changed and the procedures had not been altered at the face;

• **Collaborative teams** - need to ensure that the critical interactions between the various functional areas are defined (procedures), understood (training & communication) and carried out (define accountabilities for work, audits etc).

**Internal customer-supplier relationships**

• **One big team** - no clear process concept so no specific supplier and customer relationships;

• **Many independent teams** - teams and functions have internal suppliers and customers but needs are not understood eg purchasing officer buys timber for underground support at cheapest price, which leads to inadequate roof support; one shift produces a production record requiring next two shifts to stop producing to clean up etc;

• **Collaborative teams** - define and ensure understanding of internal supplier and customer requirements (analogous to that between partners in a contract mining operation) to ensure that decisions are taken to produce optimum overall result.

**Resource usage**

• **One big team** - no clear procedures for allocating resources and no particular ownership or accountability for equipment; no effective support functions to ensure that equipment gets to the right place on time eg crews hide equipment to ensure it is available to them when they need it;
Many independent teams - competing destructively for company resources eg when one mine in a company shut, functional leaders from the other two company mines raced in to claim the equipment first, irrespective of where it would best benefit the company;

Collaborative teams - company leaders need to make it clear that what is best for the company as a whole is the key criterion on which the leaders are going to judge outcomes; leadership behaviour and performance reviews emphasise best outcomes for colliery/company rather than function.

Sharing ideas

- One big team - no structured approach to sharing ideas because it is assumed that people will hear somehow if necessary eg staff in similar/complementary roles in the same organisation but at different locations more likely to meet each other at an external conference rather than within their own organisation;

- Many independent teams - "it will not work here because our mining conditions are different" (not invented here) eg leaders not giving crews the opportunity to learn the good operating practices of those in neighbouring panels or mines;

- Collaborative teams - need to structure opportunities for idea sharing eg internal site visits, recognition for ideas being provided to others and to those who adapt others' ideas; monthly leaders' meetings at a participant's site including a site visit; develop shared data bases on equipment performance/lessons learnt as carried out by contractors (Cutifani, 1997).

Decisionmaking processes

- One big team - decisions often taken without appropriate input from functional specialists eg a very high capacity, expensive conveyor built and installed which delayed production due to long installation time when a surge bin & smaller capacity conveyor may have been the best option;

- Many independent teams - recommendations for improvement dominated by functional outlook eg colliery personnel were asked to recommend what was the highest priority to improve mine performance: mining engineers wanted larger capacity coal cutting equipment while maintenance personnel wanted more sophisticated monitoring equipment - when production data was analysed it showed existing equipment was performing poorly under certain mining conditions and the best improvement project was to modify existing equipment which proved successful;

- Collaborative teams - where critical decisions are being addressed set up a problem solving process or task force to have independent review of options, involving internal suppliers and customers to ensure upstream/downstream impacts and potential bottlenecks covered.

Utilisation of support/services functions

- One big team - line personnel have wide span of control and there is often limited involvement of support/services functions to improve the colliery's performance;

- Many independent teams - support/services functions usually put in place but roles, authorities, accountabilities and interactions with line personnel often not clarified by colliery leader;

- Collaborative teams - if support/services functions put in place then colliery leader needs to ensure accountabilities/authorities are clear and effective interactions with line personnel are established (otherwise, why have them?) - should be similar to a partnering/contractor model (Elliott, 1997).

Measurement systems
• **One big team** - as the membership of crews could be readily changed (particularly evident after the 1988 CIT decision) there were no clear accountabilities for specific performance areas eg if there were absences from a high production area, other crews would be broken up to fill the gaps so that there was limited ownership of performance effectiveness in many areas;

• **Many independent teams** - can lead to contradictory measures eg maintenance team has priority measure “to eliminate breakdowns” which can lead to high cost maintenance plus low availability of equipment to operators (machine being repaired for too long); on the other hand, the production team is being measured primarily on short term output which can lead to delays in putting equipment in for maintenance, resulting in breakdowns and/or inefficient maintenance eg maintenance workload not spread out;

• **Collaborative teams** - identify key performance drivers for the colliery and develop functional team measures to be consistent with these ie ensure measurement systems are consistent with the overall behaviour and outcomes you want; complementary and consistent measures should clearly be spelt out as in contractor/partnering arrangements.

**Recognition systems**

• **One big team** - predominantly recognised for mine wide performance eg mine wide bonuses are essentially dominated by one component only, such as longwall production;

• **Many independent teams** - individuals or functional teams being recognised for their own performance only and not also their impact on overall site performance eg the technical expert who refuses to listen to ideas of others; the roadway development team which “cuts corners” to achieve meterage targets but creates unstable gateroads, thereby adversely impacting longwall performance;

• **Collaborative teams** - set up recognition system which not only focusses on individual’s or single team’s actual work outputs but also their impact on overall colliery performance.

**Involvement**

• **One big team** - as there are no clear accountabilities the tendency is for employees to be involved or expect involvement in inappropriate areas and miss involvement in relevant areas eg may expect to make decisions about management re-structuring, but cannot develop simple ideas to improve own workplace;

• **Many independent teams** - each team may wish to develop an independent approach to a mine wide system eg safety approach;

• **Collaborative teams** - mine leadership needs to identify “what’s negotiable and what’s not” eg safe work principles may not be negotiable, but employees will be involved in developing safe work procedures in their area based on these principles.

**OPTIMISING COLLIERY PERFORMANCE**

**Action?**

As the examples of behaviour under the “one big team” and “many independent teams” concepts indicate, sub-optimal outcomes from a colliery or overall business perspective often result. These deficiencies can often be overcome if leader behaviour and organisational systems are based on a “many collaborative teams” approach. Some of the key components of this concept are summarised in Table 1.

Assuming that the leaders of a colliery or a company containing a number of mines want the organisation to follow a collaborative model then a process for achieving this outcome could be as follows:
1. Leaders agree what personal behaviours and organisational systems they want - Table 1 could be used as a starting point and checklist for developing these ideas;

2. An assessment is conducted to determine how far the organisation is from the ideal - this could be a process of structured interviews or workshops conducted by someone independent of the organisation to facilitate openness of those providing input;

3. Leaders determine the priority behaviours and/or organisational systems to change based on a number of criteria e.g. size of impact on organisational performance, ease of changing behaviour/system, initiatives already under way etc - this step may need to have an external facilitator to ensure that the leaders do not ignore where their own behaviour may need to change;

4. Leaders determine the priority initiatives for change (i.e. the “critical initial few”) and the resources required to support them - it is essential that this step is considered carefully, otherwise initiatives may fail due to under resourcing and competing objectives e.g. the need to maintain production;

5. Leaders develop an action plan for the priority initiatives but identify other behaviours/systems which may be affected by these changes so that these may be monitored to ensure they do not cause the priority initiatives to fail;

6. Trials of the priority initiatives take place in a part of the organisation e.g. a change to the performance measurement system for the colliery leader’s own team;

7. Lessons from the trials of the priority initiatives are considered by the leaders who then modify the change, if necessary, before implementing across the whole organisation.

This process involves the leadership team in a colliery/company reviewing and agreeing any initiatives undertaken so that issues such as competing priorities can be addressed and lessons learnt from trials in one area will be transmitted throughout the whole organisation. If the process is carried out effectively it will show that “teamwork begins at the top” and that “the not invented here” syndrome is not acceptable in the organisation.

Other improvement approaches

From the description of the “many collaborative teams” model, it can be seen that this approach is predominately about what leaders do i.e. their behaviour and the organisational systems they support. It is the leaders who create the organisational environment which determines the limits to performance of individuals and teams. Therefore, this approach should be seen as complementary to other organisational improvement initiatives which aim to improve individual or team performance. For example, a colliery may have a quality programme to train operators in problem solving to improve their processes. Inevitably some solutions generated will involve cross-functional interactions. If the leaders’ behaviour and the systems they support (e.g. measures of performance) are not consistent with achievement of optimal colliery outcomes, then ultimately the quality initiative will either not reach its potential or even fail altogether. Furthermore, front line personnel will clearly see the waste that lack of cross-functional collaboration causes and will become increasingly cynical about any improvement initiative.

In summary, this “many collaborative teams” model is about leaders’ behaviour and teamwork, together with the organisational systems which support these objectives.

Unless a collaborative approach is evident from the top of the organisation, optimal performance will not be achieved. Lack of success in a particular improvement initiative, which has been successful in other organisations, may indicate poor collaboration across functions. This outcome may then become a trigger for initiating the seven step process outlined in the previous section.
As coal mines have become more complex (e.g., through the move to longwall in underground operations, larger capacities in multi-seam open cuts), organisation issues and relationships between functions have also become more challenging if optimal overall outcomes are to be achieved. Key components in addressing these challenges are the way the colliery is structured into functions, organisational systems and leaders' behaviour. An approach to achieve optimal outcomes has been described in a "many collaborative teams" model. The challenge for colliery (or company) leaders is to ensure that their own behaviour and the organisational systems they support are consistent with the desired outcomes.
REFERENCES


Modern Management Systems for Higher Margins

M Roberts¹

ABSTRACT

Maximising margins in today’s competitive international commodity markets demands superior productivity. This requires more than selecting the best equipment. It requires continually improving the efficiency of production, commercial, marketing and administrative processes under the control of solid modern management systems.

This paper introduces an integrated way of working that includes proven management systems for successfully changing work practices and attitudes, developing managers and increasing accountability. It introduces a proven methodology for systematically involving people in sustained true continuous improvement of processes for maximising productivity, safety and revenue while minimising costs.

The paper gets down to basics and shows how traditional systems entrench outdated and sub-optimal management and work practices. It describes how traditional performance measurement and reward systems cause managers to miss opportunities to control and improve basic production and service processes.

It also describes the basic methodology for using the principles correctly and provides examples of successfully implementing modern systems in mining - to achieve higher productivity.

INTRODUCTION - PROVEN SUCCESS

The improvement in underground development rates at a progressive client’s mine within just one and a half months of starting to implement the principles described in this paper - and after successfully completing only the first step in a seven step process of performance improvement is shown in Fig. 1

Seven months after starting implementation and after completing just the first three steps, the sustained improvement in productivity was over 50% - with reduced capital equipment!

Around the world, operations using the principles have reported sustained productivity improvements of up to 100% and cost savings of as much as 50% - without additional capital investment.

Improving productivity boils down to:

- identifying what to change using modern correct Measurement, Analysis and Reporting systems,
- applying a systematic methodology for improving operating and commercial processes,
- changing people’s behaviours and attitudes - to build a new culture for ongoing improvement

In many businesses, all three actions are flawed. Even managers’ attempts to change people’s work practices fail simply because their methods ignore the basics of human behaviour.

This paper will cover the first and third topics after briefly examining the fundamentals for improving productivity

¹ Catalyst for Corporate Performance Pty. Ltd., Brisbane
Before examining why many managers and executives fail to achieve lasting significant changes in work and management practices, please consider the following brief yet important note. Many managers state obvious agreement with this note yet unfortunately their systems often prevent them from complying with it.

To improve results, first improve processes!

A process is defined as a series of (usually) repeatable tasks. There is a wide variety of production, commercial, marketing and administrative processes in mining.

An obvious example is the breakage of rock in underground roadway development and production faces. In these processes, outcomes are measured in metres or tonnes together with dollars and coal quality. Coal preparation processes also have outcomes with similar measures.

Other less obvious but important examples include processes for relocating production mining faces (and their equipment) and processes for providing maintenance, clerical, marketing and commercial services. Such processes have outcomes measured in time, dollars and work quality.

A less tangible but very important example includes management processes such as planning where outcomes are measured in speed and quality of making decisions and providing information.

To improve organisational performance, the first fundamental is to think in terms of processes. Overall business margins do not improve through needless or wasteful investment in higher capacity capital, slashing indiscriminately at cost structures, wishing or simply shouting for better performance. Sustained improvement in margins requires sustained changes to the processes that produce the results. It requires process thinking.

Further, it requires an holistic approach over the whole organisation - the overall process.

Now, let's discuss why many managers fail to achieve changes in work practices.
Why the common approach to managing change fails

Managers of business need to lead people to change work practices and attitudes to improve productivity, quality, reliability - to maximise margins and minimise risk. This is a universal challenge faced by managers and executives.

The combination of behaviours and attitudes and to a lesser extent symbols defines culture. Workplace culture (including management culture) is the greatest determinant of business performance.

The common approach to changing attitudes and behaviour is to simply provide training and communication alone. Managers hope that such effort aimed at changing attitudes will then lead to changes in work practices. Fig. 2.

This concept is not correct - and wastes valuable management time and effort. Instead, the reverse is true. ie, humans change attitudes to align with behaviours.

Consider how humans develop attitudes toward the world around them. People develop attitudes based on their personal experience and in particular based on aligning their attitudes to be consistent with what they actually do - their actions. Fig. 3.

Hence, the key to developing new attitudes is to provide managers and operators with new actions - new experience - by changing what they do. By changing people’s work and associated actions. This is achieved simply by changing systems that drive behaviour (Fig. 4).
Dramatic changes in attitude will then follow. Mackay (1994) provides a good, concise summary of this in everyday terms and Brown (1954) provides solid material to further describe the social behaviour of people at work.

A system is defined as a combination of procedures, equipment, people, policies and/or values that drives people’s behaviour. It may be formally recognised and documented or informal. i.e., within an organisation, a system is anything that drives ways of doing things. To change behaviour, managers must first change the systems in which people work. Managers need to provide systems that drive the desired behaviour.

Far from being the most difficult step, this is actually the simplest step - providing there is a basic operating philosophy to guide management. And providing management uses Measurement, Analysis and Reporting systems that drive teamwork and focus on learning about outputs and the processes that produced the outputs.

Modern management systems

Changing people’s actions can be relatively simple. By far the greatest driver of human behaviour at work is the Measurement, Analysis and Reporting system. This establishes what is seen by people to be important and demonstrates the results of their efforts. It becomes the target on the wall toward which people work.

Unfortunately, as explained below, most businesses use Measurement, Analysis and Reporting systems that mislead people and drive sub-optimal and even counterproductive behaviours particularly among managers.

Fortunately Measurement, Analysis and Reporting systems can be changed unilaterally, simply and easily to drive desired behaviours.

At the other extreme, changes to pay systems usually need considerable discussion and usually negotiation. Such changes can sometimes be made quickly. More often they need to be made in stages. Regardless, even though remuneration systems are generally the second most powerful drivers of behaviour few remuneration systems drive behaviours aligned with the business’ goals. Indeed, many pay systems drive behaviours that are counterproductive.

Between these two extremes lies organisation structure. While managers have the prerogative to change structures unilaterally, development and implementation of the optimal structure often requires consultation which needs investment of management time and effort.

One North American client immediately lifted core process availability from 85% to 89%, simply by changing its organisation structure away from the traditional departmental structure to one based on processes. The client quickly started reaping additional major ongoing benefits in higher morale and pride as operators, mechanics and managers worked together. Additional benefits are coming from increased ore recoveries estimated to be worth $2,000,000 plus lower costs.

Ten key systems for driving behaviour:

- performance measurement, analysis and reporting
- organisation structure;
- management processes (planning, communicating);
- personal development and performance feedback;
• remuneration;
• accounting, budgeting and forecasting;
• core and service work processes and standards;
• general communication systems and processes;
• systematic ongoing involvement and recognition of people; and
• methodology for improving processes.

Other systems for driving behaviour include:

• safety management systems;
• planning and design;
• promotion; and
• selection and preparation (recruiting).

Additional systems are listed in Roberts (1995).

It is amazingly easy to change people's attitudes once the fundamentals are introduced correctly and behaviours are driven on-the-job with consistent, integrated systems.

As a mine manager, general manager and now as an external resource, the author has initiated and witnessed simple changes in management systems that have led to dramatic changes in management attitudes and significant increases in accountability.

The importance of correctly designing management and work systems to ensure effective culture change is detailed by Roberts (1995). To successfully manage change, do not focus on attitudes. Focus on behaviours - and change the systems.

The paper will next examine broadly the most powerful system for driving behaviour and building attitudes.

**Accurately identifying what to change**

By far the most powerful driver of behaviour and attitudes is the performance Measurement, Analysis and Reporting system. It determines what becomes people's targets and is usually far more powerful than pay systems.

Broadly, the two main purposes of measuring and analysing processes and businesses are:

- improving processes - identifying accurately what to change and assessing the effectiveness of changes;
- understanding performance levels relative to targets (whether imposed by markets or internal budgets)

In this section the focus will be on improving processes.

**Variation is a law of nature**

There is variation in all dimensions, objects and tasks. No two people are identical. No two cars off the same assembly line will have exactly the same panel gaps even though from the same process. No individual or team will install roof bolts in identical times on every occasion. No two loads of ore will be trucked from face to crusher in exactly the same time. No two loads of coal will have exactly the same ash content when fed into the preparation plant. Variation occurs in everything.
Hence, the outputs from a process will vary even if the process remains unchanged. Such variation is known as natural variation. To react to each such variation by allocating resources or changing the process in reaction to each output point is obviously foolish and counterproductive. Yet this is exactly the behaviour that traditional measurement systems encourage.

**Society's Poor Understanding of Variation**

Traditional Measurement, Analysis and Reporting systems lack understanding of variation. In traditional systems, work processes and their performance are assessed merely by comparing the latest output data point to a target/budget, or to the previous data point or to the average output. This reinforces poor understanding of variation - and leads to poor understanding of the process itself.

Decisions are commonly made as a reaction to the location of only the last data point.

Fig. 5 shows a headline and graph from a supposedly reputable finance and business journal. The graph shows the "last data point" evidence used to justify an article implying the economy was slumping - at a time when the economy was clearly in full recovery. The incorrect analysis and conclusion reflected in the erroneous headline were based on comparing the last two data points - "0" and "N"! Yet the reader will quickly see the error by simply examining the graph visually.

![Housing market stumbles: Net sales of new homes, monthly, seas. adj. 000s](image)

**FIG. 5 Poor understanding of variation**

Traditional managerial and business analysis leads to many such errors. A quick glance at the graph shows there are two different “processes” for selling houses. The first portion of the run chart shows results during an economic recession while the second portion shows a dramatic and sustained recovery in sales. This shows a change in the process!

The two dashed lines added by the author highlight the change. Note that within both periods there is monthly variation due to natural variation.

Similarly, output (such as production, safety, quality and cost) will vary. Managers must not make incorrect conclusions from natural variation. After all, who would expect to sell exactly the same number of houses every month?

The headline’s poor understanding of variation is representative of management systems guiding managers to operate businesses and manage processes by drawing conclusions based merely on the last data point output.
It is not just the use of graphs that is important. It is the graphical presentation and the analysis of data using an accurate understanding of variation.

Fig. 6 shows a change to an actual mining process. The change was missed by site operations managers and corporate executives using traditional analysis systems based on relatively few data points. This obscured the real improvement and prevented the changes from being identified and locked into place. Instead they were lost from the business!

**FIG. 6 Identification of a process improvement**

Simply graphing their existing data and merely eyeballing the run chart of process output data using an understanding of simple rules for analysing variation makes the improvement obvious.

How many such improvements in industry are not detected and therefore not locked into the process and thus lost? How many drops in performance are not detected and therefore not identified and locked out of the process and thus continue to suppress performance?

Many!

**Budgets are not Process Improvement Tools**

Traditional measurement systems are characterised by reliance on comparison with budget and the use of tables. Such systems are valid only for assessing performance relative to targets.

While they can usually identify that something needs to be done they cannot identify what to change in a process.

Used alone as is the case by many managers, the use of budgets as process improvement tools leads not to focusing on root causes but to suboptimal management decisions and behaviours, finger pointing and creative excuses. This wastes resources and undermines management’s authority and reduces discipline!

Budgets are important planning tools and scoreboards. They are not process improvement tools.
Contrast this with the use of modern, accurate analysis provided by run charts, Pareto charts and cycle time charts in the hands of managers who understand the simple yet powerful principles for understanding variation to make accurate decisions!

Run charts, Pareto charts and cycle time charts are just three of many simple yet solid graphical tools for unearthing precious knowledge immediately - and for accurately forecasting and improving process behaviour.

**Run charts**

Leading businesses use run charts (Fig. 6) to enable managers to accurately and quickly identify significant points indicating a process has changed — and to assess the impact of proactive, planned changes. This involves plotting and analysing process output data points in accordance with a handful of simple basic rules.

The human eyes are the world’s most powerful and accurate statistical analysis tools. Eyeballing in accordance with basic statistically sound and simple rules is recognised as statistically sound and is the only way to accurately and simply analyse data to obtain real knowledge of processes.

Once a change has been identified it is then possible to drill down to the causes of the change. If the change is beneficial, it is captured by locking it into the process through standardisation. If the change is detrimental, it is locked out of the process. And so on for each change so that true continuous improvement occurs and is locked in.

**Pareto charts**

When proactively making changes to production and commercial processes it is always best to first make accurate analysis before determining what to change.

Pareto charts (eg. Fig. 7) provide the ideal analysis tool and can be coupled to existing data bases with minimal effort.

As stated by a client’s manager after seeing the use of Pareto charts applied to analysis of production delays, “from now on we won’t be picking the jobs we want to do, we’ll be doing the jobs that need to be done”!

**Cycle time charts**

Fig. 8 shows actual improvement of cycle times in development of an underground coal mine’s roadways. At 20 days the first pillar development cycle was lengthy. Successive improvements reduced cycle time dramatically by standardising work methods. Once the process was stabilised and in tight control it was then possible to attack the sub-processes and productivity components. Modern analysis methods were being used at the mine to direct and monitor progress - to lead.

Contrast this with a commonly used measure of drivage rates in mines - metres/shift. This focuses on maximising individual shift performance and leads to sub-optimisation. Instead, in reality, drivage depends on the performance of all shifts and departments interacting around the clock.

Additional benefits of using cycle time charts include leading people to improve planning and to perform as many activities as possible simultaneously - in parallel. Cycle time charts move people’s focus from just current tasks to the whole cycle. They communicate process improvement graphically and quickly. This builds work satisfaction and pride of achievement. Cycle time charts also provide direct indication of waste - the longer the cycle the greater the waste.

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**COAL98 Conference Wollongong 18 - 20 February 1998**
Available Time Pareto Analysis

Organisational Delays Pareto Analysis

(Data has been truncated to maintain confidentiality and ensure brevity)

FIG. 7 Pareto chart for systematic improvement
FRAMEWORK FOR EFFECTIVE ANALYSIS

Many organisations are wasting valuable management time in collecting unnecessary data and in neglecting analysis. Such businesses have lots of data yet lack knowledge and understanding of processes!

It is simple, easy and low in cost to build and use a suitable database or to modify most existing databases to provide solid analysis. The main prerequisite is to establish an overall approach to minimise the volume of data and maximise knowledge.

Remember, it's not measurement alone that matters. Such limited thinking drives counterproductive behaviour. Instead, it's measurement, sound analysis and reporting that drives correct behaviours. In process monitoring and business performance improvement the key is analysis.

Consider two facts:

- without data and sound analysis, you're just another opinion!; and

- Measurement, Analysis and Reporting systems are by far the most powerful drivers of behaviour and culture


The importance of controlling variation to understand the process and identify opportunities leads to putting the business and process knowledge to use in a disciplined, efficient and effective method for improving performance.

PROVEN METHODOLOGY FOR IMPROVING PRODUCTIVITY

It is easy to tickle up performance with immediate changes. To sustain and then continually improve performance though managers need to first get tight control of processes.
The methodology for improving performance is based on these broad steps:

- define the process
- modify management systems to support the process and drive desired behaviours
- get control of the process, then,
- raise the level of performance by attacking the productivity components:
  - hours of production time
  - production rates per hour
  - resource allocation and utilisation
- Then continually improve the process

This broad outline forms the basis of the proven seven-step methodology for improving processes which unfortunately beyond the scope of this paper.

The methodology is based on the fact that wastage of resources increases (and decreases) as variation increases (and decreases). This fundamental is now proven in commercial, production, marketing and administrative processes worldwide. It has been explained in theory by Taguchi in Deming (1986). The relationship between variation and wastage of resources is detailed in Roberts (1995).

It is particularly important in mining and agricultural processes which obviously deal with more highly variable input than do processes in manufacturing and service sectors.

Reducing variation provides two immediate benefits:

1. increased managerial control of processes; and
2. reduced wastage of productive resources for higher performance.

Return to Fig. 1 which shows the impact of reducing natural variation by standardising the mining processes. The reason is simple. As variation decreases, the wastage of resources such as labour, ideas, time, money, capital and materials decreases! Processes with lower variation are also easier to manage and to improve. Planning is more effective and work easier - and thus safer and more productive!

Managers and their people have greater control over their work processes.

Mining examples highlighting the increase of wastage with variation are provided in Roberts (1995).

(There is also an internationally known methodology that applies at the micro-level of process improvement and enables systematic assessment, implementation and standardisation of new ideas and suggestions. This Plan-Do-Learn-Act loop is tied to the overall seven-step performance improvement methodology.)

Remember though that the control of variation and processes begins with accurate measurement and analysis of variation!

The benefits in accountability, discipline, safety, metreage, tonnage, throughput, yield, recovery, product purity/quality, total cost and unit cost in mines and prep plants go straight to the bottom line.

Strategic Aspects of Putting it to Work - Solid Overall Plan

To ensure the development of all systems drives consistent behaviour, managers need to build and implement an overall plan for changing systems An example is provided in Fig. 9. This requires and provides a strategic, holistic approach over the whole business.
This also has benefits in showing each manager his part and responsibilities in changing systems and displaying clearly the dependence of other managers on his/her effort. It is highly effective in communicating change and instilling confidence in people at all levels that management knows where it is going. Significantly, it is extremely useful in reducing uncertainty - and after all people do not dislike change they dislike uncertainty.

Obviously the most powerful driver of behaviour must be one of the first systems to build - the performance Measurement, Analysis and Reporting system.

**PRACTICAL IMPLEMENTATION IN MINING**

Ideally, implementation starts at “the top”. Concepts are introduced and systems are built at senior corporate management level and then progressively down through the organisation structure to the people in the core processes of each production, commercial and marketing department.

Where there is need for immediate performance improvements it may be necessary in practice to first build and implement basic systems at the management level which makes daily operational decisions. In such instances implementation should then proceed quickly up and down the organisation. Fig. 10
FIG. 10 Implementation sequence for rapid improvements

Where senior management is not committed to the way of working, this second implementation sequence has higher risk than with starting at the top.

Thus, the starting point depends on balancing the client management’s immediate and long-term needs, skills and commitment.

Never commence implementation with the operators. Unfortunately this is where many attempts to implement the principles start - and therefore fail. While it’s easy to start at the face and “tickle-up” performance, the improvements will not be sustained.

For higher performance to be sustained, appropriate supporting management systems must be in place.

Managers who try to start implementation at the operator level often do so because they mistakenly think the concepts are merely a human resources initiative and a bag of process improvement tools for operators. Such managers fail to fully understand the principles. Implementation traps are detailed in Roberts (1995).

In improving businesses there is rarely a need for extensive training or lengthy, expensive strategic analysis. Most organisations already contain the necessary talent. It simply needs to be harnessed and united in a common way of working under appropriate and consistent modern management systems.

During execution of the plan it is important that in addition to understanding and using the overall steps, managers and operators are left with tools for continuing to improve productivity themselves as part of their ongoing daily work.

Above all, this is a strategic initiative. It is most leveraging at corporate and senior site management levels.

**SUMMARY AND CONCLUSIONS**

The proven principles discussed in this paper form a complete way of working, a template for best practice management systems, a methodology for improving processes and an integrated group of productivity improvement tools for continually improving processes and results.

This paper has focused on the Measurement, Analysis and Reporting systems and on the importance of systems in general for driving behaviours and attitudes.
Remember, to improve results, first improve processes.

Secondly, people using traditional measurement systems fail to recognise natural variation and cannot correctly analyse business processes. This prevents full understanding of work processes. Core problems are often not identified and thus the biggest opportunities for improvement missed. Instead of fixing core problems, people react to the last data point and chase symptoms. This wastes resources and drives sub-optimal and even counterproductive behaviour.

A proven alternative exists. The use of simple yet solid statistical tools provides real knowledge about work processes and waste. When people are armed with real knowledge they are effective in identifying and killing waste using a proven methodology for maximising productivity.

Do current measurement, analysis and reporting systems in the reader's business encourage real understanding of work processes? Or, do systems focus managers on allocating resources in reaction to the last data point?

Thirdly, in changing work practices, don't focus on attitudes. Instead, focus on behaviours. Build systems that drive the desired behaviours - which will then develop the desired attitudes.

The Measurement, Analysis and Reporting system is also by far the most powerful driver of behaviour aligned with any organisation's business goals.

Pause and consider the formal and informal systems that currently drive people's way of doing things. What do current systems tell people? What behaviours do current systems foster in managers and their people?

Adopting this consistent way of working and using its proven methodology for process improvement will improve workplace cultures and optimise business processes to maximise productivity, margins and return on investment.

World best practice systems - isn't that what is wanted?

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How to use pre-employment medical examinations and comply with Anti-Discrimination Legislation

D Scholz

INTRODUCTION

The law including legislation such as the Occupational Health & Safety Act 1983 imposes stringent obligations on employers to ensure the health and safety of their employees. The use of pre-employment medical examinations is one tool that employers can use to assess the suitability of a job applicant for a particular position and protect themselves from prosecution or claims for compensation or damages.

At the same time, however, legislation such as The Disability Discrimination Act 1992 (Commonwealth) ("DDA") and The Anti-Discrimination Act 1977 (New South Wales) ("ADA") affords protection to individuals against discrimination.

This legislation is intended to ensure, as far as is practicable, that people with disabilities are treated equally to other members of the community and are brought into the mainstream of our society as far as possible. It makes it unlawful for employers to discriminate against prospective employees because of their disability.

In the context of pre-employment medicals, the protection from discrimination is designed to ensure that job applicants with disabilities have as much opportunity to obtain employment as able bodied applicants.

Employers must ensure that they use pre-employment medical examinations in a way that is both relevant for their workplace and complies with the requirements of Anti-Discrimination legislation.

The definition of discrimination in the Federal and State Acts is virtually identical except for variations in the type and extent of various exceptions.

Complainants are free to choose between State and Federal jurisdiction in situations which are covered by both. While the Equal Opportunity Tribunal (State) cannot order damages in excess of $40,000.00 there is no limit on the amount of damages the Human Rights and Equal Opportunity Commission (Federal) can award.

RANGE OF DISABILITIES COVERED BY THE ANTI-DISCRIMINATION LEGISLATION

Section 4 of the DDA and Section 4 of the ADA define disability as:

- Total or partial loss of a person's bodily or mental functions (eg being paraplegic, having epilepsy);
- Total or partial loss of a body part (eg. by amputation);
- The presence of organisms causing disease or illness in the body (eg. hepatitis, HIV positive);
- Malfunction, malformation, or disfigurement of a part of a person's body (eg. hearing loss, loss of sight);
- A disorder or malfunction resulting in a person learning differently from a person who does not have the disorder or malfunction (eg dyslexia), (DDA);
- A disorder, illness or disease that affects a person's thought process, perception of reality, emotions or judgments

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or that results in disturbed behaviour (e.g. schizophrenia, psychiatric conditions).

The ADA also forbids discrimination against a job applicant because of the existence of disability in a friend, relative or associate of the applicant.

Must the disability be present? Under the legislation, disability includes past, future and presumed disability. Section 49A of the DDA defines "disability" as including disability which:

- previously existed but no longer exists;
- may exist in the future; or
- is imputed to a person.

The ADA forbids discrimination on the basis of a disability which a person has;

- is thought to have;
- is thought to have had or;
- will have in the future
- whether or not the person in fact has; had or will have the disability in the future.

Example:

*Barry v State of Victoria (1994) EOC 92/598*

The complainant was a 27 year old man who had been diagnosed with Hodgkins Disease. He applied for a job as a prison officer with the Department of Corrective Services. The medical officer who conducted the pre-employment medical examination stated that he was physically fit to perform the duties required of a prison officer but because he had recently suffered from cancer, he was not eligible for employment until he had been free from recurrence for 2 years.

The Equal Opportunity Board found that Mr Barry had been unlawfully discriminated against on the basis of his past illness. The decision not to employ him was not based on medical grounds as such but on the basis of the future risk to his prospective employer that Mr Barry's past medical condition would recur.

**FORMS OF DISCRIMINATION - DIRECT AND INDIRECT**

Under Section 49B of the ADA and Sections 5 & 6 of the DDA discrimination can be both direct and indirect.

**Direct discrimination**

This refers to discrimination on the grounds of disability taken to have occurred because of the aggrieved persons disability. For instance an employer refuses employment to an applicant simply because he has a disability, for example, hearing impairment or hypertension.

**Indirect discrimination**

This will occur if the discriminator requires the aggrieved person to comply with a requirement or condition:-

with which a substantially higher proportion of persons without the disability comply or are able to comply but

which is not reasonable having regard to the circumstances of the case and

with which the aggrieved person does not or is not able to comply.

Indirect discrimination will occur, for example, where a company has a blanket policy governing employment which people who fall into the group defined as having a disability cannot satisfy.
DISABILITY DISCRIMINATION IN THE WORKPLACE

Under Section 15 DDA and Section 49D to 49K of the ADA, it is unlawful for an employer to discriminate against a person on the grounds of that person’s disability or disability of that person’s associates in relation to:

- The arrangements made for the purpose of determining who should be offered employment.
- Determining who should be offered employment.
- Terms on which employment is offered.
- Denying the employee access or limiting the employee’s access to opportunities for promotion, transfer or training or to any other benefits associated with employment.
- Terminating employment.
- Subjecting the employee to any other detriment.

EXCEPTIONS TO THE DISABILITY DISCRIMINATION PROVISIONS

There are three major exceptions to the disability discrimination provisions namely;

- inability to perform the inherent requirements of the job;
- unjustifiable hardship;

1. if it is necessary for the employer to "discriminate" in order to comply with any other act e.g. to ensure compliance with the Occupational Health and Safety Act (1983) New South Wales. (Section 15(4) of the DDA and Sections 49D(4) [which replaces the former Section 49I(2)] and 54 of the ADA.).

- Inherent requirements of the job

1. If an individual is not able to perform the inherent requirements of the job because of their disability, it is not a breach of the legislation to refuse to employ that person. Care should be taken to note that the inherent requirements of a job are those which are necessary for the goals of the job to be achieved. Such inherent requirements should not be confused with the manner in which a job, function or task is to be carried out.

2. It is not appropriate for an employer to apply a blanket exclusion to employment for a particular type of disability. An exclusion must specifically relate to the requirements of the particular job for which the person applies.

What are Inherent Requirements?

Inherent requirements are the essential duties and responsibilities of a job. Any duties which are not essential should not be taken into account when considering the suitability for the job of an applicant with or without a disability.

Why should the emphasis be on inherent requirements and no other requirement?

The reason is that because non-essential duties can be removed from the job description; can be performed by someone else or can be altered to suit the person's disabilities.
Necessary adjustments, services and facilities

Employers must be able to demonstrate that they have provided any services or facilities which are needed by persons with a disability to carry out the essential duties of a job unless they can prove that to provide such services or facilities or adjustments would cause unjustifiable hardship. For example they must:

- Make facilities which are already existing and used by employees readily accessible to individuals with disabilities and also useable by them;
- Purchase equipment or devices;
- Modify equipment or devices;
- Modify training materials or policies; and even
- Reorganise the job.

Example:

*Bugden v State Rail Authority (1991) EOC 92/360*

Mr Bugden was a carriage trimmer in the employ of State Rail Authority. He requested transfer to a running depot, a more lucrative area of employment than that in which he was employed, on three separate occasions. His application had been refused on the grounds that he was colour blind in accordance with State Rail policy that people working at running depots have perfect vision.

State Rail argued that it was not possible to allow Mr Bugden to be employed at a running depot because it was a necessary component of the job that he be able to give and take colour signals.

The Tribunal found that, in some cases trains were immobilised by flags of certain colours but that it was not the colour but the fact of fixing a flag which denoted that a train was immobilised. Therefore the Tribunal found that it was not an inherent requirement of the job to be able to read a green/blue signal.

Unjustifiable hardship

In determining what constitutes unjustifiable hardship, all the relevant circumstances of each case are to be taken into account including:

- The nature of the benefit or detriment likely to accrue or be suffered by any persons concerned.
- The effect of the disability of the person concerned.
- The financial circumstances and the estimated amount of expenditure required to be made by the employer.
- The only way to determine whether an action or adjustment needed to allow the employee to carry out the essential duties of the job would cause an employer unjustifiable hardship to provided is to consider:
  - The type and range of adjustment, change or additional services or facilities which are required by the person with the disability.
  - The cost which the employer would need to expend to make the adjustment.
  - The financial position of the person claiming unjustifiable hardship.
  - Whether the modifications/adjustments etc also could be used for the benefit of other employees or clients.
• Whether any additional resources are required.
• The likely benefits and disadvantages to the organisation.
• The likely financial cost to the organisation.

Compliance with other legislation

Care must be taken with this defence. The occupational health and safety argument can only be used if the employer can demonstrate:

• A real objective risk to the health and safety of employees.
• That it is not open to the employer to take any steps which can be reasonably taken to eliminate such risks.

Most employers who have argued these before the Equal Opportunity Tribunal have failed.

OHS Requirements must be specific

Again, any occupational health and safety requirements in the workplace must be stipulated on an individual basis and not on the basis of "blanket" exclusions. Employers must be able to demonstrate objectively and by means of specific evidence, the capacity of the individual, at the time of testing, to carry out the essential requirements of a position safely.

For example, when an applicant suffers from "epilepsy" the applicant must not be ruled out on this basis as being one of class of persons who has epilepsy and therefore not fit to operate machinery. The employer must look at the individual specific medical history, past experience, work history and the specific job requirements. Relevant factors will include:

• Type of job.
• Degree of control of seizure.
• Type of seizure.
• Whether the applicant has any warnings that seizures will come ("aura"),
• Medication taken.
• Reliability in taking medication.
• Side effects of medication.

The view of the relevant Tribunals is that some epileptics may not pose a threat to workplace safety if they experience sufficient warning signs of an attack or have their condition sufficiently under control.

Example:

_Hurley v The Electricity Commission of New South Wales (1994) EOC 92/624_

The complainant, Mr Hurley, suffered from mild to moderate hypertension. Following the respondent's refusal to employ him in the position of cleaner/labourer, Mr Hurley wrote to the Anti-Discrimination Board. A report from the respondent's medical officer referred to Mr Hurley's hypertension stating that it was not controlled at one of the examinations and that Mr Hurley was therefore considered to be unfit for a job where moderate to severe physical effort could be detrimental to his health.

The Tribunal found that Mr Hurley's hypertension did not provide the employer with sufficient reason to assume that he
would be unable to carry out the full duties of the position without a serious risk to his health. It rejected the submission put by the respondent that because Mr Hurley's hypertension was uncontrolled he would have been unable to carry out the full duties required without unreasonable risk to his health.

The Tribunal was not satisfied that the respondent had been able to identify any grounds on which it formed this view and even if the respondent had established such grounds, it would not have been reasonable to rely on them without taking into account the complainant's individual circumstances, especially his medical, personal and work history.

It was also noted by the Tribunal that no routine checks of hypertension were conducted in persons currently employed by the respondent. Given this, the Tribunal concluded that "the respondent may have overstated the degree of risk posed by hypertensive cleaners/labourers".

**Bugden v State Rail Authority**

In this case State Rail also argued that their actions were justified in order to comply with the Section 15(1) of the Occupational Health and Safety Act which states: "Every employer shall ensure the health, safety and welfare of all his employees." This argument failed. The Tribunal found that although the complainant did have a colour vision deficiency which prevented him from complying with State Rail's policy, the policy was not reasonable in that it was not necessary for trimmers to be able to give and take colour signals in order to work safely at a running depot. Various signals were used to denote when it was safe to cross and in no instance was it necessary to be able to take a colour signal in order to cross safely.

**PRACTICAL IMPLICATIONS FOR EMPLOYERS**

**What do these Restrictions mean to Employers?**

The thrust of the Legislation is that it is the responsibility of an employer to select the best person for the job and in so doing not to exclude an applicant just because they have a disability.

Before an employer uses the results of a pre-employment medical examination to exclude a prospective employee from employment it must have evaluated:

- what are the essential requirements of the particular job
- whether any of those requirements are unable to be performed safely by someone with a particular disability and if so;
- whether anything can be reasonably done to modify the tasks or the manner in which they are conducted to suit the applicant's disability.

The examination must be used only to elicit information which is relevant to the person's capacity to perform the essential functions of the job and the employer's compliance with legislation such as the Occupational Health and Safety Act.

**When should pre-employment medical tests be used?**

If the job requires some particular physical or psychological capacity such tests should be an intrinsic part of the selection process. However, they are only allowable if the attributes tested are attributes or characteristics that are reasonable in all the circumstances.

Therefore, employers must take extreme care to ensure that pre-employment tests are linked directly to the particular duties of the job and do not relate to other factors. A company policy which prevents hiring an applicant who has a history of back problems regardless of the duties of a position will constitute unlawful discrimination.

Example:
Mr McQuillen applied for a position of grounds person with the University of Melbourne. He attended two interviews and was offered the position subject to a satisfactory medical examination. It was not explained prior to this that the job necessitated any particular level of physical capacity.

Mr McQuillen was found to have a longstanding back problem and he was refused employment on this basis.

The Equal Opportunity board found that the job description was misleading but even though the employer had been not been sufficiently clear in expressing the need for the applicant to be able to perform physical labour it accepted the employer's evidence that this was in fact a requirement of the position. It accepted that the complainant's inability to perform heavy manual labour meant that he could not reasonably perform the job or was at risk of injuring himself or exacerbating his existing injury. Hence the employer's actions were not unlawful.

Holdaway v Qantas (1992) EOC 92/295: 92/430

The complainant was a flight attendant whose employment was terminated after he was diagnosed as an insulin dependent diabetic. The claim by Qantas that the complainant was unable to carry out the work required of him because of this condition was rejected on the basis that Qantas had not made any inquiry as to whether the complainant was able to perform the work.

Qantas did not call evidence from the examining doctors and it was therefore not clear whether they held the view that the use of drugs to control diabetes made the complainant incapable of carrying out the duties of a flight attendant or whether they had merely adopted the inflexible requirements of Qantas' policy and applied it to the complainant.

What an employer must do to develop a non-discriminatory medical test

- Avoid or remove any blanket employment policy relating to disabilities unless they can be justified as reasonable or arise from specific legislative requirements.
- Analyse what tasks the job entails carefully and thoroughly and classify these into:
  1. Essential requirements; and
  2. Non-essential requirements.
- Identify accurately the necessary skills and physical attributes for the job in so far as they relate to the essential versus non-essential duties - make sure there is a direct link.
- Identify the type and level of physical attributes required to perform the essential duties of the job versus the non essential duties of the job.
- Carefully investigate whether there are any other ways the job can be designed or performed so that people who do not have the physical attributes required for the essential duties of the job, can perform these duties.
- Identify the types of services or facilities that could be used to assist people with disabilities to carry out the essential requirements of the job and ensure that these are provided unless they cause the employer unjustifiable hardship.
- Identify the medical tests which are relevant and appropriate for assessing the required physical attributes.
- Constantly review and reassess the job requirements so as to take account of any changes in the way the job is to be carried out since such changes may affect the type of skill and physical attributes required for the job.

It is prudent to have in place procedures for ongoing medical testing of employees in order that it may be established that:

- They continue to meet the requirements of the position and
They are not at risk of injury.

Example:


A police force instruction required potential applicants to have vision in both eyes. Consequently, a one-eyed applicant was told he was not suitable for the position.

The Police Commissioner relied on the defence that the complainant could not perform the duties required. The Tribunal found that this defence was based on a generalised assumption about losses of vision and was therefore not reasonable. It held that the respondent had not considered the application on its merits but on the basis of general assumptions as to Mr Clinch's ability to do the job. The Tribunal did however, note that it could not substitute its opinion as to the employers work requirements for that of the employer and the employer was asked to review the application.

The complainant completed a fresh application and was taken through each of the steps involved in considering his application, even though they informed the Tribunal that under normal circumstances the application would have been rejected in the early stages. Once again, Mr Clinch was unsuccessful in his application and once again, he complained alleging that he had been discriminated against on the basis of his impairment in two respects, that he only had sight in one eye and also that he suffered from a colour vision deficiency.

The members of the police selection panel stated in their evidence that the reason for their failure to appoint Mr Clinch related to such matters as his educational qualifications, his traffic convictions and his lack of commitment to the position. i.e. on aspects in addition to his visual impairment as well as the high standard of many of the other applicants.

The respondent also led evidence as to the relevance of its requirements for vision in both eyes and no colour deficiency. The evidence included evidence of that distinguishing colour could be critical to determining the guilt or innocence of an accused person; evidence of the effect of loss of one eye on ability to judge distance and that colour vision could be significant to police officers both in carrying out their active duties and in subsequently giving evidence in court.

This time, the Tribunal dismissed the employee's application holding that proper consideration had been given by the employer to the question of whether the complainant could perform the required tasks.

**MEDICAL RECORDS**

The Human Rights and Equal Opportunity Act makes it unlawful to discriminate on the grounds of medical records (Section 31(b)).

In practical terms this means that if a worker is currently healthy but has a medical history which suggests that he has had to take time off work for a particular medical condition it is unlawful to refuse him a position because of his medical history unless the medical condition is directly relevant to the job.

For example, epilepsy may preclude an applicant from becoming an air pilot but not preclude him/her from performing the position of a clerk.

**HIV/AIDS TESTING**

HIV and Aids are listed as notifiable diseases under schedules of the New South Wales Diseases Public Health Act 1991. This requires that, where there are found to be present by a medical practitioner, the medical practitioner must notify the Director - General of Health.

However, people who fall into the AIDS/HIV category must have their identity protected and applicants are not under any duty to disclose the HIV status to prospective employers. Nor are there grounds for routine testing where the HIV status of the applicant is not directly relevant to the job. Direct relevance only occurs in a limited number of instances such as where the inherent job requirements involve procedures which require skin penetration such as acupuncture, podiatry or ear piercing.
If employers do opt for routine testing without the required grounds, they are liable to a claim under Anti-Discrimination legislation as well as actions claiming assault and battery if testing is done without the consent of the job applicant.

CONCLUSION

In the context of positions which could expose employees to injury, a pre-employment medical assessment plays a crucial part in the employer discharging its obligations under legislation and at common law to provide a safe workplace.

Employers can make a job offer conditional on the applicant passing a pre-employment medical examination as long as care is taken that any conditions which are taken to be "disqualifying conditions" relate directly to the essential requirements of the job and cannot be overcome by reasonable modifications.

The assessment must be subjectively based on the particular job and the particular applicant. Any risk factors relating to the employment of a disabled person must be:

- Real;
- Specific; and
- Current.

Incapable of reduction through reasonable adjustments to the workplace or the way in which the job is performed.

Employers can use the occupational health and safety defence if they can demonstrate that there is a real risk attached to the applicant's condition and not simply a chance which is merely remote.
Contemporary Developments in Training

N Parish

ABSTRACT

This paper will examine some of the significant developments in the training field in Australia in the past decade and the application of these changes to the mining industry generally, and the coal sector specifically. Three major aspects will be examined:

1. **Competency** - the use of Industry Competency Standards and how these can be used for achieving enterprise performance requirements

2. **Assessment** - making judgements about an individual's competence, and using a rigorous assessment system as the basis for the "authorisation" system at a mine

3. **Evaluation** - ways to evaluate the return on investment in training, and convincing CEO's/shareholders that your training dollar is being spent wisely.

Before examining these issues, it is important to understand the drivers of change in the training systems in Australia, and the functions that Industry Training Advisory Bodies (ITABs) perform.

Over the past decade considerable effort has gone into developing a new basis for the Vocational Education and Training (VET) system in Australia. This change has occurred across the whole range of Australian industries, and has involved the development of what is called the Competency-Based Training (CBT) system.

There have been two major drivers of change for this past decade of reform of the VET sector of education. Firstly there has been the insistence of enterprise employers and their industry representatives that the training system provides people with skills relevant to employment. Employers have expressed continuing disappointment for seemingly decades about the irrelevance of a lot of the skills learned through "off-the-job" training. Secondly, employees and their union representatives saw the need for increased skills, greater portability, and in a lot of cases, career progression and increased rates of pay linked to increased skills. Employers, unions and governments have supported and financed the reforms, not only for their own narrow reasons, but because a modern and relevant VET system was seen as a critical element in micro-economic reform.

One element of the VET reform process, has been the formation of Industry Training Advisory Bodies (ITABs), which are tri-partite (employer, union, government) or bi-partite (employer, union) organisations formed to provide policy advice to government and industry parties. There are approximately 20 ITABs representing the major industry sectors in the Australian economy, with 2 "layers". National ITABs, responsible to the Australian National Training Authority (ANTA) and the Federal Government, are primarily responsible for the development of the Industry Competency Standards required by their industry. State/Territory ITABs are responsible to their State/Territory Training Authority (in NSW this is the Department of Education and Training), and are primarily responsible for liaising with industry stakeholders and training providers (particularly TAFE).

NSW Mining ITAB represents four industry sectors (black coal, metalliferous, quarrying and drilling) and has a bi-partite board comprising employer and employee representatives (not necessarily in equal numbers). The Mining ITAB is a "not-for-profit" company with two employees - an Executive Officer and an Office Manager. Policy is determined by the ITAB

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1 Executive Officer, NSW Mining ITAB
Board of Directors. The function and major activities of the NSW Mining ITAB are summarised in the Mission statement in its "Strategic Plan 1997-99":-

“To provide to the industry and key stakeholders effective and timely training-related services by:-

• producing an Industry Training Plan;
• transferring information;
• fostering industry networks;
• influencing Government training policies/priorities/resource allocations;
• promoting Best Practice quality-based training systems to continually improve industry training standards.”

Unlike some other industries who have a centralist view of how training should be developed/delivered and who rely to a large extent on funding and delivery of their training by governments, the mining industry in general, and the coal sector in particular, is unique in two important aspects:-

1. there is a large commitment to training, with the mining industry spending the most per employee on training of any Australian industry;
2. most of this training is carried out “at-job”, either using mine-site trainers or trainers/consultants delivering on site.

These two aspects make the role of NSW Mining ITAB different to NSW ITABs covering other industries. The major difference is that the mining industry sees that the role of the ITAB is to ensure that the primacy of the enterprise in training in the industry is maintained and enhanced. In the following discussion on the three aspects being examined in this paper (competency, evaluation and evaluation) it is important to remember this difference. Other industries are evolving different versions of the CBT system to suit the needs of their industry. The coal industry’s system is based on implementation at the mine-site level and integration with existing mine training systems.

COMPETENCY

Competencies

The major distinguishing feature of the Competency-Based Training system is that it is based on outcomes. That is, what a person can actually do in a work environment. As the name suggests, the basic foundation of this system are Competencies.

Competencies which are developed and recognised by an industry are referred to as:-

• Industry Competency Standards; or
• National Competency Standards; or
• Industry Competencies
Competencies can also be developed by individual organisations and are referred to as:

- *Enterprise Competencies;* or
- *XYZ Competency Standards (eg. McDonalds Competency Standards)*

Competency Standards are defined as: *The specification of the knowledge and skill and the application of that knowledge and skill to the standard of performance required in employment.*

It can be seen from this definition that the terms "specification" and "performance" are used. In this respect Competency Standards are similar to "technical" standards such as Australian Standards - they specify an outcome in terms of the required performance.

Competency is not simply about performing a narrow task in a controlled environment, but also encompasses the requirement to:

- perform individual tasks (*task skills*)
- manage a number of different tasks within the job (*task management skills*)
- respond to irregularities and breakdowns in routine (*contingency management skills*)
- deal with the responsibilities and expectations of the work environment (*job/role environment skills*), including working with others.

**BLACK COAL COMPETENCIES**

The first version of the Black Coal Industry’s Competency Standards were developed by industry “subject matter experts” in 1993. These covered four areas:-

1. Underground production
2. Open-Cut production
3. Mechanical Engineering
4. Electrical Engineering.

Extensive review of these standards commenced in late 1996, and have just been completed in January 1998. There are now a total of ten areas:-

1. Core (entry - level and common to everyone)
2. General (able to be accessed by everyone)
3. Underground production
4. Open-Cut production
5. Coal Preparation
6. Mechanical Engineering
7. Electrical Engineering
8. Building & Construction
9. Water & Waste Water
10. Management and Leadership

The competencies in the first five categories are largely unique to the coal sector and have involved extensive work by industry people to ensure that they reflect the reality of work. The last five categories have been largely adopted from other industries who have had relevant competencies, avoiding the need for the coal industry to "reinvent the wheel". In some cases these competencies "imported" from other industries have been modified/adapted to reflect specific and/or additional requirements of the coal industry. An example of this is the Mechanical Engineering competencies which have been adopted from the Metals and Engineering industry standards, but have had "overlays" developed which contain specific coal industry requirements.

Two aspects of these new Black Coal Industry Competency Standards require particular mention.

Firstly, in the Underground Production section there are now approximately 24 "units of competency" covering "underground statutory management functions" - Deputies, Undermanagers and Mine Managers. These have been developed as a direct result of the recommendations contained in the Taskgroup 3 report arising from the Wardens Moura 2 Inquiry, and have involved extensive development, validation and consultation among affected groups in NSW and Queensland.

Secondly, it is worth noting that a set of "Leadership and Management" competencies have now been included in the Black Coal Industry Competency Standards. These have been adopted from a set of competencies referred to as the "Frontline Management Initiative", which was a Federal Government response to the Karpin Report which highlighted the need to enhance supervisory and management competencies across most Australian industries. We now have in the Black Coal Industry Competency Standards not only the "technical" aspects of competency, but also the "soft" competencies - people management, planning, problem-solving, etc.

Now that Industry Competency Standards exist for both "underground statutory management functions" and "leadership/management", discussions have already commenced between a range of parties (NSW Mining ITAB, Qld Mining ITAB, DMR, DME, CMQB, Qld Board of Examiners, etc) to explore a range of issues and potential for integration/improvement of existing training systems in this area. The issues include:-

- use of the Competency Standards for course content;
- use of the Competency Standards for assessment;
- integration of industry-wide and enterprise-specific training and assessment;
- re-assessment of competency; and
- continuing education to maintain competency;

Addressing of these issues will require considerable industry input in the next 12 months.
ENTERPRISE PERFORMANCE REQUIREMENTS

The coal-mining industry, along with a wide range of other industries in Australia, has increasingly been adopting relevant components of what could be called in a broad sense, quality-based systems and/or philosophies. These may have names such as QA, TQM, TQC. Some organisations have spent considerable effort of comprehensive programs and some have invented their own hybrids or just adopted some components.

Whether through the adoption of such programs or not, a major theme in any business in the 1990's has been the concentration on outcomes. Similarly, in the training area, the major shift in the past decade has been to stop concentrating on the inputs of the process (the design of the training program and the qualifications of the teacher) and to concentrate on the outputs (whether the person had the required skills, knowledge, attitude at the end of the course). The specification of the required outcome is of course a competency standard.

How useful is an Industry Competency Standard to an individual coal mine? We've all heard the argument that each mine is different, so how useful can Industry Competency Standards be? They can only be useful to the extent that Competencies are common across the industry. The commonality across the industry is far more widespread than the differences.

Industry Competency Standards should be the starting point for the development of your own enterprise performance requirements for how work is to carried out by people at your mine. Unless you have generous resources and even more time, most of what you need to define your enterprise requirements will exist in the Industry Competency Standards. Industry Competency Standards may be the starting point, but rarely will they be the end point. They do require customisation to incorporate the differences which will inevitably exist from mine to mine. This customisation rarely involves major alteration to the Industry Standard, but it must occur to address the particular risk factors and the procedures established to address those risk factors at each individual mine.

Once Enterprise Competency Standards have been developed for a mine by customising the Industry Competency Standards, how does this allow us to achieve our enterprise performance requirements? The answer to this is simple. You now have the standards defined which allows the development of a training program (if required), but more importantly you have the standards to allow judgements to be made to determine if a person has the skills to the level required by the mine's performance requirements. This is the process of assessment, and it is the second key component in the Competency-Based Training (CBT) system.

ASSESSMENT

The way to measure whether the standard has been met in the CBT system is by assessment.

Assessment is defined as: The process of collecting evidence about competency and making judgement about whether or not competency has been achieved.

To ensure that the outcome has been achieved, this assessment needs to be against the relevant competency standards (whether industry or enterprise).
Gaining Competency

A person may acquire competency in a number of ways:-

**Training** - formal or informal training, either "off-the-job" or "at-job"

**Development** - formal or informal activities such as job rotation, project teams, acting in other roles, etc.

**Learning** - covers a wide range of other activities through which a person gains skills and/or knowledge, including activities outside work

Recognising Competency

An important element of CBT is the recognition of competencies which people gain through formalised training/development activities, but also those competencies that they already possess. This is sometimes called RPL (Recognition of Prior Learning) or RCC (Recognition of Current Competency), and simply involves the person undertaking the assessment against the required standard (sometimes called a "challenge test").

Whether the person needs to acquire the competency (through Training, Development or Learning) or already holds the competency, assessment is still used to determine if he/she is competent against the competency standard.

**Mine Manager's Authorisation System**

If the steps outlined so far are followed, then it follows that this is a very effective way for the designated Mine Manager to ensure that his "duty of care" has been fulfilled in his appointment of people at his mine to drive machines, maintain assets, operate processes, and generally perform work at the mine.

But how can this be done without "bogging the mine down with paper warfare"? It is inescapable that some traceable system (whether it is paper- or electronically-based, or both) is required. But there is no necessity to invent new systems. The systems of issuing Mine Manager's "authorisations" should be linked to your mine's assessment systems. The logic of
this is that the assessment system is based on achievement of the required Competency Standard, so the Mine Manager can be sure that the person has met the requirements needed at that mine.

The system does need to have some "rigour". It needs to be:-

1. documented in terms of policies and procedures;
2. demonstrably adhered to by all persons at the mine in all circumstances (it defeats the purpose if anyone is allowed an "exemption");
3. monitored and audited;
4. capable of providing a "due diligence" defence in the event of an incident/investigation/prosecution.

Re-Training

The notion of "re-training" people requires serious challenging with the advent of the CBT system. Why would mines devote large amounts of resources to putting people through "refresher" training courses? Why de-motivate people by putting them in a class-room situation and teaching them content which they may already know?

There is no logic to "refresher" training unless you have identified that there is a need for the knowledge/skill to be regained or enhanced. The way to identify if there is such a need is to carry out "re-assessment". This can be done in a range of ways, and does not necessarily involve the person completing all assessment events which may have been required when he/she was assessed as competent in the first instance.

One significant advance undertaken by some mines in recent times in NSW has been the determining of the interval (in one case ranging from 1 to 4 years) of when re-assessment of a competency needs to be carried out according to the "risk ranking" of the task. This ensures that re-assessment is targeted to competencies according to their risk, not on some arbitrary measure such as a blanket requirement for re-assessment every 12 months.

EVALUATION

One of the critical elements in any training system (whether it is CBT or not) is the capacity to evaluate whether the organisation's investment in training is worth it or not. It is reasonable to criticise trainers, training managers, and other managers for not setting up proper ways to evaluate the effectiveness of training itself, but also effectiveness in terms of return on investment. This criticism does not just apply to mining, but can be levelled across most industries.

The days of evaluating training solely on the "feel-good" factor (how the participant felt at the end of the course) are almost over. Significant research has been done over a long period of time on ways to evaluate the effectiveness of training, including analysing returns on investment.

Volumes have been written on this area, and trainers/training managers should be able to uncover large amounts of research. One of the better summary articles in the opinion of the author is by Ann Evans and appeared in the March 1996 issue of "Training and Development in Australia". The title of the article is "Are you Spending your Training Dollar Wisely: Evaluating the Return on Investment in Training"

The article identifies four levels of evaluation of effectiveness of training:

<table>
<thead>
<tr>
<th>Level 1 - Reaction</th>
<th>Measuring the feelings and perceptions of participants in a training activity - the &quot;smile sheets&quot;</th>
</tr>
</thead>
<tbody>
<tr>
<td>Level 2 - Learning</td>
<td>Measuring the participant’s ability to perform a task or demonstrate desired behaviours following a period of training</td>
</tr>
</tbody>
</table>
Level 3 - Behaviour  Measuring the participant’s changes in on-the-job behaviour.
Level 4 - Results  Measuring tangible improvements in business results in the effects on the bottom line performance of the business


This model is easily understood by both trainers and others, and it would not be too difficult to establish relevant evaluation methods for a mine-site’s training based on the four levels outlined.

Increasingly managers, CEO’s, shareholders and other stakeholders will be asking trainers and training managers more about their training systems and their effectiveness. The Competency-Based Training system has most of the answers.
Use of Simulation to Support Mining Industry Operators

N Harper¹ and B Harper²

This paper presents two design examples that illustrate instructional issues in the design and development of interactive computer simulations. The examples have been designed as exploratory tools that support a real activity for mining equipment operators.

The first example, the Australian Conveyor Engineering simulator, was designed as not only a simulation tool, but also a sales and marketing tool. Simple, yet deceptive in its complexity, the conveyor simulator allows users to simulate the actions of a conveyor in a real-world underground environment. The user can adjust a wide range of options relating to the conveyor, and see the results (belt tension, power output of drive head etc) not only in mathematical and chart form, but also through a graphical representation, the conveyor. Components under stress react by glowing yellow, then orange, then red. Users can test a given situation before actually implementing the design underground.

The second example, the Joy Mining Machinery 12CM12 continuous miner simulator, has been designed to support operators in the initial phase of training for use of the continuous miner, where training on the job could be dangerous and lead to expensive losses. The simulator allows potential operators to perform all miner operations on-screen while using a multimedia training program. In an attempt to add more power to the simulation the program also incorporates a replica of the actual machine controls, so the user can operate the simulator not through a mouse and keyboard, but also use the actual controls.

INTRODUCTION

The mining industry has developed a comprehensive view of the need to have production personnel well trained in the use and maintenance of mining equipment. A strong commitment from the industry to this view has been instrumental in creating an efficient, profitable and productive industry based on modern production processes, sophisticated equipment and a multi skilled workforce. As the industry continues to develop, and the skills and knowledge needed by productive workers continues to expand, the training processes adopted by the industry will need to develop in parallel, not only making use of the latest advances in learning theories, instructional design and technology, but also using these new ideas in more efficient ways.

The mining industry has a high level of mechanisation with the production personnel, for the most part, working in small groups in isolated locations. Multi skilling of this workforce has been instrumental in improving the efficiency of the industry over the past ten years with much of this efficiency coming from the ability of workers to carry out routine maintenance and repairs to the complex equipment. For the production personnel to take on this type of role, they need to be well trained in the functioning and routine repair procedures for this machinery.

The successful transformation of learning and accomplishment in the next decade requires the effective bringing together of two agendas - an emerging consensus about learning and training and well integrated uses of technology. Each agenda alone presents possibilities for training design of a very powerful sort. In the past, approaches that consider trainees as active learners and, that aim for trainees to understand content and be able to think were seen as important for only highly skilled technicians. Now in contrast these goals and approaches are urged as priorities for multi skilled production personnel in many industries, and in particular in the mining industry. Use of new media technologies to support this training process is seen as an essential tool in increasing the efficiency and effectiveness of the training process.

Training materials developed to control and maintain complex machinery need to develop in trainees not only appropriate knowledge, but also a broad range of specific skills based on use of the machinery. Production personnel in the mining industry do not have high levels of literacy so the necessary training could be well supported through clear initial specification of goals, the use of simulation of the skills needed, constant review of the key concepts and

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assessment of the learners achievements on a regular basis. This instructional approach should also be supported with achievement of the stated goals being recognised and conveyed to the learner.

**SIMULATION AS AN INSTRUCTIONAL STRATEGY**

Much of the complex and expensive equipment that is used in underground mining needs to be constantly involved in the production process. In order to have operators that optimise the production process, they need to be skilled and expert in use of this equipment with minimal time spent on use of the equipment for operators to develop skill.

Simulation of such equipment in providing a realistic, 'risk free', learning environment has the potential to support the skilling of operators with minimum training time used in the production line. Computer based simulations are programs in which the computer acts as an exploratory tool (Bliss and Ogborn, 1989), supporting a real world activity while helping users understanding of processes involved in controlling a complex dynamic system which may otherwise be inaccessible.

The current level of sophistication of interactive multimedia applications provides an incentive for designers to produce software which fully utilises the capabilities of such applications. This is particularly evident in many of the simulation based packages being developed today which exhibit a tendency to move away from the earlier reliance on a 'pre-set', 'fixed and repeatable outcome' model which provided a very simple approximation of the real world that it was trying to mimic.

There is a considerable volume of literature on the value and nature of simulations (Gredler, 1996). They may be defined as the dynamic execution or manipulation of a model of an object system for some purpose (Martin 1988). A simulations is considered as "... a special kind of model representing a 'real world' system, governed by a set of rules" (Crookall et al. 1987). During the use of such models, the user often comes to see the simulation itself as a 'real world' in its own right. Such models represent systems as either "in-place of" or a "bring to life" format. The question posed by some as to whether the terms 'model' and 'simulation' have similar meanings in this context may prove a pivotal point around which a better understanding of the cognitive outcomes for the users may be achieved.

The 'in-place of' interpretation of representation applies to the 'standard' widely accepted notion of a model while, as suggested by Crookall et al. (1987), both the "in-place of" and "bring to life" representations may be applied to and equated with simulations.

The exact nature of what constitutes a 'true simulation' is not agreed upon amongst researchers or designers alike but we would suggest that, commonly the goal of the simulation must be to provide interactive experiences which approach or mimic the 'real world' experience as closely as possible.

We would suggest however that within this 'prescription', simulations fall into two broad categories, those which we might call 'Illustrative simulations' and those we might call 'Interactive simulations'. We would suggest that these categories are not mutually exclusive and that much of the software developed for teaching, learning and gaming environments, use simulations exhibiting varying degrees of hybridisation of the two types.

Purely 'Illustrative simulations' are those which require a minimum of interaction and provide simple feedback, often in the form of images. The user is not required to provide significant input data, often only being required to 'click on' the start button. If the simulation is under pinned by algorithms, it is usually of a 'closed type' and as such is not predictive of 'real world' outcomes. The outcomes are fixed and at best such simulations provide little more than an 'illustration' of the process. As a consequence, they are less educationally useful as they do not allow the user to make inferences regarding cause and effect relationships nor do they allow for the testing of hypotheses. Such 'simulations' in their simplest forms are little more than animated sequences depicting a process.

On the other hand, purely 'Interactive simulations' are highly interactive in that the user is able to fully manipulate inputs and receive and manipulate output in various forms from the system. In its most powerful form, the algorithms are such that they provide an 'open model' which fully mimics the 'real world' processes and can be fully predictive. User's also receive feedback on their actions, thus allowing them to study 'cause and effect' relationships and test hypotheses.
Such simulations promote the adoption by the user of the active learner mode and in so doing support the active construction of knowledge by the student during the process of solving a problem.

Placing such simulations within a multimedia environment and providing an interface which takes full advantage of the capabilities of such environments can provide unparalleled support for the user.

Regardless of the type or format of the simulation, the overriding purpose for simulating and modelling systems remains; to provide a substitute experience for those processes and systems which by reason of cost, scale, time or risk, would not normally be accessible and the ultimate aim of the developer must be to meet such essential criteria as:

- Producing a simulation that emulates as closely as possible the ‘real world’ experience.
- Design decisions are based on some appropriate educational paradigm.

TWO MINING EXAMPLES

Australian Conveyor Engineering Simulator

The Australian Conveyor Engineering (ACE) conveyor simulator was designed to allow mine personnel to perform "what if" simulations of the load and power applied to a conveyor system before making changes to the real world system.

![Australian Conveyor Engineering Simulator](image-url)

**Fig. 1 - The Australian Conveyor Engineering Simulator**
The program opens with all conveyor values set to default - the conveyor carrying no coal - and the simulation begins. The conveyor is represented visually in the top portion of the screen in three-dimensional form. The fact that the conveyor system is operating become more apparent as coal is supplied to the system through the input controls in the bottom portion of the screen.

By adjusting the numerical input controls the user has the ability to adjust the amount of coal which will be placed on the conveyor system (tonnes per hour), the set point for the tripper drive (kilonewtons), the pressure applied by the winch (kilonewtons), the belt rise over the whole conveyor system (metres), and the total conveyor length (metres).

The arrows on each side of the inputs are operating by clicking with the mouse until the desired setting is reached. Short clicks increment the setting by one unit while longer clicks can be used to increase the setting by a greater amount. The left hand arrow reduces the setting and the right hand arrow increases the setting. The controls are not designed to simulate the real world controls for the conveyor system, as this was not a goal of the simulation, so they are relatively simple, clear and intuitive.

Feedback is provided through the numerical outputs (located adjacent to the input controls), the belt tension graph (located on the right hand side of the screen) and the three dimension model of the conveyor system. Output consists of the power required by the drive head (kilowatts), the tension of the conveyor belt measured just before it reaches the drive head (kilonewtons), the power required by the tripper (kilowatts), the tension of the conveyor belt measured just before it reaches the tripper (kilonewtons) and the tension of the conveyor belt measures just after it passes through the tripper (kilonewtons).

A graphical representation of the length of the conveyor system verses the tension occurring in the system is located to the right of the numerical outputs. This belt tension graph provides users with a graphic representation of the relative tension on the conveyor belt at different points within the system.

The three dimensional representation of the conveyor system which dominates the screen provides users with a more accessible and visually appealing form of output. It provides a "rough" guide to the current state of the system, which can be used in conjunction with the numerical and graphical data to create a detailed analysis.

![Fig. 2 - The Australian Conveyor Engineering Simulator with coal on conveyor.](image-url)
Using the numerical outputs, the model displays warning indicators (visually represented as colour changes) to determine whether the power required by the motors or the tension applied to the belt is likely to cause damage to the machinery. This becomes a rough guide to the amount of coal it is safe to load on the belt given the input parameters. The warning indicators occur on the drive head and tripper motors and on the conveyor belt before the drive head, before the tripper and after the tripper. By viewing these colour changes the user can determine whether their input settings are appropriate for the conveyor system, or whether they are likely to cause damage to the equipment. By combining the colour changes with the numerical outputs, the user can also determine the exact tension at each of the measuring points, as well as the power demands for the drive head and tripper motors.

If the user enters all required inputs and finds that the system will not be able to function correctly, they can then 'experiment' with the inputs performing "what if" simulations to determine the most appropriate settings for their conveyor system.

The ACE conveyor simulator provides an interactive experience which attempts to mimic the real world operation of a conveyor system, but fails to mimic the actual controls and physical operation of the machine, and therefore can be considered to be a hybrid 'Illustrative simulation' and 'Interactive simulation'. The simulator has elements of an 'Illustrative simulation' - simplified controls, relatively simple visual representation of a conveyor system - while taking advantage of some of the highly interactive features of an 'Interactive simulation' - fully manipulate inputs, displays output in various forms from the system, and immediate user review of their actions. This type of system strongly supports active learning in that the user can study cause and effect relationships and test hypothesis in a safe, controlled environment.

Joy mining machinery 12CM12 continuous miner simulator

The second example, the Joy Mining Machinery 12CM12 continuous miner simulator, has been designed to support operators in the initial phase of training for use of the continuous miner, where training on the job could be dangerous and lead to expensive losses. The simulator allows potential operators to perform all miner operations on-screen while using a multimedia training program. The simulator was designed to allow users to learn how to operate a 12CM12 continuous mining machine through experimentation. The simulator combines rich three dimensional animation with replica controls to enrich the learning process.

The 12CM12 continuous miner is operated by a small team of personnel, each with specific tasks to perform. The simulator is designed to augment the training of the miner driver. The driver operates the movement and coal shearing on the machine through a radio remote control unit. The remote control unit features a collection of switches which are used to send instructions to the machine.

The 12CM12 simulator is part of a larger computer based training program which features maintenance procedures, electrical and hydraulic circuit education and fault finding trees.

The simulator initially presents the user with a cutaway side view of a roadway showing the strata above the roadway, the floor below, and the miner side-on (three dimensional model). The miner is positioned at the left hand side of the screen with the roadway continuing to the right. Below the miner is a graphical version of the radio remote control panel which is used to operate the machine.

The graphical version of the remote control features a collection of switches which are used to operate the machine. By moving the mouse over a switch, holding the mouse button down, dragging the switch up, down, left or right and releasing the mouse, the user can trigger an operation on the machine. The operation is displayed by the three dimensional model, which was created using the actual engineering drawings for the 'real world' machine, ensuring visual accuracy. The model moves or animates to display what would happen on the actual machine. All operations are timed and synchronised with the actual machine, ensuring that the user gets a true indication of the real world activity which is being simulated.
Fig. 3 - Continuous Miner Simulator

Fig. 4 - Continuous Miner Simulator in different positions
The simulator can perform a number of operations including tramming, shearing, gathering head operations, cutter head extend and retract, cutter head on and off, stabilising jack operations, and full conveyor swing operations. When an operation cannot be displayed from the side-on perspective, an inset window appears in the centre of the screen to show the operation.

![Photo of Replica Control](image)

**Fig. 5 - Photo of Replica Control**

In an attempt to add more power to the simulation the program also incorporates a replica of the actual machine controls, so the user can operate the simulator not through a mouse and keyboard, but by using a replica of the actual controls. Although internal operations were created from scratch for serial communications with the computer, the replica remote has the same visual appearance as the remote control used on the 'real world' machine. The miner simulator allows users to fully manipulate inputs and outputs, use the controls which are used in the 'real world' and extend the simulated aspects of the learning experience in the tradition of 'Interactive simulation'.

**Conclusion**

The mining industry makes use of complex and expensive equipment that must be operated by skilled miners. Developing operator skills must be achieved with minimum down time for the equipment in circumstances that minimise risk to operators, co-workers and the equipment. One approach is to simulate the skills necessary to develop well trained operators using sophisticated visual representations of models that can be manipulated in ways that simulate as closely as possible operator skills.

The two simulations described in this paper are now in use in various mines in Australia, as well as internally at Joy Mining Machinery in the United States. Evaluation of the learning outcomes will be a priority before development of further more extensive products based on the same instructional strategies.

**REFERENCES**


Improving Coal Mining Production Performance Through the Application of Total Production Management

J C Emery

ABSTRACT

This paper describes the application of the Total Productive Management (TPM) technique as a performance improvement initiative for a coal mining operation. It discusses the objectives of TPM, with the driver for improved production performance being the Overall Equipment Effectiveness (OEE) of the equipment or process, and with the development of "ownership" as the behavioral approach to equipment management and continuous improvement through cross-functional and area-based teams. It illustrates the concept of equipment management as defects management.

The scope for application of TPM to the coal mining industry is immense. The harshness of the operating environment can be a major generator of equipment defects, and a current paradigm in the industry accepts these defects as an unavoidable outcome defining maintenance costs in this environment. However recent benchmarking studies have highlighted that maintenance costs per operating hour in some mining operations are more than double the vendor's estimate of "best practice". The paper refers to these studies which also compare maintenance costs of fixed and mobile plant and equipment to "best practice" outcomes in comparable process industries.

The ultimate goal of any operating strategy must be to translate results to the bottom line through adding revenue from increased volume and quality of operations output, better safety performance, and reducing costs of production through lower operating and maintenance costs. These lower costs result from removal of defects generators, improved maintenance planning, and identification and reduction of hidden operating costs resulting from poor equipment maintenance. The paper discusses methods of evaluating the progressive improvements brought about by a successful TPM strategy to achieve this goal in a highly visible format to provide the incentive to both management and the workforce to push on for more improvement.

Finally the paper outlines the minesite procedures required for successful implementation of TPM to sustain these desired results for all stakeholders. It suggests that TPM can be integrated with existing business improvement initiatives by structuring these other minesite programmes (safety, cost reduction, restructuring, capital replacement, etc.) into the "Eight Pillars of TPM" framework as part of the overall business plan. Resulting interface redundancies can then be identified and eliminated, and a timeline developed for effective implementation of the overall minesite initiatives programme.

INTRODUCTION TO THE TPM PROCESS

TPM As A Business Improvement Initiative

Total Productive Management (TPM) is a proven concept of equipment management for maximising capacity, productivity, quality, employee morale, safety and bottom line results.

Like the Quality movement, TPM had its genesis in the Japanese car industry in the 1970's. However, it has only been in recent years since the late 1980s that TPM has started to rapidly spread throughout the western world, significantly improving the operational areas of initially manufacturing and now mining industries. TPM has evolved as a vital and necessary response to the need to develop a competitive advantage by substantially improving capacity through enhanced plant and equipment performance along with output quality, while significantly reducing not only maintenance costs but

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1 Principal and Director, Devman Consulting Pty Ltd. Consultant to The Centre for TPM (Australasia).
overall operating costs. Successful implementation of TPM has resulted also in the creation of safer and more environmentally sound workplaces.

For change to be sustainable, it requires the focus of a tangible and effective driving mechanism. TPM applies the principles and practices of quality management, especially "prevention at the source", but focused on plant and equipment rather than on customers as in TQM. TPM uses Overall Equipment Effectiveness (OEE) as the driver that focuses the TPM initiative and provides the vehicle for sustained continuous improvement, with all employees becoming involved in preventing defects from developing in plant and equipment. Defects are identified at the earliest possible time so that they can be removed in a cost-effective manner before they lead to deterioration in overall equipment or process performance. TPM challenges the traditional approach of "I operate, you fix".

An important outcome of this new approach to equipment management, supported by many success stories throughout the world in a variety of operational industries, is that TPM cannot be implemented by a maintenance department alone. TPM is a company wide improvement initiative involving all employees. Changes now occurring within the Australian coal industry with the establishment of workplace agreements and the overhaul of work practices, provide the environment for the implementation of TPM to become strategically important for a globally competitive coal mining operation.

Objectives

Although each enterprise may approach TPM in its own unique way, most approaches recognise the importance of measuring and improving Overall Equipment Effectiveness (OEE), and the need to create a sense of ownership by the plant and equipment operators, maintainers and support staff to encourage prevention at the source. The three main objectives of TPM are defined as:

- to maximise the Overall Equipment Effectiveness (OEE) through loss analysis;
- to develop Ownership of equipment through area-based teams; and
- to promote Continuous Improvement through area-based and cross-functional teams.

Equipment Management as Defect Management

Defects are generated and "flow" into plant and equipment due to various reasons, some of which are:

- the poor initial design, or subsequent changes to the design parameters due to output requirements changing;
- the method and practices adopted in operation of plant and equipment and the environment in which operations are carried out;
- the imperfection in maintenance materials and spares as sourced from stores or imparted during handling or assembly; and
- the consequences of any failures which occur within the plant or equipment.
The principle of identifying equipment management as defect management, as depicted in Fig. 1.1, was developed by Dupont and is embodied in the Manufacturing Game (Lidet 1994). Defects are "stored" in the plant and equipment and progressively eliminated through maintenance activities. An "overflow" results in breakdown. The "level indicator" of defects to trigger planned maintenance activities is the outcomes of inspections, condition monitoring, etc. However, different approaches to equipment management in controlling this system will have different impacts on both the plant performance and total cost structures.

The basic principle of TPM is the recognition and elimination of these defects AND the defect generators (as the root causes of failure) which lead to accelerated deterioration resulting in poor performance and ultimately in plant and equipment failure. These activities of recognition and elimination are carried out by the miners, plant operators and maintenance personnel as cross functional and area based teams.

A Paradigm Shift To An Operator-Ownership Environment.

Over the past ten years we have seen the pendulum of change in the coal industry swing towards a multi-skilled workplace. However, as companies have gone through this experience, the importance of the issue of "ownership" has become apparent. Through bitter experience, many companies have now come to realise that the pendulum may have swung too far. Without a sense of "ownership", employees tend not to care for equipment. Although multi-skilling has been successful in creating a more flexible workforce, experience now highlights that, while employees move from equipment to equipment, or area to area, they lose the motivation to seek out basic equipment conditions problems or defects which, if left unchecked, will cause failure in the future.

An area-based team approach that promotes the development of both base skills and mastery skills provides a means to achieve both flexibility and ownership within the workplace. Correctly formed area-based teams create an environment where employees recognise the benefits for themselves in adopting the proper way to operate their equipment and how best to care for their equipment by maintaining basic equipment conditions. TPM implementation experience has shown that there is a definite relationship between failures and these basic equipment conditions of correct lubrication, no contamination, and no looseness.

This focus on equipment defects has a large bearing on the way that everyone at the minesite becomes involved with TPM. Defects are often difficult to identify and correct because they are traditionally accepted as the norm. All employees need to adopt the attitude of questioning whether their individual actions are focused on avoiding defects or merely addressing the issues associated with defect removal. The paradigm shift required by management and all employees at the mine is to accept that they are able to, and want to, identify and correct equipment and process defects and then find their source so that they can be avoided in the future. This paradigm shift is a major ingredient in the implementation of the TPM process.

So, again, it is fundamentally important to realise that TPM is not a maintenance management technique but is a process that is applied throughout the total mine organisation as a framework for the application of business improvement initiatives.
The Eight Pillars of TPM.

The Centre for TPM (Australasia) approach, resulting from development and experience gained over the past five years, is based on:

- establishing a structure which will foster the introduction of continuous improvement techniques and the adoption of those techniques as part of the normal business processes;
- recognising that TPM success stems from a management commitment to involving all employees to assess and question equipment and process losses in all areas of the operation;
- challenging existing mind-sets;
- setting in place the appropriate tools and skills to ensure that the improvements made are sustained and expanded over time.

The “Eight Pillars of TPM” that form the framework supporting this process are:

- Focused Equipment & Process Improvement;
- Operator Equipment Management;
- Maintenance Excellence;
- Education & Training;
- Safety & Environmental Management;
- New Equipment Management;
- Process Quality Management; and
- Administration & Support Systems Improvement.

These Pillars of TPM interact in a polychronic way to form a support structure to underpin and promote the improved performance of the whole company through the TPM process. To successfully apply the principles of TPM, management and the workforce must realise and mutually accept that these Pillars require the whole company to be involved to take advantage of the significant gains that can be achieved.

The first five Pillars are most commonly applied in the operating process during the initial stages of the TPM Implementation Plan. As the culture of the workforce changes and equipment effectiveness improves, the remaining three Pillars complete the supporting loop to ensure the perpetuation of the improvement outcomes in the operating process. The application of the five initiating Pillars in a coal mine environment is discussed below.

APPLICATION OF TPM TO THE COAL INDUSTRY

Scope for Application.

The scope for application of TPM to coal mining is immense. The harshness of the operating environment in most coal mining operations is a major generator of defects. And the coal mining industry paradigm accepts these defects as the unavoidable outcome of this environment. This attitude results in loss of productivity due to

- equipment failure or other unplanned stoppages, both recorded and unrecorded;
- equipment in a mining operation or a coal washing plant process idle while waiting on set-up time for, or availability of, critical equipment; and
• reduced output or increased waste due to equipment or processes operating below OEM specifications.

These losses also result in, or at least reinforce, the lowering of workforce morale due to frustration in malfunctioning equipment, and the resulting outcomes of absenteeism, poor safety performance and industrial unrest.

![Image: The Total Cost of Maintenance](image)

Source: Kennedy (1997).

**Fig. 2 - The hidden costs Of poor equipment management**

The current paradigm also results in maintenance strategies that incur major costs to the operation. Fig. 2 illustrates the exposure of the operation to those costs, both exposed and hidden, associated with consequences resulting from poor equipment management. A recent benchmarking study in maintenance practices in the mining industry by Strategic Industry Research Foundation (SIRF) (Holmes, 1997) has highlighted that, in over 50% of the mines included in the study, the maintenance costs per operating hour were more than double the equipment vendor’s estimate of “best practice”. Also the cost of maintenance of mobile equipment, ignoring consumable costs, was up to one third of the capital value of the fleet. This compares with expenditures on maintenance in best practice process industries of between 1% and 3% of capital value. The maintenance cost, again ignoring consumables, of mine fixed plant tended to be in the range of 2.5% to 6.5%, compared to best practice for comparable process plant of 3% or in the range of 3% to 4% in exceptionally difficult circumstances.

Another outcome from the SIRF benchmarking study was that, contrary to the currently accepted belief that age of equipment, mine conditions and equipment vendor are the predominant factors defining maintenance costs, these were of second order importance compared with the factors of work practices and culture at the site.

**TPM Framework In A Coal Mining Operation.**

The production losses for an item of equipment, such as a dragline or longwall face system, or a production process operating at a mine, such as a truck and shovel fleet or a coal washing plant, can be represented, on a time related basis, by a block diagram relating available time to the effects of loss categories as shown in Fig. 3. Overall equipment effectiveness is defined as the ratio of value adding time, after accounting for all losses, to scheduled production time expressed as a percentage. Although in continuous operations, planned maintenance time is included in scheduled operating time, in non-continuous operations this activity is excluded to remove the mechanism of skipping planned maintenance to improve OEE.
The **Focused Equipment and Process Improvement Pillar** is normally the starting point for a TPM implementation programme, focused on strategically important equipment or process. The procedure, as illustrated in Fig. 4, involves the establishment of the "current situation" from continuous recording of losses or by sampling, and the identification and analysis of losses identified using first, second and third level pareto charts. Solutions are developed for the reduction of losses, and the resulting improvement of OEE, by cross-functional teams using root cause analysis and PDCA cycle techniques. Following trials, refinement, and implementation of successful solutions, each cross-functional team is disbanded. The task of achieving further gains to OEE, by continuous improvement techniques applied to that equipment or process, is handed over to the relevant area-based team.

### Fig. 3 - Production Losses – overall equipment effectiveness

The **Focused Equipment and Process Improvement Pillar** is normally the starting point for a TPM implementation programme, focused on strategically important equipment or process. The procedure, as illustrated in Fig. 4, involves the establishment of the "current situation" from continuous recording of losses or by sampling, and the identification and analysis of losses identified using first, second and third level pareto charts. Solutions are developed for the reduction of losses, and the resulting improvement of OEE, by cross-functional teams using root cause analysis and PDCA cycle techniques. Following trials, refinement, and implementation of successful solutions, each cross-functional team is disbanded. The task of achieving further gains to OEE, by continuous improvement techniques applied to that equipment or process, is handed over to the relevant area-based team.

### Fig. 4 - Focused Improvement & Process Improvement Model

The **Focused Equipment & Process Improvement Model** involves the establishment of the "current situation" from continuous recording of losses or by sampling, and the identification and analysis of losses identified using first, second and third level pareto charts. Solutions are developed for the reduction of losses, and the resulting improvement of OEE, by cross-functional teams using root cause analysis and PDCA cycle techniques. Following trials, refinement, and implementation of successful solutions, each cross-functional team is disbanded. The task of achieving further gains to OEE, by continuous improvement techniques applied to that equipment or process, is handed over to the relevant area-based team.
The Operator Equipment Management Pillar is introduced at the appropriate time to achieve a self-managed equipment-competent workforce. Operator equipment management is about "caring for equipment at the source" so as to ensure that the basic equipment conditions — correct lubrication, no contamination, and no looseness — are established and maintained. This is a staged implementation by area-based teams consisting of operators and maintainers for equipment or process areas. These stages might be as follows:

- Recognising equipment defects and making improvements so as to achieve Basic Equipment Conditions;
- Understanding equipment functions and mechanisms so as to achieve Zero Breakdowns;
- Understanding the relationship between production and basic equipment conditions so as to achieve Zero Production Defects; and
- Managing the workplace so as to achieve Zero Accidents.

This does not result in a take-over by operators of the maintenance function. However operators become responsible for knowing when they need to carry out the simple defect avoidance and maintenance service work themselves, and when they should call in the maintenance experts to repair problems which they have clearly identified.

![Maintenance Excellence Pillar](image)

Source: Kennedy (1997).

**Fig. 5 - Maintenance excellence pillar.**

The establishment of this shared task zone provides the maintenance organisation at the minesite with the time to focus its resources on the Maintenance Excellence Pillar to optimise reliability and equipment management support. As illustrated in Fig. 5, this involves the application of leadership, capability and maintenance management processes, together with maintenance planning and improvement methodologies such as reliability centred maintenance (RCM), maintenance process redesign (MPR) and benchmarking, to move the level of maintenance management along the "best practice" continuum towards maintenance excellence. Without the foundation of a clear and well-communicated maintenance management strategy supported by an appropriate organisation structure, human resources and knowledge base, the introduction of TPM almost always fails.

Implementation of the Education and Training Pillar supports the progress of these other Pillars, and requires a significant commitment to education and training both to challenge mind-sets and impart new skills. TPM is a "new way of working" for an organisation, focusing on the importance of equipment management for the success of the company. This Pillar ensures that this focus is clearly understood and held by all employees and including management. Following initial awareness workshops for all employees, TPM training should then be achieved, wherever possible, through "dirty education" processes where you "learn as you do" on the job.
The **Safety & Environmental Management Pillar** employs the resulting change to the behavioural approach of the workforce culture and also the safe work environment resulting from the improved basic equipment condition state of plant and equipment.

The other three Pillars are introduced at the appropriate time. These eight Pillars of TPM are applied to the overall business through the three areas of implementation activity that, as shown in Fig. 2.5, are OEE Improvement, Maintenance Improvement, and Workplace Effectiveness.

**Current Progress in Introduction of TPM in Australian Mines**

Although TPM is well established in the manufacturing industry throughout the western world, little information is available that describes the application and/or results of a TPM program implemented at mine sites in Australia or overseas. While many minesites in Australia have introduced some of the techniques described in this paper, either separately or as part of a previous TQM programme, with varying degrees of success, the author is unaware of any operations fully employing the TPM framework. However some pilot programmes are in progress and interest in TPM has been indicated by a number of mining companies that have attended TPM workshops run by The Centre for TPM.

![Diagram of applying the 8 Pillars of TPM through the three areas of activity](image)

Source: Kennedy (1997).

**Fig. 6 - Areas of implementation activity**

As an example of the outcomes achievable, the following improvements were observed by the author at Oaky Creek Mine in Central Queensland during implementation of particular techniques in the early 1990s as described below:

- **Focused Equipment and Process Improvement** (cross-functional teams) and **Operator Equipment Management** (area based teams) techniques applied to gateroad development resulted in a 65% improvement in metres per unit shift achieved in Main Gate 4 over previous main gate development rates in the No.1 Colliery, after cross-functional team project outcomes, and attention to ongoing improvement by the area based teams.

- **Operator Equipment Management** techniques applied through area based teams in the coal preparation plant resulting, in part, to an increased throughput of 8% to a record output from the plant.

Some North American mines have in the past had considerable success through commitment to techniques and working practices that are now embraced by the TPM Pillars. These include US mines at Cyprus Twenty Mile Coal Company, Colorado, Western Fuels Association Deserado Mine, Colorado, Sabine Mining Company, Texas, Homestake Gold Mine, South Dakota, and in Canada at Syncrude Canada, Alberta. All of these operations rank among the most successful and highest productivity mining operations in the region.
EVALUATING THE SUCCESS OF TPM STRATEGIES

The focus for improvement activities in TPM is the trend of the OEE of the equipment or process considered. Run charts of OEE will provide trends or patterns of improvement over a specified period of time. This technique is useful in providing the teams, and the workforce at large, with a comparison of OEE before and after implementation of a solution, to measure its impact, and celebrate in its success. Progressive changes in Pareto Charts are also useful in showing progress in a highly visible format that provides incentive to push on for more improvement.

Other means of measuring the success of a TPM strategy, in terms of the changes in effectiveness of the overall operation, include the progressive review of the company’s rating on operations innocence-to-excellence and maintenance innocence-to-excellence matrices developed by The Centre for TPM. The level of progress in organisational change is similarly gauged by progressive ratings on a culture innocence-to-excellence matrix using repeated employee surveys.

The success of a TPM strategy will translate to the bottom line through increased revenues from greater operations output, better safety performance, and reduced costs of production. Production cost reductions are achieved through lower maintenance costs (resulting from removal of defects generators), improved maintenance planning, and identification and reduction of the hidden costs of poor equipment maintenance. The continuous improvement techniques employed in the TPM process lend themselves to the establishment and charting of KPIs for measuring the progress in the critical success factors identified as affecting the achievement of these goals for the business enterprise.

TAILORING THE TPM PROCESS TO INDIVIDUAL ORGANISATION NEEDS.

A major characteristic of the TPM process is its flexibility of implementation. The order of introduction of the first five pillars and the variety of techniques applicable to the process provide the means of tailoring the application of TPM to suit the individual needs of a coal mining operation.

However some issues should be seen as underlying the introduction of TPM in any situation. Recent feedback suggests that the implementation model that works best involves the initial introduction of the process into a small number of targeted pilot areas within the mine operation as a learning experience, and for familiarisation of the management and workforce with TPM principles and techniques. The process is then cascaded progressively across the operation based on the early successes gained and the learnings achieved in those pilot areas by all stakeholders. Experience has also reinforced the need for a sound strategy in place to address the employee relations aspects of the implementation of TPM, including enterprise agreements, or similar understandings with the workforce, which include support for the process. This is of critical importance to demonstrate the strong commitment of management, from the CEO down, which is essential to the cultural changes involved in TPM. It is also essential in defining an agreed position with the workforce when issues arise which threaten the process, or the integrity of those employees involved in pilot schemes.

The minesite procedures required to successfully introduce TPM so that it will be sustained, and will achieve the desired results for all stakeholders, involve three main phases:

- **Awareness and Preparation**
  
  Creating a critical mass of initial understanding within the organisation for the need and potential impact of TPM, determining an appropriate implementation strategy, and motivating participation to move forward.

- **Assessment and Planning**
  
  Identifying the “stake in the ground” and the most appropriate implementation methodology outlining the potential benefits, costs and resources required, drafting a realistic implementation plan with measurable milestones, and gaining senior management commitment together with that of a sufficient number of other relevant stakeholders.
• Implementation

Finalising the implementation plan and assigning initial resources, executing the plan with continuous feedback and regular reviews, and promoting the success to encourage progress so that expectations are realised.

However the methodology used must be flexible enough to ensure that the key issues of “how do we get all employees to contribute and participate in TPM”, and “how do we ensure that TPM is integrated into existing business improvement initiatives” are adequately addressed. The Centre for TPM (Australasia), based in Wollongong, NSW, provides information exchange, training and consulting support to achieve these outcomes.

One common reason put forward as to why TPM has not been introduced at a minesite is the current time and resource commitment to other improvement strategies. A suggested approach to implementing TPM to overcome this problem is to structure other minesite programmes (safety, cost reduction, restructuring, capital replacement projects, etc.) into the TPM Pillars framework as part of the overall business plan, remove identified interface redundancies, and develop a timeline within that framework for effective implementation of the overall minesite initiatives programme.

CONCLUDING REMARKS

TPM implementation is not a short term fix for an ailing maintenance programme or a non-performing coal mine operation. TPM is a company-wide business improvement initiative involving all employees. Like Quality, it is a continuous journey. Experience gained in Australia and overseas indicates that significant improvement should be evident within six months. However, full implementation can take many years to allow for the full benefits of the new culture created by TPM to be sustained. Suitable business planning horizons must be adopted to allow this to occur.

REFERENCES


Mine Maintenance – The Cost of Operation

O Kreilis¹ and T Singleton²

ABSTRACT

Increasing world competition puts pressure on sales volumes and prices. This in turn reduces potential profit margins. This in turn increases the focus on costs.

Increasing demands on quality and service puts pressure on delivery performance - the right product at the right time. This in turn reduces the scope for errors and delays - and this in turn increases focus on equipment reliability. In the mining industry, both costs and equipment reliability have one significant thing in common - they are driven substantially by maintenance.

Maintenance, once the Cinderella of the boardroom, is a pivotal function and demands management attention and, if managed well, can be a source of competitive advantage. They made the decision to put maintenance high on their agendas because they realised that good maintenance is a vital factor in achieving excellence.

Maintenance, because of its impact on return on capital, is a key driver of performance. By reducing maintenance costs, companies can improve their performance. Top managers are increasingly recognising that maintenance is an area in which they must be involved. As Australian mining and metallurgical companies look to the year 2000 and beyond, maintenance will become an increasingly strategic function, capable of delivering sustainable competitive advantage to those companies that get it right.

INTRODUCTION

In the current economic climate, minimising costs assumes even greater importance, so that equipment reliability must be stepped up to reduce delays. Equipment reliability means effective maintenance. Maintenance costs in the mining industries are commonly between 30% - 50% of minesite total operating costs. BHP Minerals spends alone between $1 and $1.5 billion each year on maintenance (Ellis, 1994).

Maintenance in the mining and metallurgical industries can benefit from successful examples of maintenance practice in industry generally, and vice versa. Maintenance improvement involves a vision of future requirements, the changes necessary and to achieve those improvements, and understanding how to accomplish them. Maintenance suffers by poor planning, and management that is too occupied by crisis maintenance to institute preventive maintenance. In this stressful atmosphere, production losses are reduced by overlarge inventories of replacements and spares and consequently capital costs are inflated.

The solution to these inherited problems lies in a complete rethink of hitherto purely technical maintenance. Maintenance should be a component of production activities. All participants in the workplace should be involved in the broader thinking absent in former traditional maintenance by involvement of equipment/spares suppliers and by involvement of efficiency, motivational and business specialists. Thus a new production-oriented maintenance is incorporated as a business management tool.

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MAINTENANCE: A STRATEGIC FUNCTION

The ultimate objective of the maintenance function is to provide competitive advantage by increasing the efficiency of maintenance actions and increasing reliability and availability of equipment through effective strategies, planning and continuous improvement. High levels of equipment reliability and availability improve product quality and delivery performance, reduce asset intensity, and also reduce direct operational and maintenance costs.

To achieve excellence in maintenance requires the following:

1. Maintenance goals and objectives set to suit the business
2. A strategy to achieve those goals and objectives
3. A system to measure and manage the maintenance function
4. The right resources

Goals and Objectives

Application Design

Firstly it is critical that the business objectives of a mining operation are set out correctly so the operating system criteria can be determined to match the business needs of the future. Given the economic climate, equipment supplied today has very little performance to spare in excess of the contracted requirements.

Particular attention should be made of the key performance criteria which result in production performance. For example, for a longwall mining system;

- Tonnes per annum = Average tonnes/week * Number of production weeks
- Average tonnes per week = Average TPH * Operating hours/week
- Average TPH = % of Process TPH
- Process design TPH = Dependant on longwall nameplate TPH and cutting cycle factors
- Operating hours/week = Process availability * Planned longwall operating time
- Planned longwall operating time = Total manned time/week - Planned maintenance time/week
- Process availability = Function of mechanical, electrical and operational downtime for equipment, systems and resources affecting longwall production time. (Operating time / Planned operating time )

Assuming a nameplate 3000 TPH longwall with a process design TPH of 2,000 TPH, average TPH of 1,500 TPH (75% of process TPH) and a process availability of say 55% for 45 weeks per year, 15 * 8 hr manned shifts and 3 * 8 hr planned maintenance shifts per week, 3.56 million tonnes per annum would be achieved.

However if process availability was improved to 65%, then 4.21 million tonnes per Annum would be achieved, with a probable increase in revenue of say $18 Million dollars, whilst similarly if the average TPH was increased to 1,770 TPH, or planned production time increased to 14 shifts per week whilst process availability was maintained, the same result would occur.

Clearly maintenance strategy takes it's place along side of equipment design, productivity potential and operations management (see Fig. 1).
Fig. 1 - TQM Cycle (Has the works business plan in a box on top and “Decommission” in the LH box)

By careful evaluation of the production system, a suitable strategy for optimising production and maintenance is achievable.

**Determine The Preferred State - Goals and Objectives**

The best performing companies have a vision, and have a coordinated maintenance strategy in place to implement that vision. The strategy is fully supported by corporate, plant and departmental management across the company. Maintenance that is treated as a strategic function has a substantial influence over and impact on profitability, and therefore deserves top management attention and adequate resources. On the other hand, companies where management treats maintenance as a cost to be contained rather than as an activity that creates value, will not perform well.

The basis of a winning maintenance strategy is planned preventative maintenance with a focus on continuous improvement. Maintenance to prevent in-service failures must, by its nature, be planned. The maintenance plan must identify the equipment components to be maintained to ensure that the maintenance is effective. It must detail how the component will be maintained to ensure execution is efficient and safe. It must define when it will be maintained - on failure, on usage or on condition. It must define who will execute the activities required to implement the plan. It must list the special tools and spares required. Finally, it must define success - the expected performance of the equipment and the resulting costs.

To win in the future, top management must understand the strategic nature of maintenance, and have a sufficient understanding of maintenance to participate in the management of maintenance.

**Maintenance Strategies**

A number of key maintenance strategies and elements shall be discussed as follows:-
Total Productive Maintenance

The prime objectives of TPM are to:

- Maximise overall plant and equipment effectiveness through the elimination or minimisation of the six big machine losses;
- create a sense of *ownership* by plant and equipment operators through a process of training and involvement; and
- promote continuous improvement through small group activities involving production, engineering and maintenance personnel.

Although each enterprise may have its own unique definition and vision for TPM, most approaches recognise the importance of measuring and improving overall plant and equipment effectiveness, and the need to address the root cause of failures and output losses through the elimination of chronic losses.

**Overall Plant and Equipment Effectiveness**

For plant and equipment to be effective, it must be able to run when required, at the right speed, and be capable of producing output to the specified quality. *Overall equipment effectiveness* measures these requirements by combining three key performance indicators *availability*, *performance rate*, and *quality rate* (see Fig. 2).

![Diagram of Overall equipment effectiveness](image)

**Fig. 2 – Overall equipment**

These three measures can be subdivided into breakdowns and set-up adjustments relating to availability, idling/minor stoppages and equipment speed relating to performance rate, and process defects and start-up losses relating to quality rate. These six measures are referred to as the six big losses (see Fig. 3).
The six big losses need to be investigated holistically. Attention to only one or two of these losses will produce a suboptimal result. For example, if availability is the only measure that is stressed, plant and equipment will often be run at a slower speed to make the measure look good, whilst the effectiveness of the plant and equipment is downgraded. Conversely, plant and equipment may be running at the right speed and be available but because of excessive wear is producing output that is out of specification.

When many organisations first measure overall equipment effectiveness, they find they are achieving in the order of 40 per cent to 60 per cent. International best practice is recognised to be 85 per cent.

Sporadic and Chronic Losses

The driving objective of TPM is to eliminate or minimise, the six big losses not just reduce them. To achieve this TPM is an ongoing journey to excellence which challenges the paradigms. One such important challenge is the traditional mindset that focuses on sporadic or breakdown losses and largely ignores the chronic losses which are the root cause of output defects and breakdowns.

Sporadic or catastrophic losses are the infrequent or unusual events that cause a sudden breakdown or loss of quality. They are obvious and the traditional solution is to create systems to react to them quickly and attempt to reduce them.

Chronic losses are subtle and not obvious. They are much more difficult to identify and correct because they are traditionally accepted as the norm. Chronic losses are caused by hidden defects in machinery, equipment and methods. They resist traditional remedies because their roots are hidden in the structure of the plant and equipment and the methods used to operate and maintain it.

Capital versus Repair and Maintenance Expenditure

The total life costs for a piece of equipment need to be analysed as the Repair and Maintenance portion can be significant as a ratio of the initial purchase price. Financial elements such as Depreciation, Investment and a value on Obsolescence are added to Maintenance and Operational costs to determine the Total Economic life. Cost drivers for maintenance need to be identified with the major impact costs being those that are focused on, and from these options for possible lowering of cost can be assessed. Outsourcing of rebuilds, changing the whole site strategy on maintenance from a repair on failure to ‘On Condition”, or determination of the operational influencers will help in the formulation of a work plan for

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### "Six big losses" Target

<table>
<thead>
<tr>
<th>Loss</th>
<th>Target</th>
</tr>
</thead>
<tbody>
<tr>
<td>Breakdown</td>
<td>Zero</td>
</tr>
<tr>
<td>Set-up and adjustment</td>
<td>Minimise</td>
</tr>
<tr>
<td>Reduced speed</td>
<td>Zero</td>
</tr>
<tr>
<td>Idling and minor stoppage</td>
<td>Zero</td>
</tr>
<tr>
<td>Defects and rework</td>
<td>Zero</td>
</tr>
<tr>
<td>Start-up</td>
<td>Minimise</td>
</tr>
</tbody>
</table>

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**Fig. 3 - The 'six big losses'**
improvement. All of this must be done with a clear link between costs and the productivity of the unit. Higher utilisation with the same base costs of maintenance will give the greatest return on asset. The key focus always must be the close management of all maintenance cost drivers.

Reduce Maintenance Activities

To reduce maintenance costs, maintenance activities need to be reduced. This can only be achieved by extending the life of equipment components and avoiding in service failures (which can also lead to subsequent damage to equipment). From a total cost perspective, the key drivers of maintenance costs are the mean time between repair/changeout (MTBR) or equipment life; the mean time between failure (MTBF) or equipment reliability; and the mean time of repair (MTTR) or equipment maintainability. By extending MTBR and MTBF and reducing MTTR, costs will be reduced and product quality, production availability and yield will be improved.

Three generic approaches can be taken to reduce maintenance costs, improve equipment reliability, and hence increase profitability; eliminate waste, plan to win and improve equipment (Fig. 4).

Eliminate Waste

This approach increases the efficiency of the resources used to perform maintenance activities - although without improving equipment availability and reliability. It includes project managing major maintenance activities (such as shutdowns), reducing shift crews, making the right contracting and sourcing decisions, eliminating restrictive work practices and standardising work requirements.

In many plants in Europe, the United States and Australia, McKinsey has found that shift crews have been larger than could be reasonably justified on economic grounds. McKinsey has also noted that substantial savings could be achieved by moving some tasks to day shift, allowing shift crews to be reduced. In many instances, shift crews have simply been too big - usually because operations managers are risk averse and have allowed maintenance departments to staff the shift crews so that they can cope with the worst case scenario, which rarely eventuates. In addition, shift crews are often not reduced when the breakdown rates has been decreased through a process of continuous improvement.

The trade-off with reducing shift crews is that when a major failure does occur, more time will be taken to carry out a repair because there are fewer mechanics and electricians in attendance. However, this occurs more rarely than expected, because operations and maintenance staff respond to small shift crews by improving their diagnostic systems, using operators on breakdown work, and instituting appropriate call-out arrangements for maintenance personnel.

Fig. 4 – Total cost versus Availability/ reliability

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Another major opportunity is to improve approaches to the sourcing of maintenance services. As maintenance becomes more effective, the urgency of work requirements is reduced and work is better defined. This opens up opportunities to improve the way products and services that support maintenance activities are procured. In McKinsey's experience, the costs of resourcing maintenance activities comes down by two to four-fold as maintenance breakdown work is eliminated.

In many companies, the biggest opportunity to capture these savings has been to improve contracting approaches. In metallurgical and mining companies the contracting and purchasing approaches have been inadequate, and have resulted in higher costs. Closer cooperation between maintenance, supply and contracting functions, and the use of more rigorous contracting approaches are often necessary prerequisites to capture the full benefits of controlled equipment performance that results from improved maintenance management.

Poor work practices and demarcation between trades, and even between work groups, also reduces the efficiency of executing maintenance activities. Over the last decade most demarcation problems have been eliminated, and those which remain are being negotiated away through enterprise bargaining, or through retraining and multi-skilling the work force.

Another major avenue to improve maintenance costs is to standardise and document work requirements and manage the work done in accordance with defined expectations. This need is not apparent when equipment fails regularly because maintenance personnel are experienced. They know the job and they know the equipment. Documentation would be superfluous. This is not, however a winning strategy. If the aim is to reduce the number of breakdowns to an insignificant level, companies cannot continue to rely on their experienced trades force over the longer term. By documenting and standardising work orders and managing the work so that it is completed in accordance with the work orders, companies develop a vehicle for learning how to do tasks in the best way, which enables them to control their costs, not just report them.

McKinsey has worked with many companies emerging from breakdown maintenance eras that have failed to document the tasks that they perform. They quickly encounter a skills barrier as the breakdown work reduces and the tradesmen's familiarity with the tasks declines. A key factor for success is documentation of task requirements; what to do, how to do it, with what spares and tools, by what labour group and the expected duration.

Improve The Equipment

This strategy invokes the philosophy of maintenance prevention, focussing on improving the equipment to make it more reliable and easier to maintain. The most cost-effective means of minimising the maintenance requirement is to build or purchase equipment that requires minimal maintenance, that is, equipment that is very reliable and available. For existing equipment this is clearly not always possible but, in the long term, management should ensure that all new equipment acquisitions are evaluated on a total cost of ownership approach. Maintenance personnel should be an integral part of the equipment acquisition team that is responsible for the specification and evaluation of new equipment.

In most mining and metallurgical situations, the equipment used is relatively unique, either because there are few replicas of the equipment in other installations and/or because the mine conditions, feedstock or process are unique. As a consequence, mining and metallurgical equipment should be viewed as being prototype equipment with ample opportunity to be improved. In addition, as the life of most capital equipment is in excess of 10 to 20 years, there is an opportunity to design out problems and introduce equipment improvements.

Manage Technology

The future will be more technologically complex due to demands of capacity increases driven by cost minimisation and availability of technological improvements to provide such. This complexity, whether it be equipment and system technological complexity or the management structure and systems of the business has to be satisfactorily managed.

It the case of equipment and systems, the matrix shown in Fig. 5 applies
Maintenance Resources - People

The Learning Environment

Learning is, for most, hard work. It takes concentration to succeed. But first, there must be a desire to learn, and a willingness to make the effort. In order to want to learn, most individuals need to understand what is expected of them, and why they ought to learn. Answers to these questions serve to place issues in context, and assist in determining whether to make a commitment to a learning activity, or not.

A noted British author on organisational communication, Francis (1987) notes that:

"People need to feel part of an overall strategy and feel they have some responsibility and involvement in decisions which affect them. No amount of table-thumping in the boardroom will achieve optimum performance from senior executives, middle managers or the shop floor if they do not understand the reasoning and accept some responsibility."

It is not enough to be told what to do and then be expected to learn how to do it. There is need to understand 'what is expected of us', how this links in with 'what is expected of others', and how associated activities interact. There is a need to know 'what the real issues are', 'why things need doing', 'what the consequences might be if they were not done', 'how our work impacts on the business', 'what standards need to be and why', and so on. For most people, answers to these questions are needed before commitment can be positively made to a learning process/programme and the demands it will make.

A leading author and contributor to the field of management thinking Send (1992), makes the point that:

"From a very early age, we are taught to break apart problems, to fragment the world. This apparently makes complex tasks and subjects more manageable, but we pay a hidden, enormous price. We can no longer see the consequences of our actions; we lose our intrinsic sense of
connection to a larger whole. When we try to "see the big picture", we try to reassemble the fragments in our minds, to list and organise all the pieces. But as physicist David Bohm says, the task is futile - similar to trying to reassemble the fragments of a broken mirror to see a true reflection. Thus, after a little while we give up trying to see the whole picture altogether."

It is difficult to imagine putting a jigsaw puzzle together without having an overall picture to work from. The confusion, frustration and disinterest that must result is obvious. Yet the tendency is to expect others to undertake activities without ensuring that they appreciate how these activities fit into the larger picture. What the larger issues are is not often explained. Is it any wonder that there is so often difficulty engendering enthusiasm and commitment?

As discussed above, it is not sufficient to focus simply on the proficiency with which one performs a task, no matter how critically important that task may be. People also need to appreciate the task's wider implications and how their performance plays a part in a larger picture.

The National Training Board (NTB) (1992) highlights the importance of having - and assessing - 'underpinning knowledge and understanding', but the knowledge and understanding referred to is solely associated with performance of action-based tasks. It states:-

"...it is not consistent ... with a competency based approach to have a unit or element defined solely by knowledge, without the context of what the knowledge is required for in its application to a work situation.....", and

"...Only that knowledge which is related to the required actual workplace performance outcomes of the particular unit or element should be included in (the standard)...."

This is a serious issue. A concert pianist's skills can never be developed by concentrating on the 'training up' of each hand separately and ensuring proficiency of the hands independently. Integration and appreciation of the larger musical picture is essential. A champion boxer, likewise, can never be developed by training up each arm and leg in isolation. Lack of systems and procedures for ensuring full appreciation of context may well be the 'Achilles Heel' of the NTB's competency based education and training (CBET) movement. Unless in possession of an adequate contextual framework, it is difficult to make commitment, take responsibility, communicate effectively, anticipate consequences and persuasively convince others that one is unlikely ever - as far as it is in one's control - to carry out one's responsibilities in an unsafe, incompetent or irresponsible manner. Yet these are the very assurances that are increasingly sought, and must increasingly be provided.

Our ability to succeed in business can only be based on the ability and commitment of our human resources. As leaders of our industry we must strive to develop the skills of our people. And we must strive to develop an environment in which they can perform.

Organisations, like organisms, must be able to respond and adapt to changes in their environment. Those that can't respond fast enough will not survive. If we are standing still, we are effectively going backwards as someone else is moving forward.

With the continuing development of new technology and the interdependencies of global markets, the only thing that organisations can be sure of is that change will continue, and that it will continue at an increasing pace?

One of the critical factors in adapting to change is the ability to learn. This is as true for organisations as it is for organisms. For an organisation to survive in an environment that is constantly changing, it must be constantly learning. Individuals within the organisation must be able to detect what it is that they need to learn and meet these needs quickly. Most organisations have recognised that in order to adapt to change, people need to acquire and use new skills and knowledge. The reaction to this need has been to provide more training. Each time a new requirement is uncovered, more training is provided. This is equivalent to feeding hungry people fish. One must always keep feeding them or they starve.

A better and more sustainable strategy is to teach them how to fish. That is, provide a learning strategy where people learn how to learn and become responsible for acquiring and passing on learning.

*COAL98 Conference Wollongong 18 - 20 February 1998*
Sharing The Wisdom

The same problem occurred three times in the last year. No-one bothered to pass on what caused the problem and how to fix it. We haven't learnt a thing! (Issie Frustrated, Engineering Manager).

Joe knows how to maintain the valves and Fred knows how to maintain the pumps. Why can't they learn from each other? (I.M. Keen, Maintenance Manager).

Both of these comments are symptoms of an organisation where individuals are not sharing their wisdom. This is not an uncommon situation. Too often, the knowledge and skills possessed by an individual remain the property of the individual. Individuals may in fact feel motivated to protect their wisdom as this makes them more valuable to the organisation. If they know something that no-one else does, this increases their job security.

An effective learning strategy motivates individuals to share their wisdom and helps the organisation to learn from past problems. In this way, the whole organisation continues to learn and is therefore better able to adapt to change.

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Using Fundamentals to Break the Breakdown Maintenance Cycle

A Saffa

INTRODUCTION

During 1991 - 1993 the author interviewed approximately thirty senior managers of varying responsibilities in either the engineering, construction or operation of a heavy industrial facility. The purpose of the exercise was to determine the factors driving a deterioration of the working relationship between the engineering and the operating company following practical completion of a major project. From the research a number of conclusions have been drawn as to the root cause of the deteriorating relationship. For this paper we are drawing on this initial research and our ongoing experience of solving problems of this nature to present some fundamentals that can be used to break the breakdown maintenance cycle.

Our research showed that approximately eighty percent of all new heavy industrial plants and manufacturing facilities fall into a one to two year revenue generation slump shortly after practical completion. When hundreds of millions of dollars are invested in new heavy industrial plant or manufacturing facilities the principals rightfully expect a reasonable return on the investment once the operation has started production. The reasons for the slump in revenue generation are numerous and varied and this paper is aimed at addressing some of the factors that can be easily corrected with today’s understanding of the forces involved.

The 1991 - 1993 research indicated that a lack of formalisation prior to practical completion was the main cause of a company falling into a revenue slump following practical completion of the new plant. On the operating company side of the equation this lack of formalisation is brought about due to ignorance of the importance formalisation has on revenue generation. With no understanding of the importance of formalisation to revenue generation there is a very strong tendency by the operating company not to spend any capital for formalisation during the pre practical completion investment stage of the project. This problem is so severe on the operating company’s side that most of the companies when building a new plant do not even budget for formalisation at all. With all good intentions the operating company’s management make incorrect decisions about the need for formalisation and/or the timing of the development of the required procedures to the detriment of the revenue generation of the project.

On the engineering side of the equation this lack of formalisation is brought about predominantly by the short term view of a design and construct mentality with little or no consideration of the long term operational requirements. In some cases the engineer is also ignorant of the importance formalisation has on revenue generation for the operating company and as such is not even aware of the effect of lack of formalisation. On the other end of the scale the engineer may know the importance, but chooses to exclude the work from the tender in order to be more competitive than their competition with the view that this portion of the project can be added as an extra at a later date.

On the positive side, the 1991 - 1993 research indicated that twenty percent of companies who have not experienced the post practical completion drop in revenue either stayed at the designated name plate data design rate or were able to operate at a production rate that was within the plant capabilities and at a value higher than name plate data thus generating a revenue higher than projected. In addition to the stable revenue, some of the companies evaluated experienced a lower than average number of early life failures following practical completion. The solution to staying out of breakdown maintenance following practical completion is to formalise the following areas before the plant start-up:

1 Paradigm Shifting Pty Ltd
During the 1991 - 1993 research, in addition to looking at “the factors driving the deterioration of the working relationship between the engineering and operating company” we also evaluated the factors contributing to the achievement of the optimum Maintenance/Production cost point. From our research of the problems on this level we have determined the following elements need to be addressed to achieve the optimum cost point for each plant.

Most companies are addressing a number of the key elements to some degree or another. The solution is to evaluate and systematically apply all of the elements as appropriate for each plant. This systematic application will move a company from a severe degree of breakdown maintenance to the optimum cost point and in the process generate the desired results of reduction of costs and improved quality and productivity of product.

Attached as Appendix is a fault tree that can be used to assist the personnel in an operating company evaluate and determine their present position, determine what areas need to be improved and then using internal manpower or with outside assistance take actions that will generate the desired improvements.
The profitability of a company, or the bottom line, is the core issue as to whether an organization is allowed to continue to exist, the employees continue to work and the facility has a future. As such, the first topic that needs to be discussed, when breaking the breakdown maintenance cycle, is the relationship between maintenance and loss of production costs. Once the relationship is understood the operating company can determine the relevance of the material.

The above cost graph is a reflection of the relationship between the cost of preventive maintenance, breakdown maintenance and the loss of production over the 1st, 2nd and 3rd generation maintenance philosophies.
Minimum maintenance cost point

Starting with a 1st generation logic of zero preventive maintenance the result will be that 100% of all maintenance will be breakdown (i.e. fix it when it breaks) resulting in a high loss of production. As preventive maintenance (2nd generation philosophy) is introduced and applied to the machinery, the cost of breakdown maintenance and the cost of loss of production decrease on parallel paths. These paths will be similar until the point of minimum maintenance cost is reached. At this point maintenance cost is minimised, but still results in a high degree of loss of production. This type of operation has a high degree of emphasis placed on reducing maintenance cost rather than looking at the bigger picture of maintenance cost per unit of production.

Optimum loss of production and maintenance

If additional preventive maintenance is applied beyond what is required to reach the minimum maintenance cost point the total cost of maintenance will start to increase, but the loss of production cost will continue to decrease. If this process is allowed to continue a point will be reached where the optimum cost of producing the product will be realised. This optimum cost point is the desirable point to be at - this is where the product is at the lowest cost/unit produced.

Exceeding the optimum cost point

In some industries such as space, nuclear and others dealing with explosive or dangerous materials it is prudent to continue to apply additional preventive maintenance beyond the optimum cost point. In this region of the graph the facility is no longer operating at the optimum cost point and in fact is expending additional resources to ensure that safety requirements and environmental constraints are met.

Determining the actual position in relation to optimum cost point

To determine where a facility is in relation to the optimum cost point it is necessary to observe the contribution of loss of production. If for example, loss of production is being caused by breakdowns the facility is being maintained to the left of the optimum cost point, in a 2nd and possibly, worst case, a 1st generation philosophy. On the other hand, if loss of production is being caused by over maintenance it is an indication the facility is being maintained to the right of the optimum cost point.

Obviously, the evaluation presupposes that the key point indicators of preventive maintenance (PM), breakdown maintenance (BM) and loss of production (LOP) are being tracked.

ENTERING THE WHIRLPOOL

The decisions made prior to practical completion for fixed plant and those made prior to site delivery for mobile plant are the ones that determine whether or not a company will realise the benefits of the capital investment or whether the plant or equipment will enter a breakdown maintenance mode. If the plant and equipment goes into breakdown maintenance the revenue generated from that plant and equipment will fall into a slump. In most cases this slump will last for one to two years. The purpose of this section is to show how a company gets swept into the whirlpool and to show what is required to ensure that the whirlpool is never entered.
1. Company Procedures

2. Standard Operating Procedures for Production

3. Training Program for both Maintenance and Production

4. Maintenance Program
   - 3 Stages Of Maintenance Program
     a. Prior to Practical Completion
     b. In Breakdown Maintenance
     c. A Mature Program
Return on Investment

Investors spend large sums of money on building new facilities with the view that they will generate a reasonable return on their investment based on the production rate of the plant, i.e., Name Plate Data. Our investigations show that 80% of these new facilities do not generate the desired revenue and in fact fall into a revenue slump shortly after achieving practical completion. This slump can easily last 1 to 2 years and in some cases has been as long as seven or eight years. The impact of this lost revenue is the curtailment and/or delay of other future projects identified in the company’s long term plans. The company’s efforts instead need to be focused on the new plant that is now in breakdown maintenance.

Misconceptions Leading to High Loss of Revenue

The following points are generalisations of some of the factors that have lead to a facility dropping into a revenue slump following practical completion:

Initial Scope of Work

When an operating company goes out to tender to build a new plant, in most cases, the requirement to assist in the development of company procedures, standing operating procedures, training for maintenance and production and the development of a maintenance program are not included as part of the scope of work.

Recognising the Need for Formalisation

In a number of cases the operating companies have a negative view towards procedures and formalisation of any type and instead take the view that operators are hired because they know how to operate, and the maintenance department can develop the maintenance program after practical completion. The expenditure of any money for procedures prior to practical completion is unnecessary and is just extra money going out. These views are totally wrong and they are the major cause of companies falling into the revenue slump.

New Equipment Will Run Until the Program is Developed

The view exists that new equipment will run with no problems; the new maintenance department will be under utilised with lots of spare time; we can hire all the trades and operators at the last minute and we will have time after practical completion to develop all of the maintenance program requirements. The problem with this view is that after practical completion a plant has a number of major early life failures and poor design problems to sort out. Also if you do not perform preventive maintenance a piece of equipment will fall straight into breakdown maintenance. The period of time following practical completion is one of the busiest periods of time for maintenance over the design life of the plant. They most assuredly do not have time to develop a maintenance program during this phase of plant life.

Staying Out of the Whirlpool

From our research the 20% of companies that did not fall into breakdown maintenance had the following in place prior to practical completion:

- Company Procedures
- Standard Operating Procedures
- Training (Maintenance and Production)
- Maintenance Program
SWIMMING AGAINST THE FLOW OF THE WHIRLPOOL

This section of the paper deals with breaking the breakdown maintenance cycle. How do we get out of the whirlpool once we have been sucked into the vortex. What do we do that is different from what we have been doing and still stay cost effective. To break the breakdown maintenance cycle and to obtain the goal of “Optimum Cost Point” the following key elements need to be put in place. The order is not especially crucial and in most cases an organisation will have facets of each key element in place and may be working on all seven simultaneously.

Fig. 4 - Maintenance / Production cost relationship

Loss of production, and maintenance costs, decrease as you approach an ideal maintenance program

Baseline maintenance

Baseline maintenance is the ground level and starting point of an effective maintenance program. This level of the maintenance program addresses documentation, skilled labour, parts and materials and special tools. Plant and equipment will fail and some of these failures will be unexpected. It is the development of baseline capabilities that determines how
effective the organisation is in dealing with these unexpected failures. Some of the consistent weak points observed during plant audits are as follows:

- Poor care and control of insurance spares;
- Poor identification, care and control of special tools;
- Purchase of alternate spares to save a few dollars;
- Access to parts and materials during back shifts; and
- Poor technical library and few “As Built” drawings;

Maintenance infrastructure

Maintenance infrastructure deals with the way maintenance is organised to carry out maintenance. This level of the maintenance program addresses organisational structure, the responsibility, authority, accountability and ownership factors and the work process system. Some of the consistent weak points observed during plant audits are as follows:

- Organisational structures are put in place, but are seldom an accurate reflection of the field practice;
- There is an imbalance between the assigned responsibility and authority, but the individuals are still held accountable for non-performance, generating negative feelings towards ownership and company loyalty;
- Work process systems are ad hoc or not in place at all
- Companies are in a high degree of breakdown maintenance requiring the maintenance supervisor to perform supervision while still expecting them to carry out the planning functions;
- Breakdown crews are formed and empowered in such a manner that they become self fulfilling and continue the breakdown mentality; and
- Management decides to control costs by reduction of labour in the maintenance workforce making it difficult or impossible to increase preventive maintenance to realise the desired results.

Preventive maintenance

This key element is the most cost effective and most crucial in that it has a direct impact on the length of time that a piece of equipment operates. Lengthening the time of equipment operation results in higher production. This level of the maintenance program addresses the formalisation of evaluation, planning, organising, scheduling and executing those activities that will prevent failures of plant and equipment. Some of the consistent weak points observed during plant audits are as follows:

- Disregard of the basics of maintenance, clean, inspect, lubricate and carry out minor adjustments
- Ad hoc approach with no formalised maintenance procedures,
- Adverse attitude of management, supervision and trades towards formalised procedures,
- Management attitude that if you don’t have a wrench in your hand you are not working,
- General lack of communication,
• The feedback loop not being closed, and
• At this point the flow and location of parts and materials is often still a problem.

Predictive maintenance

Predictive Maintenance is the second most cost effective activity in the maintenance program. Once the maximum lifespan of a piece of equipment can be achieved through the application of preventive maintenance activities, the organisation needs to know the possibility and timing of potential failures through the application of predictive maintenance techniques. This key element deals with the application of on-condition monitoring, the collection and analysis of the data and the prediction of the failures based on the data. Some of the consistent weak points observed during plant audits are as follows:

• Pieces of equipment are allowed to run to failure without operators communicating the potential failure to maintenance;
• Pre-start checklists are nonexistent or disregarded;
• Poor shutdown planning with a large number of additional items added at the last minute;
• Resistance to utilising contract labour; and
• A view that on-condition monitoring is high tech and very costly;

Company infrastructure

Company infrastructure addresses the way the company is organised to support the maintenance activities. This key element deals with the upper tier infrastructure factors and is typically not recognised as having an impact on maintenance. Some of the consistent weak points observed during plant audits are as follows:

• Poor recognition by management that the product of maintenance is “equipment availability” and therefore the foundation of high productivity;
• Promotion of the importance of production to the detriment of maintenance
• Accounts clearing the warehouse of all slow moving parts and materials without consulting maintenance for a proper evaluation of their requirements;
• Selection of a computerised maintenance system that is good for accounts but poor for maintenance;
• Shift competition and bonus schemes encourage one shift to run equipment to total failure to benefit the shift, but to the detriment of the company; and
• Engineering is an empire unto itself, allowed to design, procure and build without consultation with production and maintenance;

Quality assurance

Quality assurance addresses the formalisation of the entire organisation in its desire to carry out repeatable activities that can be improved in a systematic, step by step process to generate a high quality product at a reasonable cost. Some of the consistent weak points observed during plant audits are as follows:

• No quality assurance program exists at all;
• The company did not know the direct cost of poor quality prior to implementing quality assurance;

• The company tries to recover the cost of quality assurance application through the increase of the product selling price, rather than the reduction of direct cost of poor quality or the resultant increase in sales;

• The introduction of quality assurance was too far removed from the shop floor, ownership does not exist and the documents are now dust collectors; and

• Quality assurance was adopted to meet some outside requirement rather than seen as an internal mechanism that could generate systematic improvements;

Reliability centred maintenance

Reliability Centred Maintenance (RCM) is the most recent addition to the series of key elements that can be used to assist in the attainment of the optimum cost point. The RCM approach deals with a systematic analysis of plant and equipment to determine the corrective (breakdown), preventive and predictive maintenance required to ensure the reliability of the plant and equipment in the most cost effective manner. Some of the consistent weak points observed during plant audits are as follows:

• Most organisations must first improve the sophistication of their existing 1" and 2" generation programs, before considering RCM;

• Companies have backed away or withdrawn from the RCM analysis techniques due to the perceived cost of implementation;

• Companies have fallen into breakdown maintenance because of the view held during implementing "that the overall outcome was a reduction in maintenance so lets stop performing preventive maintenance now and spend our extra time on RCM";

• Most companies have not conducted any type of formalised analysis to determine the requirements to ensure reliability;

• Most companies do not capture root cause of failure data via the feedback loop; and

• Most companies do not track nor analyse the cause of early life failures.

SUMMARY

In summary this paper has been designed with the idea that operating companies can be improved by concentrating on the following areas:

• Making the correct decisions regarding the need for formalisation prior to practical completion;

• Realising that to achieve the optimum cost point management may be required to allocate additional funds to the maintenance department and trying to achieve minimum maintenance cost may be counter productive;

• Utilising the "self analysis check sheet" and applying to the following to areas:
  • Audits,
  • Training,
• Formalisation

New Greenfield site

When building a new plant the problem of only 20% of companies achieving the desired results where 80% of all new projects fall into a revenue slump within 3-6 months of practical completion and the fact that the slump will last for one to two years must be addressed. This is a mismanagement problem where the operating company - in most cases - does not understand the need for formalisation and training prior to practical completion. The engineering contractor further contributes to the problem by not addressing the negative impact that little or no input into design, from maintenance and operations will have on the revenue generation following practical completion. To solve these problems operating companies must provide additional funds at the start of a project to formalise maintenance and operations prior to practical completion. If this view is embraced, projects will become more profitable, the risk of building new projects will be lowered and the investors will be more inclined to further invest in additional new projects.

Success factors of the 20%

The following four success factors were present in the companies that built new projects, and then, following practical completion, did not fall into a revenue slump. The development of the maintenance program was predominantly the most important of the four.

Company procedures

Sorting out the company procedures prior to practical completion had the effect of setting the infrastructure in place before it was required. During a relatively non-hectic time, individuals were able to develop and formalise a whole series of procedures addressing day to day requirements. This meant that when the frenzy of activities required to commission and reach practical completion was taking place, the company procedures were in place providing support to the process, rather than being developed at the same time under pressure during the commissioning program.

Standard operating procedures

Standard operating procedures specifies how you want the operators to run the plant. If the plant does not have a standard for operation, each operator will make decisions and adjustments to the process based on how they personally feel it should be operated. Invariably at the end of every shift the plant and process will be stable, producing a quality product but when the new shift starts the control set points are incorrect, from their point of view, and so they will change the process. In some cases this tendency has been so severe that the first six hours of each twelve hour shift produced out of spec product.

Other problems that can be present are incorrect operations, non transferable activities, different valve line ups from shift to shift and lack of consistent application of safety, environment and hazard considerations.

Training for maintenance and production

Specialised training must be provided for maintenance and production. The training should be skills based for the specific role each individual is to fulfil and should also include some upper level cross training between the two departments. This cross training will ensure that maintenance and production personnel have a greater appreciation of each others role and will then be able to consider the larger impact of the decisions they make on a day to day basis.
Maintenance program

The most important requirement is to put in place an effective maintenance program. If this is not done the plant will fall into breakdown maintenance.

This is a manhour intensive exercise. To develop a preventive maintenance program for 500 pieces of equipment will require about 1500 individual preventive maintenance activities and it will require approximately 3000 manhours of labour to complete the work. The two most cost effective things to put in place are the preventive maintenance basics (clean, inspect, lube and minor adjustments) and the on-condition monitoring activities that can generate the data that once analysed, can be used to predict potential failures.

Starting with a 1st generation logic of zero preventive maintenance the result will be that 100% of all maintenance will be breakdown (ie. fix it when it breaks) resulting in a high loss of production. As preventive maintenance (2nd generation philosophy) is introduced and applied to the machinery, the cost of breakdown maintenance and the cost of loss of production decrease on parallel paths. These paths will be similar until the point of minimum maintenance cost is reached. At this point maintenance cost is minimised, but still results in a high degree of loss of production. This type of operation has a high degree of emphasis placed on reducing maintenance cost rather than looking at the bigger picture of maintenance cost per unit of production.

If additional preventive maintenance is applied beyond what is required to reach the minimum maintenance cost point the total cost of maintenance will start to increase, but the loss of production cost will continue to decrease. If this process is allowed to continue a point will be reached where the optimum cost of producing the product will be realised. This optimum cost point is the desirable point to be at - this is where the product is at the lowest cost/unit produced.

MAINTENANCE CAPABILITY SELF-EVALUATION CHECKLIST

Maintenance evaluation at a glance

The Fault Tree gives you a brilliant snapshot of all the steps required to take any plant from Baseline Maintenance, through Maintenance Infrastructure, Preventive Maintenance, Predictive Maintenance, Company Infrastructure, Quality Assurance, and Reliability Centred Maintenance to an Ideal Maintenance Program.

The fault tree is both a diagnostic tool and a success roadmap

The Fault Tree allows you to set realistic maintenance improvement targets that are logical, manageable and achievable. Say you work through the fault tree and evaluate your plant as operating at the level of Preventive Maintenance. Then Predictive Maintenance would be your next goal. We have found that companies achieve significantly better results when they proceed methodically and logically, one step at a time.

How it works

Start with Baseline Maintenance. Read the criteria and then answer YES or NO. If your maintenance costs are greater than 20% a year, you’ll answer YES and go to the key point indicators box, immediately below. If you answer YES to the Key Point Indicators, you’ll go down to the Actions Required box which tells you what you need to do to remedy the problems identified above.

However if you answer NO to the question in the Baseline Maintenance, you’ll move across to Baseline Infrastructure. When you answer YES, you’re moving DOWN. When you answer NO, you’re moving across to your right. Eventually you will arrive at your current level of maintenance capability and what needs to happen next to make the improvements needed.
APPENDIX

1.0 Baseline Maintenance

Is less than 20% of the maintenance department's resources (labour, tools, parts, and materials) expended on unplanned failures of plant and equipment?

No > 20% No

Do some or most of the following key point indicators exist?

• > 20% Breakdown maintenance
• Equipment has failed and is waiting for parts and materials
• Equipment runs, but, does not meet performance criteria
• The trades do not know what they will be working until they arrive at work
• Work packages are reactive to plant and equipment condition

Yes

2.0 Maintenance Infrastructure

Does the way the maintenance department is organised have a positive effect on the trades ability to carry out maintenance and/or is the work order system formalised into a systematic method of completing the required work?

<20% No

Do some or most of the key point indicators exist?

• A barrier exists between mechanical, electrical, and instrumentation
• Very little consistency between one maintenance supervisor and the next
• Maintenance infrastructure changes every time a new manager joins the organisation
• No formalised work order system exists
• The feedback loop is not closed

Yes

Paradigm Shifting Solutions

Consulting

• Conduct Maintenance Audit
• Develop and revise Vision & Goals
• Develop Maintenance Improvement project p/w a work order system
• Present Plan to Site Management

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3.0 Preventive Maintenance

Does the maintenance department personnel systematically carry out the basic maintenance tasks such as cleaning, inspecting, lubrication and minor adjustment that are required to ensure that plant and equipment reaches its maximum expected lifespan?

No

Do some or most of the following key point indicators exist?
- Equipment fails due to dirt ingress, lack of lubrication and/or lack of minor adjustment
- Preventive maintenance takes a back seat to breakdown maintenance
- Production do not understand the importance of preventive maintenance
- Production do not assist in carrying out any maintenance

Yes

Yes

4.0 Predictive Maintenance

Do the maintenance department and production department personnel systematically carry out the on-condition monitoring tasks that are required to determine condition of plant and equipment while in operation and once the data has been collected do they use the data to predict condition and dates of potential failures.

No

Do some or most of the following key point indicators exist?
- Production run equipment to failure
- Little or no preplanning takes place
- A lot of process data is collected and filed away but, never analysed
- The data collected in the maintenance management system is a continuation of the old manual system data
- History is being collected, but the feedback loop is not closed by analysing the data.

Yes

Technical Writing

- Provide technical writers and/or a site technical writing coordinator to assist in developing the required procedures. Note: Some companies want their own trades to write the procedures so conduct One Day Procedure Writers Course.
5.0 Company Infrastructure

Does the way the company is organised have a positive effect on the interworking relationship between m and production and do the senior and middle management view the m of the facility to be the foundation of achieving high levels of productivity?

No

Do some or most of the following key point indicators exist?
- A barrier exists between production and maintenance
- Production has higher priority than maintenance
- Planning meetings are not effective and production change their minds after committing to a plan of action
- Procurement procure parts that are cheaper, but have a major impact on the mean time between failure

Yes

6.0 Quality Assurance

Does the company have a formalised quality assurance program consisting of policies and procedures covering all department activities that are necessary to produce a high quality product and more importantly does the system work?

No

Do some or most of the following key point indicators exist?
- A series of company procedures exist, but basically take up shelf space and collect dust
- A third party developed the procedures in isolation of the people on the shop floor
- The procedures have never been accepted by the shop floor personnel
- The system was only developed to satisfy the client's and management.

Yes

Consulting
- Conduct Audit
- Revise Vision & Goals
- Revise Maintenance Improvement Project
- Plan to Management
7.0 Reliability Maintenance

**Ideal Maintenance Program**

- The amount of maintenance resources that goes into unplanned failures is < 10% and is approaching 5%
- An excellent on-condition monitoring program exists and most of the maintenance department personnel are involved in activities that maximise mean time between failure and minimise mean time to repair.
- Production and maintenance personnel view the work process as a team effort and all activities are aimed at everyone performing the multi skilled activities that are required to produce a cost effective high quality product.
- Contractors are used properly and viewed as part of the team effort
- An excellent relationship exists between engineering, maintenance and production and all the parties are engaged in the identification and solving problems through a formalised redesign or design out process
- Management are totally behind the improvement process, have delegated the required responsibility and authority and actively support the team efforts.

**Does the company have a formalised reliability maintenance program of formally and systematically analysing equipment and processes to determine how to generate step by step improvements to improve quality and reliability?**

**Yes**

- The amount of maintenance resources that goes into unplanned failures is < 10% and is approaching 5%
- An excellent on-condition monitoring program exists and most of the maintenance department personnel are involved in activities that maximise mean time between failure and minimise mean time to repair.

**No**

- **Do some or most of the following key point indicators exist?**
  - The plant and equipment have exceeded their design life and need to be replaced
  - The trades when closing out a work order do not indicate root cause of failure
  - Some equipment needs to be replaced but insufficient data exists to justify the capital investment
  - Numerous process bottlenecks exist

**Do some or most of the following key point indicators exist?**

- The plant and equipment have exceeded their design life and need to be replaced
- The trades when closing out a work order do not indicate root cause of failure
- Some equipment needs to be replaced but insufficient data exists to justify the capital investment
- Numerous process bottlenecks exist

**Yes**

- **Have you been instrumental in the application of steps 1.0 through 7.0 to generate the improvements?**

**No**

- **Please contact Paradigm Shifting Managing Director for possible job vacancies**

**Technical Writing**

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The use of seismic methods for the detection of dykes

B J Evans¹, M Urosevic¹ and J Cocker¹

ABSTRACT

Seismic methods have become common for the detection of low-throw faults ahead of underground coal mining. Surface seismic methods cannot theoretically be used where dykes occur, because seismic waves transmit from the surface down to the seams, and reflect back to the surface. Consequently, where sub-vertical structure such as dykes occurs, the surface seismic method fails.

The ability of seismic methods to image dykes depends on the geometry used, the dyke thickness and the seismic wave propagation mode in relation to dyke composition and internal structure. Surface seismic methods find it difficult to distinguish between faults/fractures and very thin dykes (1-2m in thickness) when the dyke’s thickness is less than the seismic wavelength. Consequently, borehole seismic methods have to be used to detect the presence of such thin dykes.

This paper presents the first results from an ACARP project, which in part is a breakthrough in seismic technology for the detection of dykes. It explains how surface seismic methods were used to detect a thick dyke and associated faulting. An alternative approach, that of going downhole with seismic sources and receivers (borehole seismic profiling), shows that dyke sides can be imaged at depth, and that in future, it should be possible to produce an image of both sides of a dyke, in its correct orientation, using existing boreholes.

INTRODUCTION

The conventional geophysical approach to the detection of dykes is by sensing their induced and/or remanent magnetism. If a dyke is magnetic, then it may be detected by ground or aeromagnetics and mapped. However, when dykes have no magnetic signature, their presence goes undetected and can thereafter cause extreme interruption to underground mining. This is often the case.

The surface seismic method has been developed since the early 1950s to image sedimentary layering. Seismic energy is exploded at the surface using a charge of explosive, and the seismic-wave travels downward, reflecting off seam tops to return back to the surface, where it causes ground vibrations. These vibrations are felt by geophones placed along a line at the surface, which act like microphones when connected to a recording instrument (Evans, 1996).

Seismic reflections occur best when the downgoing seismic wave is reflected from horizontal seams, and when there is a termination of a seam with a change in seam depth, then a fault is inferred (Fig 1).

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Seismic section

**Fig. 1 - A fault is a termination in the seam image**

If dykes or vertical structure have intruded through the seams, the dyke's thickness becomes important. If a dyke is relatively thick (greater than 20 m), it may be similar to the wavelength of the seismic wave passing through it and affect the seismic wave propagation by diffracting it. If it is shorter than the wavelength, the seismic wave may be unaffected by the dyke and pass through it without any interference. In this latter case, the dyke is undetected. In the wide-dyke case, an image of the dyke is produced where the lines of the seam reflections break (Fig 2). Unfortunately, such an image has been rare to find.

**Fig. 2 - A dyke is a break in seam reflections**

An alternative seismic solution may be to go down a borehole with a seismic source and receivers effectively tilting the earth on its side, so that reflections can be received from the sides of the dyke (Fig 3). This would provide an image of a dyke at depth, and potentially map the dyke's sides at depth.
SURFACE SEISMIC METHODS

The surface seismic method applied in mining uses a shot fired at the surface and geophones also at the surface detecting the reflected energy. The geophones also respond to waves travelling in any other direction, so waves that travel near to the surface will pass horizontally through the top of a dyke.

Seismic reflections from coal seams depend on there being a velocity and/or density contrast between the top of the seam and the overburden. Where dykes exist at depth, any seismic waves which reflect from their hard-rock sides reflect downwards and away from the surface geophones. Otherwise, the seismic energy passes through them, being modified by the dyke in proportion to the thickness, velocity and quality of the dyke rock.

Compared with the hard-rock nature of dykes at depth (with a velocity around 5000 m/s), the weathered tops of dykes can often be very soft and similar to clay in texture. Such material often has a slower velocity (around 1800 m/s) than the adjacent sedimentary rock (around 2500 m/s), while the weathered dyke-top may act like a sponge to the seismic wave, attenuating the wave’s amplitude and reducing its velocity as it passes through. Consequently, it is possible that dykes could be observable on field records as waves arriving late (slower velocity giving a ‘statics’ anomaly), and being weaker in the vicinity of the dyke (attenuation). This knowledge may be used as an alternative key to magnetics for the detection of dykes.

Figure 4 shows two conventional field records of seismic data. The records show 96 seismic ‘wiggle’ traces which are moving positive or negative over a period of time, dependent upon the amount of seismic (and therefore electrical response) energy they experience. The first arriving seismic wave is either a ‘refracted’ or ‘direct’ wave which travels from the shot location to each receiver. In Fig 4a), the refracted wave is the first wave to arrive along the receiver line. In this figure, the first arrival is weaker amongst a group of receiver stations. Figure 4b) shows a normal record, and how the first arrival would appear with no dyke present. In the case of Fig 4a), we know that the geophones were located precisely across the dyke, so this is the effect of a large dyke some 40 m wide.
If a shot is fired in a deep borehole, say at 180 m depth some 155 m from this 40 m wide dyke, the direct waves coming from the explosive to the surface receivers will pass through the weathered dyke as shown in Fig 3 and again affect the wave's amplitude detected by the geophones. The result is shown in Fig 5, where clearly there is a 'notch' in the first arriving seismic wave as it passes across the geophone stations. Such responses are observed frequently both in the field and during seismic data processing, but ignored as readily as they are observed because their use is not understood.

A seismic line was recorded for Powercoal at their Morisset lease over a dyke, and the data carefully processed to retain all of the dyke's character. This produced a seismic section (Fig 6a) which contains all of the typical coal sedimentology hallmarks of faulting and seam rolls. Note the subtle variations in lithological response across the section which properly acquired and processed seismic data can produce. Faults are observed to be almost vertical, while short vertical faults are probably linked by seam floor or roof-stress transfer breaks.

However in this case for the first time, we have captured a profile of the dyke passing upwards through the coal section, probably intruding along a fault or plane of weakness.
Fig. 5 - Shots at 190 m and 180 m depth, showing effect of a dyke

The interpreted section is shown in Fig 6b) where the dyke is observed to be as narrow as 30 m at 400 m depth, sinuous in shape, and broadening in champagne-glass form within the weathering. At depth, the dyke's intrusion has certainly caused a buckling of sediments at 700 m. The dyke passes up through the Fassifern seam at 200 m with a width of some 35 m, and forewarning of this dyke's presence is a clear requirement prior to mining the seam. In this case, a recommendation not to mine here would be expected.

Ground magnetics had put the dyke's width at 38 m. The seismic data indicates that the dyke is not straight sided, and varies in width between 30 and 45 m. Both data sets were within metres of each other at the surface and this is considered to be an excellent correlation between two different geophysical methods.
BOREHOLE SEISMIC METHODS

An alternative to surface seismic methods is that of borehole seismic, in which both sources and receivers are positioned down a borehole, and the source is fired in similar manner to the method of surface seismic. This method has not been tested to our knowledge in mining previously, because it is considered that firing an explosive beneath a string of receivers is asking for trouble.

During a seismic test program at BHP Coal’s Goonyella site, a 12 channel hydrophone cable was lowered down a borehole which was some 150 m from the centre of a suspected dyke (indicated from aeromagnetics). The Goonyella Middle seam was at a depth of 180 m in this case. Small explosive charges (25 g) were fired up the borehole coming no closer than
20m to the cable. The cable was raised step by step as a number of charges were fired, until 90 m when it was accepted that the hydrophones were losing their sensitivity. The arriving pressure wave after each shot was too strong to bear for the nearest hydrophone which finally lost all sensitivity and the test was abandoned.

The seismic data were then processed using conventional CMP stacking to produce the section of Fig 7a). Depth conversion was performed using velocities observed from other seismic data so that in this section, distances are shown down the side and across the top, with the borehole being located down the left-hand side. From this figure it can be seen that the section is an image of geology between 105 and 185 m vertically, and up to 300 m laterally away from the borehole.

Figure 7b) is the interpretation of the same data, in which the closest face of the dyke to the borehole would be the black-peak to the left of the interpreted line. The interpreted line in the white-trough could be the other side of the dyke, which would fit with drilling indications of dykes no more than 5 m wide in this area. A number of reflection surfaces are also clearly apparent, the nearest one at 135 m, a second about 160 m away from the borehole and another some 220 m away. If the reflection at 135 m is the nearest side of a dyke, the image details how a dyke's surface appears in greater detail than in the sedimentary section of Fig 6b), albeit only for a small distance in depth.

However, this is the first observed image of a dyke's form at depth, and with modifications to equipment, such an approach could be developed into a standard logging tool at some stage in the future.
CONCLUSIONS

Two seismic methods have been presented for imaging dykes at depth.

The surface seismic method has shown that dykes can be detected in field records if the field operator or data processor has the experience to understand its affect on first arrivals. The surface method has also shown that if the data is processed with care, an image of the dyke can be obtained to great depths - in this case some 700 m - and its form provides a clue to its vertical movement up and through the coal seams.

The downhole seismic method has shown that reflections can be obtained from dykes away to one side of any chosen borehole. In this case, a string of home made hydrophones were used which would position the dyke in any direction. If the hydrophones were replaced with three component wall-locking geophones, it should be possible to obtain the precise location of the dyke and its three-dimensional shape in space as an aid to mining operations.

ACKNOWLEDGMENTS

The authors wish to acknowledge the support of ACARP Project No. C6041, and the release of data owned by Powercoal Ltd and BHP Coal Pty Ltd. The Powercoal surface data were recorded at their Morisset lease in NSW, while the borehole data were recorded at BHP Coal’s Goonyella mine site in Queensland.

REFERENCE

Coal Seam Modelling and Mine Planning Using Results of a 3D Seismic Reflection Survey – An Example from Huntly Coalfield, New Zealand

D A Fergusson

ABSTRACT

Geological hazards such as faulting, basement ridges, and zones of “thin” (< 6m) coal have a major impact on mining economics of underground operations at Huntly Coalfield. Experience has shown that drillhole-based investigations do not yield sufficiently detailed models of the coal seam to allow management of the planning risks associated with these hazards. Consequently, operational performance is affected by unplanned costs associated with lower productivity, loss of coal reserves, and expensive strata control remedies in problematic ground conditions. Huntly Coalfield is a challenging environment for acquisition of good quality seismic reflection data due to the very thick (10-85m) weathering layer. Since 1994, successful acquisition of 2D and 3D data has been achieved through a combination of careful testing of technical parameters and experimental trials prior to committing to production recording. High resolution seismic reflection (HRSR) is proving to be an investigations technique which results in accurate and reliable models of the coal seam and associated structures such as normal faults and paleo-topography of the surface on which the coal seam rests. The HRSR technique has been applied and developed in the Okowhao Sector where Huntly East Mine is developing and extracting coal reserves. Results of a recent 3D survey demonstrate that this technique is capable of revolutionising risk management in mine planning for underground mines in structurally complicated coal deposits.

INTRODUCTION

In the 1980’s high resolution seismic reflection was utilised as an investigation technique in the Waikato coalfields (Gumbley 1988). The results from the Huntly Coalfield were disappointing, largely because the acquisition was undertaken without “tuning-in” of technical parameters (Fergusson 1997). Evidence from the petroleum industry generally, and some Australian and US coal mining companies indicates that seismic reflection delivers major benefits for geological interpretation and hazard mapping (Davies 1992; Gochioco 1990, 1991; Harman 1981; Lamb, Saunders and Sweeney 1992; Lambourne, Evans and Hatherly 1989; Lambourne, Hatherly and Evans 1991; Nestvold 1992; Palmer 1987; Tilbury and Bush 1991; Urosevic, Evans and Hatherly 1992). This technique is especially relevant to the Huntly Coalfield where the thick Kupakupa coal seam is affected by numerous, large-scale geological hazards which defy definition by drilling. Undetected, these hazards have unplanned economic impacts on the operation related to productivity, development efficiency, reserves recovery and safety.

In January 1994 Solid Energy began research and development of 2D high resolution seismic reflection (HRSR) techniques at Huntly East Mine with the acquisition of a 900m experimental 2D line. The purpose of this work was to determine whether HRSR could reliably detect geological hazards for mine planning purposes. The results were very encouraging. A few months later, an additional 5 km of 2D data were acquired. The limitations of 2D seismic for hazard definition were soon realised after development of South 4 Panel. A 3D HRSR survey was undertaken over the Ralph Block in 1996-97 (Fig. 1). The implications for mine planning using the 2D and 3D seismic data are discussed using recent examples from Huntly East Mine.

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Solid Energy North, Huntly, New Zealand

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Huntly Coalfield

Coal resources in Huntly Coalfield occur in two economic seams near the base of the Waikato Coal Measures in the Te Kuiti Group (Eocene-Oligocene). The seams rest close to or directly on Mesozoic greywacke ("basement"). West of the Waikato River, in the Okowhao Sector of Huntly Coalfield, the Renown and Kupakupa coal seams merge to a single seam with an average thickness of 16 metres (Fergusson 1994a). Geological controls on seam geometry include a combination of structural and depositional factors such as seam splitting and merging, deposition over an undulating basement surface, wash-outs from an adjacent paleo-fluvial system, and structural thinning along normal faults.

Around Huntly, Te Kuiti Group has a regional dip of 5-10° N-NW. Locally, coal seam dips may be as high as 40° adjacent to faults and above basement ridges (Fowke 1987, Fergusson 1994a). Exposures in underground mine developments have revealed a fault system consisting of persistent NNW trending faults with throws of 10-50m, and NE trending faults with throws of 5-25m. In the broader setting, the southern end of Huntly Coalfield is situated between the tips of two opposed-dipping N-S fault systems, the Maungaroa-Kimihia and Waipa-Wilton/Karaka faults (Fig. 2). These faults are separated by a 10-15 km NE-SW step in the Huntly area. This step is marked by the Hakirimata Range in the Taupiri-Huntly region. Faulting close to this step is thought to be more intense and complex than faulting further north and south (A Nicol pers. comm.).

Underground mining

Present-day coal mining in Huntly Coalfield occurs at the up-dip southern and southeastern end where the coal seams are less than 350 m deep (Fig. 2). Two underground mines produce coal, Huntly East Mine (450,000 tonnes per annum) and Huntly West Mine (20,000 tonnes per annum). Since 1992, coal investigations have been concentrated in the "Western
Sector" of the Huntly East Coal Mining Licence (CML) area. The Western Sector constitutes the CML west of the Waikato River, and is within the Okowhao Sector of Huntly Coalfield (Fig. 1).

Huntly East Mine is a thick seam mining operation. The coal seam attains a maximum thickness of 24 metres and is a very weak to weak rock (UCS = 5-15 Mpg), yet it is stronger and more durable than the enclosing mudrocks (“fireclays”) of the Waikato Coal Measures (Mills 1986, Tan and St George 1989). Development tunnels are planned with two to three metres of roof coal to help ensure roof stability. The minimum practical workable coal seam thickness for underground extraction is six metres (i.e. 2-3 metres roof coal; 2-3 metres tunnel height; 1 metre floor coal). Coal is currently won by bord and pillar and bottom-coal extraction using continuous miners.

Geological hazards and mining risk

Reliable seam geometry and structural models are essential for effective mine planning if mining risk is to be managed. At Huntly Coalfield, mine economic performance is more related to geological hazards than to any other factor. Areas where the coal seam is thinner than 6m pose the greatest risk for mine planning. There are several types of geological hazard which cause adverse mining conditions and therefore impact on the productivity, efficiency, recovery and safety of the mining operation (Table 1).

Limitations of drilling for mine planning

Historically, coal seams in Huntly Coalfield have been investigated using surface drilling methods. Outcrop and natural exposures of Te Kuiti Group are very rare because the coal-bearing strata are concealed below a very thick (up to 85 m) sequence of Quaternary sediments and volcanic ash deposits. Drilling programmes have been undertaken periodically since the early 1900's, consequently, a mixture of wash-drilled to fully cored holes exist in the drillhole database, with touch-cored holes being the most common type. Wash-drilled and touch-cored holes yield very little structural data.

Fig.2 - Regional geological setting of Huntly Coalfield showing basement distribution and major structural features (modified from Edbrook, Sykes and Pocknall 1994)

Geological modeling of drillhole data with spacings as low as 100 m, such as in the Huntly East Mine east of the Waikato River (Huntly East Sector), was not able to resolve the seam structure (Fergusson 1988). Although the mine was planned for longwall mining, extraction using this method would have been extremely difficult and almost certainly unproductive,
due to the frequency, size and number of sets of faults. The longwall was not installed and the longwall panels were extracted using a Wongawilli type of mining system during 1987-1991.

### Table 1- Geological hazards in the Huntly Coalfield

<table>
<thead>
<tr>
<th>Geological Feature</th>
<th>Nature of Hazard</th>
<th>Impact on Operation</th>
</tr>
</thead>
</table>
| Normal faults      | • localised poor coal mass conditions  
|                    | • water seeps     
|                    | • steep grades due to drag folding  
|                    | • very weak       
|                    | • low durability coal measures exposed | • closer supervision required to negotiate fault  
|                    |                  | • unplanned stone drivage  
|                    |                  | • on-going tunnel instability and maintenance  
|                    |                  | • higher reinforcement and/or support costs (cable bolts, steel sets)  
|                    |                  | • pumping to handle water make  
|                    |                  | • loss of reserves and/or lower productivity  
| Thin coal (less than 6 metres) | • insufficient roof or floor coal  
|                    | • greater vulnerability to effects of faulting | • roof bolts ineffective if anchored in fireclay  
| Basement ridges    | • thin disturbed coal - flexural shears and tension fractures  
|                    | • stress concentration  
|                    | • steep grades | • lower productivity and/or loss of reserves  
|                    |                  | • higher reinforcement or support costs  
|                    |                  | • on-going tunnel instability and maintenance  
|                    |                  | • unplanned stone drivage through seam floor  
| Fault troughs     | • poor quality coal mass  
|                    | • intensified minor faulting and shearing | • lower productivity  
|                    |                  | • increased reinforcement and support costs  
|                    |                  | • lower recovery  

In 1992-93, a drilling programme consisting of 39 touch-cored drillholes was undertaken, reducing the drillhole spacing from 250 m to 175 m over the consented part of the Western Sector (Fergusson 1994a). 25% of these holes intercepted faults, with approximately half having estimated throws exceeding 5m. All of the drillholes were geotechnically and geophysically logged (with dipmeter) to improve the structural interpretation, but the data were generally insufficient to reliably and accurately map the fault pattern and basement ridge extents.

The drillhole spacing in the adjacent Huntly West CML (Figs. 1 and 2) averages 200 m in the developed part of the mine. This drilling was not sufficient to model and predict the frequency and severity of basement relief and faulting as encountered in the pit bottom area resulting in major unplanned development costs (Fowke 1987). Over the remaining part of the Huntly Coalfield, north of the two CMLs, drillhole spacings are in excess of 300 m implying that additional geological investigations will be needed prior to mine development.

Modeling and interpretation of these drillhole datasets indicates that spacings of around 200-300 m are adequate to measure coal quantities and to provide indicative quality data (Fergusson 1994a). However, the data are insufficient to accurately define geological hazards for mine planning. The limitations of surface drillhole data for seam geometry modeling relate primarily to the low number of seam intersections i.e. point sample (Lambourne et al 1991), the inherent limitations of using (often disturbed) drill core for interpreting several types of geological structures, and the relatively
small scale of many structural features. The drillhole spacing required for high reliability structure mapping is probably in the order of 50-100 m (Fergusson 1994b), and therefore cost-prohibitive.

**High resolution seismic reflection data acquisition in Huntly Coalfield**

Huntly Coalfield is a challenging geological setting for HRSR data acquisition. The main reason for this is the thick weathering layer - the Tauranga Group (Quaternary). Seismic energy is readily absorbed and attenuated by a combination of surface peat deposits in the valleys, thick unconsolidated sediments, pumice-rich gravel layers, hard ignimbrite lenses, and buried organic-rich muds. In addition, the soft, unconsolidated nature of these materials, especially the peat, encourages the formation of guided waves (ground roll), which obscure first arrivals during recording.

A combination of modern acquisition techniques and recording equipment, together with close attention to experiments and trials before finalising the technical design of acquisition and processing methods, greatly increases the likelihood of successfully acquiring usable HRSR data (e.g. Lamb et al 1992). The technical strategy at Huntly East Mine involved the following:

- field experiments to “tune in” charge size, depth and spacing; receiver type, configuration, and spacing;
- placing geophones below the surface peat layer;
- drilling very deep shot holes to ensure the signal to noise ratio was sufficient over the thickest Tauranga Group;
- quality assurance procedures during all project stages (surveying, drilling of source and receiver holes, loading, acquisition, and processing);
- acquisition of experimental data before committing to production recording; and
- emphasis on data quality not quantity

**2D HIGH RESOLUTION SEISMIC REFLECTION PROGRAMME**

**2D trial**

In February 1994 an experimental programme began to determine acquisition parameters for a trial 2D line (Table 2). This entailed recording upholes and a refraction line, testing charge size, and geophone design. Once these parameters were tuned-in, a 900m trial line was recorded ahead of the West Headings face position and above the Men and Materials tunnel alignment (Figs. 1 and 4). The line position was chosen to cover a representative range of surface/access and weathering layer conditions, and, if successful, would provide information about seam structure in the direction of mine development.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Experimental Line</th>
<th>Production Lines</th>
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<tbody>
<tr>
<td>Charge type and weight</td>
<td>Anzomex A boosters - 260 gms</td>
<td>A and K boosters - 260 gms</td>
</tr>
<tr>
<td>Max. source hole depth</td>
<td>40m or base weathering layer</td>
<td>50m or 3m below weathering layer</td>
</tr>
<tr>
<td>Source hole interval</td>
<td>20m; between receiver stations</td>
<td>18m; on station</td>
</tr>
<tr>
<td>Receiver interval</td>
<td>5m</td>
<td>3m</td>
</tr>
<tr>
<td>Receiver pattern</td>
<td>Hills: 6 at point Peat: 1 downhole</td>
<td>Hills: 6 perp. to line; 3x3 split Peat: 3 downhole in casing</td>
</tr>
<tr>
<td>Nominal Fold</td>
<td>12</td>
<td>12</td>
</tr>
</tbody>
</table>

Fig. 3a illustrates that good quality HRSR image resulted from this trial and that the problems associated with the Tauranga Group weathering layer could be overcome by some relatively straight-forward field testing. The benefits of
using downhole geophones in peat swamp areas is also evident as data quality is consistent along the entire line. The top of the seam is clearly resolved, as are geological structures. Ralph Fault, Okowha Fault and Taupiri 1 Fault were previously inferred and modeled using surface drillhole data (Fergusson 1994a, b). The fact that these faults were successfully imaged and the resultant seismic interpretation was consistent with the existing data, was critical for the credibility of the experimental programme. The agreement between the nature and positions of faults determined from the 2D profile and the actual intercept positions, sense of offset and degree of drag folding revealed in the development tunnels has proved to be very good (Fergusson 1997).

Fig. 3 - E-W 2d seismic profiles acquired in western sector – 12 fold, deep Anzomex energy source data showing the NNW-trending Ralph Fault: (A) experimental line, (B) Production line CC94-002, (C) production line CC94-003
2D production survey

The trial demonstrated that with appropriate emphasis on technical parameter selection, high quality data can be recorded, and the position of structural geology hazards accurately mapped in the line of proposed mine development alignments. A 2D production survey consisting of five lines plus an extension of the trial line was conducted in May-June 1994 (Fig. 1). In addition to more pre-survey parameter testing (upholes), the programme included drilling and sonic logging of a drillhole to generate a synthetic seismogram for picking and correlating the target coal seam over the survey area (Fig. 1). Migration of one of the lines resulted in improved image quality and refinement of fault positions. Unfortunately, not all lines could be migrated due to budget limitations, however unmigrated fault positions were constrained to ± 10m, which was considered to be acceptable for planning of mine developments. The source hole and receiver spacing were reduced to improve processed image quality (Table 2).

The resultant profiles were of very good quality (Fig. 3b and 3c). The top of the coal seam is clearly imaged, as are normal faults and the crest of “drape” folds marking basement ridges (Fergusson 1994c). Unfortunately, the base of the seam was not consistently resolved by processing. The three E-W seismic lines (Fig. 1) allowed better definition of the Ralph Fault, a major geological hazard and planning obstacle, along strike to the south. They showed that total offset on Ralph Fault reduces southwards, and the proportion of offset represented by normal drag folding increases (Fig. 3). This has been borne out by development and extraction in South 4 Panel on the eastern (downthrown) side (Fig. 4). The three N-S lines indicated a 150-200m wide fault zone made up of several, discontinuous faults with less than half seam thickness offsets. These faults stepped the seam down to the north. The zone represents another major planning obstacle. Basement ridges are also evident south of this fault zone, indicating localised seam thinning is likely.

Despite the success of this programme in imaging the seam “in line”, several limitations with the resultant data were obvious from a structural interpretation and mine management point of view (Fergusson 1994c). The strike of the NE trending faults were inferred because the 2D lines were too widely spaced for correlation purposes. The definition of seam base was poor compared with the top of seam, such that the extent and severity of basement ridges could only be inferred. In short, the chief limitations of 2D seismic reflection had been realised, a fact amply illustrated in the South 4 Panel where a fault striking 015° was intercepted, instead of the more typical 045°, and significantly impacted upon the extraction plan (Fig. 4).

Fig. 4 – South 4 panel, Huntly east mine showing mapped faults and mine workings
RALPH 3D SEISMIC PROJECT

In May-June 1996 work began justifying a 3D survey over the Ralph Block and part of the adjoining Coal Exploration Permit area (Fig. 1). The cost benefits were estimated using the performance of South 4 Panel in terms of unplanned support, maintenance and production costs and inaccessible reserves. An internal rate of return of 14% was determined from discounted cash flow analysis (Fig. 4).

3D experimental programme

Results of field experiments involving upholes, source holes and an experimental geophone spread conducted in May-June 1996 were consistent with the 2D uphole results. The results confirmed that a weathering layer per se is not readily definable in the Tauranga Group sediments due to an absence of strong velocity interfaces below the peat layer (Bjoroy 1996). Signal to noise levels of shots degrade from 40-45m depth upwards. The drop in signal to noise ratio, signal continuity and frequency content was found to be sudden rather than gradual. It was concluded that the interval between 5-50 m can be treated as a single horizon geophysically (Bjoroy 1996).

In light of this, source hole depths in the Tauranga Group could not be based on lithological criteria. Three target depth criteria were formulated for source hole drilling:

- a minimum depth of 25 m where Te Kuiti Group was intercepted at shallow depths
- an intermediate depth criterion of drilling three metres into Te Kuiti strata where Tauranga Group was 25-47 m thick
- a maximum depth of 50 m where Tauranga Group was more than 50 m thick

To complement the field experimentation, a series of processing trials was conducted on one of the 2D lines to determine the optimum processing sequence and the effects of data interpolation (Hawkes 1996).

Ralph 3D survey

The Ralph 3D project started in November 1996, and was concluded in June 1997. The survey area was 0.82 km² (Fig. 1). The survey area was oriented such that receiver lines were perpendicular to the dominant NE fault trend determined from mapping in the adjacent South 4 Block (Fig. 4). The survey area was made as square as practicable to minimise the area of low fold peripheral data. The technical objectives of the programme were:

- Determine the extent and thickness of mineable coal;
- Define the location, extent and severity of basement ridges;
- Define the location, trend and offset of faults; and
- Vertical resolution in the order of three metres.

Recording was conducted by Geco-Prakla and took place in January 1997 using an Input/Output System 2 data acquisition system. Sub-surface fold exceeded 12 over approximately 70% of the survey area, 90% of which was 14-20 fold data. The top and base of seam were imaged at over 32,000 5 m x 5 m “bins” in the subsurface. Processing was carried-out using Gecoseis software. Time-structure interpretation and depth conversion was carried-out using GeoQuest software (IESX and InDepth).
Fig. 5 illustrates the high quality of the processed data volume. Top and base of seam reflectors are very obvious between 0.15 to 0.20 seconds two way time (TWT). Faults were interpreted where they cut the seam boundary reflectors. Some faults were seen cutting a shallower reflector at approximately 0.12 to 0.15 seconds TWT (= Pukemiro Sandstone).

Data sets containing the depth converted time-structure points were modeled using Vulcan mine planning software (McGuire 1997). Triangulations were generated of seam roof and base and faulted using the fault trace data from the 3D interpretation (Fig. 6). Seam thickness was derived from "grid arithmetic" using grids of seam floor and roof, thus accounting for thinning across faults and above ridges (Fig. 7).
The seam base model revealed a pre-seam basement topography consisting of a complex system of ridges and valleys representing the erosional surface on the pre-coal measure basement surface (Fig. 6). The basement relief indicated by drilling and 2D seismic in the south of Ralph Block has been confirmed in detail by the 3D survey. The North and South ridges are connected by the Central Horst/Ridge, which runs along the western side of the Ralph Block. Consideration of the position of this Central Ridge and the location and throw of the faults has allowed the South 6 Headings to be positioned where the geological hazards will have the minimum impact on mine development. Development tunnels have been designed in three dimensions to negotiate structures with minimal stone drivage and to meet tolerances of mine infrastructure (such as the vertical curvature of conveyor belt structures, and maximum grades for mine vehicles).

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A total of five previously unknown faults with throws ranging between 2 - 25 m, and numerous sub-5 m throw faults were defined (Figs. 5 and 6). The main faults define two fault troughs (SW Graben and East Graben) and an intervening structural high. These areas have been planned as distinct mining blocks with prescribed lower rock mass conditions (Roy 1997), requiring separate development and extraction plans. The Ralph Fault dies-out against the Southern Fault Zone (SFZ). The SFZ continues southwestwards as a series of irregular, discontinuous, minor faults which collectively form a planning obstacle.

The derived seam structure and thickness maps (Fig. 7) are consistent with observations from drillholes and development tunnels. The West Headings were stopped 50-75 m from the CML boundary because the seam started to thin and dip-over steeply. It is now apparent this is due to encountering the NW end of a basement ridge (the North Ridge - Fig. 6). The South 6 Headings have encountered thinning and steep seam grades as they develop southwards into Ralph Block consistent with climbing up the NE flank of the North Ridge.

Once accurately depth-converted, the 3D results can be used to model the distribution of coal quality parameters within a "real" geometrical framework allowing more accurate scheduling of run-of-mine coal quality. At Huntly East Mine, the first iteration of detailed mine planning and scheduling has indicated months where run-of-mine coal quality will be out of
specification, and demonstrated that the ratio of development to extraction was too high to achieve the mine’s production targets in the last two quarters of the operating plan. These problems are being corrected to achieve product specifications and planned tonnages.

A combination of basement relief (West Ridge), seam thickness and drillhole data has been used to defined the limit of extraction in the CEP (Figs. 6 and 7). Development and extraction blocks have been designed and scheduled for the next three years using the geological model derived from the 3D HRSR survey.

**RECONCILING THE MODEL**

Development tunnels in the South 6 Heading are accessing the Ralph Block coal reserves (Figs. 6 and 7). They have successfully negotiated the North Ridge, revealing its position, trend and size to be consistent with the seismic-derived model. The coal seam was less than 6 m thick locally on the North Ridge crest, as opposed to 6-9 m as modelled (Fig. 7). The difference is assumed to be due to the higher level of subjectivity in picking the base of seam reflection, which has a variable reflector due to thickening and thinning of the coal caused by relief on the immediately underlying surface on which the seam rests. Three development tunnels have now intercepted Fault “b” (Fig. 8) and have substantiated the geometry of this fault in terms of location, dip, and decrease in throw and swing in strike toward its western tip. Face drilling data from the developments have been modeled and compared to the seam roof model derived from the 3D time-structure map. The actual seam roof position agrees within ±2 m.

Although development of the Ralph 3D survey area is at an early stage, a reliability level in the order of 90% appears to have been achieved with the seam structure model. However, further development and observation is required to substantiate quantify the reliability of the model, especially in respect to the base of the seam and the fault system.

**CONCLUSIONS**

Since 1994, 2D and 3D high resolution seismic reflection techniques have been successfully applied to investigating the geometry and structure of the Kupakupa seam in Huntly Coalfield. These techniques, especially 3D HRSR, have major advantages over drillhole-based investigations for mine planning in structurally complicated coal deposits such as Huntly Coalfield.

2D HRSR is useful for planning along pre-determined alignments, such as mine development directions, but has some major limitations for defining structural trends and apparently does not fully resolve seam base. 3D HRSR yields detailed geometry models of the seam. The trend, location and nature of faults are very obvious and require minimal “interpretation”. Faults have been traced out to the limit of resolvable offset, which is 2-3 metres. Basement ridges are evident on 2D lines as subtle drape folds on top of the seam, whereas in 3D datasets their severity and lateral extent are evident on time-structure maps and can be readily modeled and structure contoured once the time-domain data sets have been depth-converted.

Geological models with high reliability allow management of mining risks such as unplanned development costs, loss of reserves and low productivity. Other benefits, especially for “greenfield” deposits, include extraction method design (especially relevant in thick seams) and definitive land-use management. Reliable models of the seam structure also have potential to improve strata control strategies. For example, fault troughs and basement ridges are known to be poor coal mass and distressed coal situations requiring planned strata control measures to be implemented during development to ensure planned development and extraction rates are achievable, and tunnels are stable in the short-term.
An investigations technique which delivers a seam geometry model with a confidence level in the order of 90% is a major step forward in risk management for planning underground mines in the Huntly Coalfield. The turn-around time from acquisition to mine planning was in the order of 10-12 weeks.

HRSR techniques can be employed independent of the mining operation, well in advance of mine development. This allows mine planning, risk assessment and economic appraisal of new reserve blocks in existing mines, mine extensions and, indeed, new underground prospects prior to committing capital expenditure. Coal quality drilling can be targeted to...
sample coal reserves which will be extracted, with improved cost-effectiveness through fewer holes and representative results.

A "stable" mine plan (i.e. one which is not altered every time a significant geological hazard is intercepted) has major potential cost benefits to the mining company. For example it allows coal quality scheduling and forward planning of run-of-mine blending scenarios to meet product specifications. In the area of reserves optimisation, mine plans based on reliable geology models allow mine management to focus on value-generating tasks such as production engineering, rather than "fire-fighting" and endless revision of the mine plan with only apparent improvement in benefits.

ACKNOWLEDGMENTS

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Coal Geology  Where to From Here?

C Turvey¹ and P Hanna²

INTRODUCTION

Coal geologists are under pressure to minimise exploration expenditure, maximise interpretation confidence and produce the results often too soon. This pressure continues to increase as companies are forced to evaluate borderline economic deposits which are usually deeper and more hazardous, such as the Togara North and Togara South projects in Central Queensland.

In response to this pressure, coal geologists are having to integrate and utilise every piece of information available including stratigraphic, structural, analytical, geophysical, geotechnical, hydrological and more recently, environmental data. In Australia, coal geologists rely on research to continually improve both data acquisition and data interpretation. Some of these developments are discussed as well as suggestions for improvements in other areas within this paper.

Following the application of modelling and remote sensing issues integral to the successful evaluation and development of coal mines, the long, medium and especially short term production factors dominate the applied mining geologist's time to a far greater extent than the development of an understanding of the mine geology.

The primary factors to be considered by the mine geologist include:

- The accurate short term modelling of mining blocks for drill and blast, reserves and reconciliation;
- The accurate delineation of in-pit structure, particularly seam splits, rolls, wants, faults and intrusions;
- Provision of accurate and relevant geological and geotechnical information to all levels of mineworker, from the general manager to the engineers, foremen, operators, coal handling preparation plant (CHPP) personnel and owners; and
- Accurate pit reconciliation to establish coal loss and dilution leading to better pit control and continuous improvement.

DRILLING DATA

As drilling budgets have tightened, one of the lessons coal geologists have learnt in the past decade is that every drillhole counts and every piece that can be extracted from the drillhole must be recorded and utilised.

Logging drill core today involves recording:

- Lithological descriptions relevant to impact on mining conditions eg. strength, state of core, and bedding.
- Geotechnical data such as rock quality density, fracture and joint locations, nature of breaks, and mineralisation of joints.
- Water flow rates and standing water levels.
- Any anomalies that occur during drilling eg. loss of circulation, gaseous odours, and hole caving. These should be noted on geological hazards maps.

¹ Bengalla Mining Co. Pty. Ltd.
² ECS International Pty. Ltd.
Core photography ensures a permanent record of drill core for reference by mining professionals for blasting, equipment selection, trafficability, roof bolting and mine design.

Advances in digital photographic equipment will allow geologists to include core "photographs" in geological reports, on borehole stratigraphic profiles and cross-sections. As the resolution of digital cameras increase, geologists will be able to examine core photographs in detail, zoomed in on their computer workstations.

Coal geologists can perform a number of tests on-site on fresh core as apparatus becomes compact. Tests include:

- Gas desorption testing to ensure minimal loss of gas over time.
- Point load UCS tests.
- Downhole geophysical logging recorded in an international standard digital format.

Geophysical logs have become an essential tool in the evaluation of coal deposits as they provide information on:

- Accurate seam and interburden stratigraphy;
- Detail correlation of seam splitting;
- Geotechnical data – fracture and rock strength; and
- Structural information – Sirolog can determine coal ash content and a variety of elements including Fe, Ca and Si.
- These data can be readily incorporated into geological database systems and utilised for detailed geological interpretation including:
  - Cross-correlation of logs with analytical results.
  - Cross-correlation of logs with geotechnical results.
  - Seam splitting correlations on cross-sectional and 3D borehole displays.
  - Interpretation of the lithologies in chip drill holes.

Chip drill holes can now be automatically interpreted using neural network technology. This system learns geophysical log responses for detailed cored drill holes and then applies this knowledge to the geophysical logs for chip drill holes (Fig. 1).

Artificial Intelligence (AI) has also been applied to correlation of drill hole lithologies (Fig. 2), designing open-cut and underground mines as well as optimally scheduling mines to meet product tonnage and coal quality targets for multiple markets.

GEOTECHNICAL INTERPRETATION

Cross-correlation of UCS strength measurements with downhole sonic logs gives a continuous record of rock strength down each drill hole. This information can be used to evaluate roof and floor conditions across the deposit by modelling cumulative UCS over one metre intervals of roof and floor strata. The resultant maps clearly indicate zones of relatively weaker and stronger strata which will help determine roof bolting requirements and areas where productivity may be affected due to difficult conditions.

The same data can be utilised in open-cut mining by evaluating the change in strengths of interburdens and overburden across the deposit. This will provide valuable information for blasting requirements and ripability.
Coal geologists are finding valuable information from the interpretation of geophysical surveys. This is due to two recent developments, the ability to record high resolution data and the advances in interpretation software.

Image processing of high resolution aeromagnetic and ground magnetic surveys is proving invaluable in the structural interpretation of coal deposits as well as delineating igneous bodies. Images are enhanced by superimposing SPOT data as well as for gridded model seam surfaces derived from borehole data.
Improved methods of filtering and displaying data have given geologists a clearer picture from which to interpret structural geology.

Surface seismic surveys have also undergone improvements in areas of data collection and interpretation. Today, seismic surveys in Queensland coalfields are confidently delineating faults with as little as a 2 metre throw. The major limitation of 2D seismic surveys, that is, determining the orientation of faults, has been overcome with the development of 3D seismic surveys in recent years. Although regarded as an expensive tool, 3D seismic costs would total less than 10% of the costs a company would suffer in an area riddled with small faults and bad ground conditions.

The major portion of 3D seismic costs is in drilling shot holes to below the depth of incompetent material. Research programs are underway to develop a cheaper and faster form of seismic source such as high frequency vibrators which are being designed to produce a stronger and clearer signal from the surface using a track-mounted compact and versatile vehicle.

Seismic interpretation software has undergone extensive improvements in recent years. Using downhole geophysical sonic logs, a synthetic seismogram (Fig. 3) can be computed, and subsequently be positioned on a seismic section using horizon markers such as seams for references. The borehole depths are then used to accurately convert seismic time sections to depth sections. Major coal seam reflectors can be automatically traced and converted to real depths, producing digital geological cross-sections which can readily be integrated into a model structure surface.
SHORT TERM MODELLING

To adequately model the short term requirements of an operating mine, or one commencing operation, it is assumed that an adequate and integrated modelling, coal quality, survey, scheduling, simulation and reporting package is in place. Without such a package, needless time is lost in data formatting, exchange and rehandling which causes losses in time, expertise, money and efficiency.

An adequate short term model implies a reasonably adequate long term model incorporating the above mentioned components. This is where the GIGO (garbage in - garbage out) principle applies, where inadequate information and poor data collection will produce a poor model.

In today’s world the buzzwords are “best practice”. Many operations do not enjoy the full benefit of the information they now have available nor are the workers often able to realize their full capabilities due to outdated methods, the “we’ve
always done it that way" attitude, lack of resources e.g. computer power and software, lack of management focus and mine equipment limitations.

Best practice in the future will include on-line, real time data acquisition from blast drills. The drilling of a blast hole involves the parameters of weight on bit; in coal mining this equates to pull-down, rate of penetration, rotary speed, rotary torque, bit size, type and jet size and the air pressure/volume. These factors may be combined to provide a drilling exponent, similar to that calculated from parameters used in drilling oil - gas wells. Drilling exponents, especially from such a small area as a coal mine usually defines, can be correlated to determine rock type and depth markers, to avoid drilling into, or too close to, coal seams. The importance of this to reducing blast damage to coal is fundamental and has been covered by many projects and reports. This information will allow the drill and blast engineer to determine relative rock strengths affecting powder factors, pattern design and blasthole charging.

The real time monitoring of drilling parameters is now possible. Further, other studies currently being undertaken include monitoring while drilling by geophysical logging methods, as practiced in the oil and gas industry and in in-seam drilling. Other projects include the use of bit vibrations to create a seismic model ahead of the drill bit of rock type. The selective and controlled drilling of blastholes to, or into, and in some cases where log - quality correlations exist, through the target coal seams, allows the rapid modelling of the blast pattern to determine drilling depths for the remainder of the pattern together with blasting profiles of the holes for explosive types, decking etc. Pre-split holes are particularly useful as they are generally drilled prior to the main pattern. In areas of rapid geological variation selected main pattern holes may be drilled to coal. Currently, many geologists rely on drillers depths, if available and recorded, drill cuttings analysis, geophysical logging and drill pull-down logs to provide the information to those few who do model these data. Such methods are usually time and labour intensive, leading to less than optimal utilization, if viewed at all. The ability to receive real-time, accurate data for every hole ensures rapid model adjustment and communication to the drill and blast crew. Further, such information will be used in the pit loss reconciliation process.

STRUCTURAL DELINEATION

The accurate delineation of in-pit structure, particularly seam splits, rolls, wants, faults and intrusions follows from the information gained prior to and during the mining process. Primary methods of collecting this information include in-pit mapping and information gained from the drilling and monitoring of blast holes as mentioned above.

In-pit information may be lost due to inaccessibility and/or dangerous conditions. The accurate and timely mapping of highwall faces may now be possible with the application of laser and GPS technology. Faces may be mapped for blast-hole drilling information, the current primary application of this technology which is often carried out by explosive companies. Future applications of this method will result in the precise and rapid mapping of lithological units, structures and seam profiles. These data could then be quickly downloaded, modelled and utilised in pit reconciliation, structural locations, seam location and lowwall locations for the following strip, reducing the likelihood of lowwall edge losses.

Further advances in the delineation of these features may result from the application of seismic data monitoring by long term fixed seismic geophones of defined in-pit blasts, probably from pre-split blast holes. Long term monitoring of this information over varying blocks and benches may yield 3-D seismic models of the strata ahead of the mining face. Correlation of the data with observed pit structures and strata may lead to very detailed images of the proposed mining area ahead of the current coal face.

COMMUNICATION

Provision of accurate and relevant geological and geotechnical information to all levels of mineworker, from the general manager to the engineers, foremen, operators, CHPP personnel and owners, together with adequate and ongoing education of all employees is essential to a successful and profitable mining operation. The primary aim of coal mining is to take a geological resource and create a saleable product as efficiently and profitably as is safely and financially possible. The aim is not to provide a scheduling or equipment challenge for engineers, a creative accounting exercise, a research thesis to a geologist nor an industrial relations exercise.

The aim is to provide real-time, online geological, mine design, scheduling, survey and performance data essential to the continued improvement of productive and profitable performance of personnel and hence equipment. This does not mean that the geologist or engineer hides behind a computer or a technical vocabulary at the expense of in-pit production issues.

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This technology allows these personnel to spend more time where it is most productive, that is balanced between the books and the coal face and not sidelined in a paper war.

Real-time information may be gained from the use of global positioning system (GPS) on equipment for accurate positioning, e.g. blasthole rigs, particularly when drilling angled face holes. The continuous real-time monitoring of data, be it geological, engineering or production is essential and information will be relayed back to operators by colour monitors including mining method, location, geology, mining horizons, productivity and future tasks. Alternative mining methods may entail changes to dig methods, dozer cleanup and so on to minimise coal loss and dilution.

**PIT RECONCILIATION**

Accurate pit reconciliation is necessary to establish coal loss and dilution leading to better pit control and continuous improvement. The first step in this process is to know how much coal is available for mining. The accurate measurement of truck loads must be made on an individual load basis, particularly when trucks of different load capacity are used for the same coaling operation. Further, accurate weight measurement must be made at the washery of the run-of-mine (ROM) and clean coal stockpiles and washery balances calculated. Real-time truck loads may be transmitted to the geologist together with washery information. The rapid determination of ROM recovery from mining blocks allows rapid feedback and identification of loss areas. ROM ash levels may be compared to expected ash levels for mining blocks, dilution calculated and action taken to reduce excessive dilution, e.g. better seam clean-up, during the mining process rather than some months later.

Clean coal yield from washed product can be related to ROM feed, affected by dilution and loss of particularly weak bright coals, and in-situ predicted recoveries. Rapid feedback of this information allows the reconciling of yield and ash data to coal losses, dilution to the observed mining operation, the correction of mining technique and equipment utilisation if necessary.

**CONCLUSION**

With advancements in drilling communication and computer technologies, coal geologists can look forward to a future where field results will be evaluated in real time. It is conceivable that soon, drilling rigs will be equipped with downhole geological logging tools, that will pulsate results to the surface whilst drilling progresses. The results will be conveyed via satellite email communication back to a project office with a workstation using artificial intelligence to convert the results into lithological, geotechnical and analytical data plotted in full colour on a 3D perspective screen display. A drilling program involving numerous drilling rigs would be interpreted, evaluated and re-appraised in real time.
Coal Pillar Design Issues in Longwall Mining

W J Gale

ABSTRACT

Coal pillar design has been based on generalised formulae of the strength of the coal in a pillar and experience in localised situations. Stress measurements above and in coal pillars indicate that the actual strength and deformation of pillars varies much more than predicted by formulae. This variation is due to failure of strata surrounding coal. The pillar strength and deformation of the adjacent roadways is a function of failure in the coal and the strata about the coal. When the pillar is viewed as a system in which failure also occurs in the strata, rather than the coal only, the wide range of pillar strength characteristics found in the UK, USA, South Africa, Australia, China, Japan and other countries are simply variations due to different strata-coal combinations and not different coal strengths.

This paper presents the measured range of pillar strength characteristics and explains the reasons. Methods to design pillar layouts with regard to the potential strength variations due to the strata strength characteristics surrounding the seam are presented.

INTRODUCTION

The strength characteristics of coal pillars has been studied by many workers and the subject is well discussed in the literature (for example, Salamon and Monroe, 1967; Wilson, 1972; Hustrulid, 1976; Mark and Iannacchione, 1992; Gale, 1996). In general a range of strength relationships have been derived from four main sources:

1. Laboratory strength measurements on different-sized coal block specimens;
2. Empirical relationships from observations of failed and unfailed pillars;
3. A theoretical fit of statistical data and observations; and
4. Theoretical extrapolation of the vertical stress buildup from the ribside toward the pillar centre, to define the load capacity of a pillar.

These relationships provide a relatively wide range of potential strengths for the same pillar geometry. In practice, it has been found that various formulae are favoured (or modified) by users, depending on past experience in their application to certain mining districts or countries.

In general, the application of empirically and statistically based formulae has been restricted to the mining method and geological environment for which they were developed, and they often relate to specific pillar geometries. Such relationships have usually been developed for relatively small pillars having width to height (W/H) ratios less than 5, and can only be used with confidence in these situations. In general these methods were developed for shallow, extensive bord and pillar operations for which the pillar was designed to hold the weight of overburden. The development of stress measurement and detailed rock deformation recording tools over the last 10-15 years has allowed much more quantification of actual pillar stresses and deformations. Little of this data were available when many of the pillar strength relationships were originally defined. Similarly, the development of computer simulation methods has allowed detailed back analysis of the mechanics of strata-coal interaction in formed up pillars.

The author and his colleagues have conducted numerous monitoring and stress measurement programs to assess roadway stability and pillar design requirements in Australia, UK, Japan, USA, Indonesia and Mexico. The results of these investigations, and others reported in the literature, have demonstrated that the mechanical response of the coal and surrounding strata defines the pillar strength, which can vary widely depending on geology and stress environment. The application of a pillar strength formulae to assess the strength of a system which is controlled by the interaction of geology, stress and associated rock failure is commonly an over simplification.

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MECHANICS OF THE PILLAR-COAL SYSTEM

The strength of a pillar is basically determined by the magnitude of vertical stress which can be sustained within the strata/coal sequence forming and bounding it. The vertical stress developed through this sequence can be limited by failure of one or more of the units which make up the pillar system. This failure may occur in the coal, roof or floor strata forming the system, but usually involves the coal in some manner. The failure modes include shear fracture of intact material, lateral shear along bedding or tectonic structures, and buckling of cleat bounded ribsides.

In pillar systems having strong roof and floor, the pillar coal is the limiting factor. In coal seams surrounded by weak beds, a complex interaction of strata and coal failure will occur and this will determine the pillar strength. The strength achievable in various elements is largely dependent on the confining stresses developed as illustrated in Fig. 1. This indicates that, as confinement is developed in a pillar, the axial strength of the material will increase significantly, thereby increasing the actual strength of the pillar well above its unconfined value.

The strength of the coal is enhanced as confining stress increases toward the pillar centre. This increased strength is often related to the width/height ratio, whereby the larger this ratio the greater the confinement generated within the pillar. Hence squat pillars (high W/H) have greater strength potential than slender ones (of low W/H).

![Fig. 1 - Effect of confining stress on compressive strengths of intact and fractured rocks (Note that “failed” rocks should read fractured)](image)

The basic concepts related to confinement within coal pillars was developed by Wilson (1972) and with the growing availability of measurement data these general mechanics are widely accepted. However, confining stress can be reduced by roadway deformations such as floor heave, bedding plane slip and other failure mechanisms. These mechanisms are described below.
Roadway development phase

Prior to mining, the rock and coal units will have in situ horizontal and vertical stresses which form a balanced initial stress state in the ground. As an opening (roadway) is created in a coal seam, there is a natural tendency for the coal and rock to move laterally and vertically into the roadway. In this situation, the horizontal stress acting across the pillar will form the confining stress within that pillar. If this lateral displacement is resisted by sufficient friction, cohesion and shear stiffness of the immediate roof and floor layers, then most of the lateral confining stress is maintained within the pillar. Consequently, the depth of "failure" (yield) into the pillar ribside is small. If the coal and rock layers are free to move into the roadways by slippage along bedding planes or by shear deformation of soft bands, then this confining stress will be reduced. Hence the depth of failure into the pillar ribside may be significantly greater.

The geometry of failure in the system and the residual strength properties of the failure planes will, therefore, determine the nature of confining stress adjacent to the ribside and that extending across the pillars. This mechanism determines the depth of failure into the pillar and the extent of ribside displacement during roadway driveage.

Pillar loading by abutment stresses

Roadways are subjected to an additional phase of loading during longwall panel extraction, as front and then side abutment pressures are added to the previous (and generally much smaller) stress changes induced by roadway excavation. These abutment stresses typically considered are predominantly vertical in orientation, but can generate additional horizontal (confining) stresses (by the "Poisson's ratio effect") if there is sufficient lateral restraint from the surrounding roof and floor. Conversely, if the ground is free to move into the roadway then this increased horizontal stress is not well developed, and increased rib squeeze is manifest instead.

This concept is presented in Fig. 2, where with strong cohesive coal/rock interfaces, the confining stress in the pillar increases rapidly inwards from the ribside, allowing high vertical stresses to be sustained by the pillar. The opposite case, of low shear strength coal/rock contact surfaces, is presented in Fig. 3. In this situation confinement cannot be maintained sufficiently, hence the allowable vertical stress would be significantly less than in Fig. 2. The diagram shows that the pillar has failed due to its inability to sustain the imposed vertical abutment stresses. In addition, lateral movement has caused floor heave and severe immediate roof shearing.

The implications of this for the strength of an isolated pillar are presented in Fig. 4, where the load carried by the pillar is the mean of the vertical stress across it. If this mean stress is equal to the average "applied load" to be carried by the pillar, then the pillar is stable (Fig. 4a). If the applied load is greater, then the pillar is said to fail (Fig. 4b) and the deficit stress must be redistributed onto nearby pillars.

Conceptually, pillar strength behaviour should fall between the two end members:

- Lateral slip occurring totally unresisted, so that pillar strength is limited to the unconfined value of the coal; and

- Lateral slip being resisted by system cohesion and stiffness, such that pillar strength is significantly above its unconfined value due to confinement.

A range of potential pillar strengths associated with these two end members, relative to W/H ratio, is presented after Gale 1996, in Fig. 5. It is assumed that the rock mass strength of the coal is 6.5MPa, and that the coal is significantly involved in the failure process. This range of pillar strengths is representative of most rock failure combinations, except in rare cases where small stiff pillars may punch into soft clay-rich strata at loading levels below the field UCS of the coal. In the punching situations, pillar strength may be lower than that depicted, but the variation would generally be confined to pillars having small width/height ratios.
Fig. 2 - Rapid build up of vertical stress into the pillar where high confining stresses are maintained

Fig. 3 - Slow build up of vertical stress in the pillar where slip occurs and confinement is reduced
Stress in pillar

Average pillar stress

Stress in pillar = load

Stable

a) Strong System.

Stress in pillar = 0.5 load

Failed

b) Weak System.

Fig. 4 - Pillar strength cases for strong and weak geologies

A comparison of these "end member" situations with a range of pillar strengths determined from actual measurement programs conducted in Australia and the UK by SCT and from USA (Mark et al, 1988) is presented in Fig 6. The comparison indicates that a wide range of pillar strengths have been measured for the same geometry (in terms of W/H), and that the data appear to span the full interval between the end members. However, two groupings can be discerned and are shaded in Fig. 7:

- The "strong-normal" geologies, where pillar strength appears to be close to the upper bound; and
- The structured or weak geologies, where the strength is closer to the lower bound and where it is apparent that strength of the system is significantly limited.

It should be noted that these two groupings are arbitrary and possibly due to a limitation of data. With more data points the grouping may become less obvious.

Effect of geology

It is clear that a wide range of pillar strengths are possible, and that these are not only related to coal strength and width/height ratio. Geological factors have a major impact on the strength achievable under the various pillar geometries.

Effect of geology on pillar strength

The effect of various strata types in the roof-coal-floor pillar systems has been investigated further by computational methods.

Computer models of four pillar systems were loaded to determine their strength characteristics.
Fig. 5 - Range of potential pillar strengths relative to width / height based on confinement variation

Fig. 6 - Pillar strength information relative to changes
The pillar systems are presented in Fig. 8 and are:


b) weak siltstone – coal – weak siltstone;

c) massive sandstone – coal – massive sandstone; and

d) laminate – coal – sandstone;

The results of the pillar strength characteristics relative to width/height are presented in Fig. 9. The results closely relate to the field measurement data and confirm that the strata types surrounding the coal have a major impact on strength and also provide an insight into the geological factors affecting strength. The results indicate that:

- Strong immediate roof and floor layers and good coal to rock contacts provide a general relationship similar to the upper bound pillar strength in Fig. 5.;

- Weak, clay rich and sheared contacts adjacent to the mining section reduce pillar strength to the lower bound areas;
• Soft strata in the immediate roof and floor, which fail under the mining induced stresses, will weaken pillars to the lower bound areas; and

• Tectonic deformation of coal in disturbed geological environments will reduce pillar strength, though the extent is dependent on geometry and strength of the discontinuities.

LAMINITE

$\text{UCS} = 49\text{MPa}$

Coal

Clay

LAMINITE

$\text{UCS} = 49\text{MPa}$

SANDSTONE

$\text{UCS} = 82\text{MPa}$

Coal

SANDSTONE

$\text{UCS} = 82\text{MPa}$

LAMINITE

$\text{UCS} = 49\text{MPa}$

CLAY

$\text{UCS} = 0.5\text{MPa}$

SILSTONE

$\text{UCS} = 15\text{MPa}$

COAL

COAL

CLAY

$\text{UCS} = 0.5\text{MPa}$

COAL

LAMINITE

$\text{UCS} = 49\text{MPa}$

SILSTONE

$\text{UCS} = 15\text{MPa}$

Fig. 8 - Geological sections modelled to assess load / deformation characteristics

Obviously, combinations of these various factors will have a compounding effect. For example, structurally disturbed, weak and wet roof strata may greatly reduce pillar confinement and, consequently, pillar bearing capacity.

Effect of geology on post peak pillar strength

The post peak pillar strength characteristics for some of the pillars modelled is presented in Fig. 10. The pillar strength is presented as a stress/strain plot for various width/height pillars. The results presented in Fig. 10a show that in strong sandstone geology, high strengths are achievable in small pillars (W/H=5) and the pillar maintains a high load carrying
capability. In sections of laminite roof these pillars may lose strength if the laminite fails at a very high load above the pillar. For pillars having a width/height less than 4/5 a loss in strength is expected at a high load due to failure of the coal.

In pillar systems having weak strata surrounding the coal, the pillars typically exhibit a strength loss after peak load is achieved. Large width/height pillars are required to develop a high load carrying capacity after failure in the weak pillar systems modelled. Two examples are presented in Fig. 10b where the post peak strength characteristics of pillars having weak mudstone or clay surrounding the coal. In these examples the strength loss is greatest in the situation of weak clay surrounding the coal.

The implications of this are significant for the design of barrier pillars and chain pillars where high loads are anticipated.

If excessive loads are placed on development pillars in this environment, pillar creep phenomena are possible due to the load shedding of failed pillars sequentially overloading adjacent pillars.

The effect of load shedding in chain pillars when isolated in the goaf is to redistribute load onto the tailgate area and to potentially display increased subsidence over the pillar area. The typical result is to have major tailgate deformation requiring significant secondary support to maintain access and ventilation.

AN APPROACH TO PILLAR DESIGN

Pillar strength

Field studies suggest that a range of strengths are possible ranging, within upper and lower bounds. If we make use of these relationships as “first pass estimates” to be reviewed by more detailed analysis later, then a number of options are available. In known or suspected “weak geologies” the initial design may utilise the lower bound curve of the weak
geology band in Fig. 7. In good or normal geologies, the Bieniawski or squat pillar formulae may be suitable for initial estimates.

Two obvious problems with this approach are that:

- Estimates of pillar size can vary greatly, depending on the geological environment assumed; and
- The pillar size versus strength data set used (Fig. 6) is limited.

This is why such formulae or relationships are considered as first pass estimates only, to be significantly improved later by more rigorous site specific design studies, utilising field measurements and computer simulation.

Design based on measurement requires that the vertical stress distribution within pillars be determined and the potential strength for various sized pillars be calculated. It is most useful to measure the vertical stress rise into the pillar under a high loading condition, or for the expected "working loads". The stress measurement profiles are used to determine the potential load distributions in pillars of varying dimension, and hence to develop a pillar strength relationship suitable for that geological site. An example is presented in Fig. 11.

Computer modelling methods have been developed to simulate the behaviour of the strata sections under various stressfields and mining geometries. For mine design, such simulations need to be validated against actual ground behaviour and stress measurements. This provides confidence that sufficient geological investigation has been undertaken, and that the strength properties and deformation mechanisms are being simulated accurately.

Computer simulation methods are being developed which can be applied to determining the strength characteristics of various strata systems. The accuracy of the computer software developed by SCT has been verified in a number of field investigations where computer predictions of stress distributions and rock failure zones have been compared. An example is presented in Fig. 12 which compares the measured and modelled stress distribution over a yield pillar and solid coal in a deep mine. The comparisons indicate that rigorous computer simulation methods can provide a good estimation of the actual stresses and ground failure zones.

One major benefit of computer modelling is that the behaviour of roadways adjacent to the pillars can be simulated. In this way the design of a pillar will not only reflect the stress distribution within it, but also its impact on roadway stability.

In mining situations where there are large areas of solid ground about the working area the potential for regional collapse of pillars are typically low. Design in these areas usually relates to optimising roadway conditions and controlling ground movements rather than by the nominal pillar strength. Yield pillars and chain pillars are obvious examples of this application. Design must assess the geometry of other pillars and virgin coal areas in determining the impact of a particular stress distribution within a pillar, and the ability of the overburden to span over a yielded pillar and safely redistribute the excess stress to adjacent ground. Fig. 12 shows an example of this process for a failed ("yield") pillar adjacent to solid ground for which vertical stress above the yield pillar has been "shed" to the solid coal abutment area.

**Abutment load estimates**

In virgin conditions, the tributary method is widely used for an approximation to the average stress across a pillar, however for other conditions mining abutment loads should be determined on the basis of monitoring data bases or computer modelling of the actual mine geometry. Estimation of the abutment geometry is often done on the basis of empirical formulae (eg Wilson, 1972; Mark, 1992) or a range of stresses possible depending on various assumptions of goaf loading and post peak pillar strengths. Measurement programs may be required to validate the load distributions in certain geometries to ensure that further prediction of loading in inaccessible areas (eg in the goaf) are justified. The goaf load capacity is a function of extraction width and the bridging capacity of the ground. Monitoring and modelling has demonstrated a significant variation in abutment stress distributions resulting from differing geological sequences and virgin stress systems.
Fig. 10 - Post peak strength of models
Typical Stress Measurement Locations.
(Not to scale).

Fig. 11 - Stress measurements over ribsides for strength assessment

Stress Distribution in Pillar from Measurements.

Fig. 12 - Stress over yield pillar adjacent to longwall
In certain situations failure may extend above and/or below the coal pillar which modifies the vertical stress abutment geometry. These failure modes are predicted by computer modelling and microseismic monitoring currently undertaken by CSIRO/SCT (Kelly et al., 1997). Lateral stress relief into the goaf in high stress environments can reduce the abutment magnitude but extend the zone of influence. Two examples are presented in Fig. 13 for a ribside for which rock failure extends over the ribside and a situation of stronger roof in a high lateral stressfield. In these examples computer modelling of the caving process within the geological section has given a very close correlation with the measured data. The use of generalised empirical methods to determine the abutment profile is also presented and indicates that their application is best utilised as initial estimates to be reassessed by site specific investigations for key design areas.

CONCLUSIONS

The strength characteristics of pillars is dependent on the strength properties of the strata surrounding the coal.

The post failure strength of pillars is an important issue to consider in design particularly in areas of weak strata, where a post failure strength loss in moderate to large width/height pillars is possible.

Failure of strata above and below chain pillars is possible and has been confirmed by microseismic investigations.

Field measurement (stress measurement, microseismic monitoring, rock displacement) and computer modelling provide methods to assess the strength of pillars and the areas of ground fracture.

Computer simulation methods in association with site measurements are recommended for the design of key layouts which require an assessment of geological variations, pillar size and stressfield changes to optimise the mining operation. This approach also assesses the expected roadway conditions or pillar response for various mine layouts and which can be monitored to determine if the ground is behaving as expected.

REFERENCES


Fig. 13(a) Longwall side abutment profiles for modelled, measured and empirical approaches. In this example rock failure occurred the pillar forming a more extensive yield zone.

Fig. 13(b) - Longwall side abutment profiles for modelled, measured and empirical approaches in a high stress mining area.
Impact of Longwall Width on Overburden Behaviour

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ABSTRACT

The longwall panels at Clarence Colliery have experienced intermittent sudden weightings on the face that have caused some production delays. These weightings have typically been more severe on the wider faces. A program of surface subsidence and extensometer monitoring was undertaken above Longwalls 4 and 5 to investigate the behaviour of the overburden strata during longwall extraction on two faces of different widths.

The monitoring indicated that a dome shaped zone of large downward movement extends up into the overburden strata to a height equal to about the panel width.

A major strata unit between 50 m and 70 m above the coal seam influences the behaviour of the overburden strata and may be a factor in the observed sudden loading of longwall face supports. Downward movement of this major unit appears to concentrate on vertical fractures. Increased loading on the face supports could then be expected. The downward movement of this major unit appears to be more significant in the overburden behaviour above the 200 m wide longwall compared to the 160 m wide longwall face.

INTRODUCTION

The longwall panels at Clarence Colliery have experienced sudden weightings on the face that have contributed to production delays. The relationship between overburden caving behaviour and longwall panel width is thought to be a contributing factor to these face weightings. To investigate this relationship more fully, a program of monitoring overburden displacements was undertaken over two longwall panels of different widths using extensometers installed and monitored from the surface (Mills and Gale 1997). This paper describes the results of that monitoring.

BACKGROUND

Clarence Colliery mines the Katoomba seam, the uppermost seam in the sequence. The immediate overburden strata comprises a sequence of competent interbedded fine grained sandstones and siltstones with some weaker coarse grained sandstones. A major sandstone unit occurs at about 25 m above the seam with another major unit some 50-70 m above the seam. The sandstones in each unit are generally massive and free from bedding.

The Clarence lease area has a number of major faults that are generally oriented NNE-SSW. Major joints in the roof are sub-parallel to these faults. There is a conjugate joint set at right angles to the main set. The longwall face orientation is normally 330°. The major joint set orientation is generally 315° to 320° but joint set orientation can vary to become parallel to the face. Both the major and conjugate joint sets are typically vertical and of a smooth planar nature. Clay infilling is common.

Fig. 1 shows the location of the four surface extensometers and two subsidence lines over Longwalls 4 and 5. The first extensometer was installed in the centre of Longwall 4 and was monitored during retreat of both panels. Three more extensometers were installed over Longwall 5 on the same cross-section, one in the centre of the panel and the other two offset 65 m toward each gateroad. Subsidence measurements were made on two cross-lines over Longwalls 4 and 5.

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In the vicinity of the extensometers, the Katoomba Seam is 3.6-3.8 m thick and essentially level. The depth of overburden changes as a result of gently rising surface topography from approximately 240 m at the centreline of Longwall 4 to just over 260 m above Longwall 5. Longwall 4 is 160 m wide. Longwall 5 is 200 m wide.

**Surface extensometers**

Each of the four extensometers had ten anchors located at intervals down the hole, the anchors being more closely spaced near the seam. The uppermost anchor is located at 40 m below the surface. The deepest anchor is located as near to the bottom of the hole as possible. In most cases the deepest anchor was positioned approximately 20 m above the Katoomba Seam.

Fig. 2 shows the surface assembly of the Longwall 4 extensometer with the extensometer head elevated and the cover removed. The three other extensometers were of similar design. Each wire (seen leading into the collar of the hole) is connected to a downhole anchor.

A tension pulley assembly maintains a constant tension on each wire. The tension pulley diameter is calibrated so that one turn equals a set displacement (0.2 m or 0.3 m). Electrical resistance potentiometers register the number of turns on each tension pulley. The extension on each wire is directly related to the resistance registered by the potentiometer. At Clarence, the extensometers were logged using a battery powered electronic datalogger located in a nearby weatherproof container.

Surveying of the holes showed some deviation of the holes from vertical. This deviation was taken into account when analysing the results.
RESULTS

The total movement of the extensometer anchors is the sum of the downhole extension plus the subsidence at the headframe. The vertical subsidence has been added to the downhole extension to give the total displacement of each anchor.

The extensometer on the centreline of Longwall 4 subsided 73 mm during mining of Longwall 4. A further 112 mm occurred during mining of Longwall 5 giving a total of 185 mm. The extensometers at the centreline and on the tailgate side of Longwall 5 subsided 280 mm and 260 mm respectively. The maingate extensometer subsided 80 mm.

Longwall 4 extensometer

Fig. 3 shows the results from the Longwall 4 extensometer. The vertical displacement of each anchor is plotted against depth for various positions of the longwall face.

Downward displacements were first detected on the bottom anchor (23 m above the coal roof) when the longwall face was approximately 15 m past the bottom of the hole.

For the next 15-20 m of longwall retreat most of the movements continued to occur within the first 40 m of overburden strata. During this period, the horizon 40 m above the coal seam moved down only 50 mm, while the bottom anchor moved down 1.2 m.

When the longwall was 44 m past the hole, the face remained stationary for a period of approximately 24 hours. During this period, the first 40-50 m of overburden strata moved downward more or less “en masse” with most of the separation occurring above the 40 m horizon. The bottom anchor displaced a further 300 mm to 1.5 m and the 40 m horizon moved downward 270 mm.

As mining proceeded, downward movements continued to increase in magnitude and progressed higher into the overburden sequence.
When mining was 220 m past the hole, the top 40 m or so of the overburden strata remained essentially unaffected by subsurface movements. This top 40 m bridged across the panel without significant cracking or vertical dilation. The only deformation was a small amount of downward sag deflection.

Fig. 3 - Displacements measure on Longwall 4 extensometer at Clarence Colliery

By the completion of the panel, the 40 m anchor had moved down some 0.55 m while the surface had still subsided less than 80 mm. The unit bridging across the panel at this stage was less than 40 m thick in the centre of the panel.

Fig. 4 shows contours of downward displacements plotted against face advance. The axes are at the same scale so angles are preserved.
There was essentially no movement in the overburden strata ahead of the face. Behind the face (ie: toward the goaf), there was a zone of relatively small movements (20-100 mm) that extended upward initially at an angle of approximately 40° from vertical to 50 m into the goaf and then at a steeper angle to almost reach the surface.

The 200 mm contour represents the line below which downward movements accelerate rapidly. The transition between relatively small displacements (<200 mm) and much larger displacements occurs within a relatively narrow zone. The 1 m displacement contour angles up behind the face at approximately 50° to the horizontal and the 2 m contour at 60° to the horizontal.

By the time that the face was 160-180 m past the extensometer, most of the major movements had occurred in the lower overburden strata as indicated by the flattening out of the displacement contours in Fig. 4. Additional movements occurred mainly in the top 80 m of overburden. These additional movements occurred over an extended period.

The extracted void width of the longwall was 160 m. It would be anticipated, based on geometry alone, that most of the overburden movements would be complete by the time that the face had advanced half the panel width past the site. However, downward displacements continued to occur well after the longwall had passed indicating a dynamic component of movement in addition to the geometry related component.

During mining of Longwall 5, the extensometer at the centre of Longwall 4 showed downward movement of 100 mm. Movement occurred more or less uniformly throughout the full overburden section. This movement is consistent with elastic compression of the chain pillar between Longwalls 4 and 5 causing general lowering of the overburden strata on one side of the panel.

Fig.4 – Longwall 4 extensometer displacement contours versus face advance at Clarence Colliery
Longwall 5 extensometers

Fig. 5 shows the downward displacements measured on each of the Longwall 5 extensometers.

![Graph showing downward movement](image)

**Fig. 5 - Longwall 5 extensometers – displacement versus depth at Clarence Colliery**

The first movement detected on the bottom anchor of the centre extensometer (23 m above the seam) occurred when the longwall face was some 10-15 m past the extensometer. Thereafter, there was an upward progression of displacements similar to that observed in Longwall 4. The progression observed was cyclical. Initial movements were concentrated below a series of parting horizons. The magnitude of movement below each horizon continued to increase until a point when there was downward movement “en masse”. At some stage during the latter stages of this process, a new separation horizon developed higher up in the sequence and the cycle was repeated.

As the cycle progressed higher into the overburden, material lower down was recompressed. The permanent dilation after recompression was of the order of 1 m in 50 m or 2%.

The longwall panels were not wide enough for large downward movements to extend through to the surface. The upper 40 m or so of overburden strata bridged across the panel. The downward subsidence in the centre of the panel was 185 mm when the longwall face was 250 m past. Approximately half of this subsidence was associated with elastic compression of the chain pillar between Longwalls 4 and 5 and the immediate roof and floor strata. The remaining 80-90 mm was associated with downward sag deflection of the overburden.

The displacement mechanism indicated by the maingate and tailgate extensometers is similar to the early stages of movement on the centre extensometer. The height to which downward movement occurred at the maingate and tailgate extensometers was lower than in the centre of the panel. In the centre of the panel, large displacements extended upward some 220 m above the seam to within about 40 m of the surface. At the maingate and tailgate extensometer locations, large displacements only extended upward 130 m and 100 m respectively.
The extensometers indicate movements on the maingate side of the panel are larger and extend higher into the roof strata than on the tailgate side of the panel. It appears as though a separation horizon at 130 m above the seam was mobilised on the maingate side of the panel whereas on the tailgate side of the panel, the separation was concentrated at 90 m above the seam.

A zone of recompression occurred in the centre of the panel in the lower 80 m or so of overburden strata. This recompression zone was not apparent at the location of the other two extensometers.

Fig. 6 shows the contours of downward displacement measured on the central Longwall 5 extensometer. This plot indicates that:

- there is effectively no downward movement in the overburden strata ahead of the face;
- the zone of large downward movements occurs below a line that extends at approximately 35° behind the face;
- the overburden strata moves down in blocks that are defined by discrete separation horizons; and
- the major separation horizons appear to be common to both longwall panels.

**Longwall 5 centre extensometer displacement contours versus face advance at Clarence Colliery**

Immediately below the 200 mm contour, the rate of ground separation increases rapidly as indicated by the close spacing of the 500 mm and 1 m contours.

The overburden strata appears to have moved downward in discrete blocks. A block between 50 m and 90 m above the seam moved downward more or less uniformly throughout its full section. The first major downward movement of the block occurred at about 40 m behind the face. At this point, the 500 mm and 1 m contours are near vertical indicating a zone of large shear displacement on a near vertical fracture surface.

The concentrated downward movement on a single vertical fracture surface is consistent with the history of periodic weighting on the face supports particularly on the wider longwall faces. Downward movements of 1 m in a 30-50 m thick unit located 60 m above the seam would be expected to cause additional loading on the face supports if it were to occur within close proximity to the face.

Three major separation horizons developed at 20 m, 50 m and 90 m above the seam. These horizons coincide with lithological boundaries in the otherwise relatively massive overburden strata.

Fig. 7 shows the zones of large downward displacement inferred from the extensometer measurements for various distances past the longwall face. The edges of this zone are somewhat arbitrarily defined because the downward movements decrease exponentially. For the purposes of discussion, the 200 mm contour has been assumed to represent the edge of this zone.

The zone of large displacement was essentially dome shaped above each extracted longwall panel. The sides of the zone were steeper than the front edge. The front edges extended back from the face over the goaf at about 35° from vertical. The sides extended upward from the chain pillars at approximately 20° from vertical.
Fig. 7 - Zones of large downward displacement above two Longwall panels of different widths at Clarence Colliery

The top of the zone of large displacements was some 1.0-1.1 times the panel width above the coal seam. For the 160 m wide longwall, the top of the zone was 170-180 m above the coal seam. For the 200 m wide longwall panel, the top of the zone was 200-210 m above the coal seam.

The zone of large displacements did not appear to fully develop until the longwall panel had retreated in excess of 160 m past the extensometer. The rate of development was similar for both longwall panels.

The top of the zone of downward movement appears to be a linear function of panel width for the lithological sequence at Clarence Colliery. The intersection of the top of this zone with the surface coincides with the point at which bridging of the overburden strata across the longwall panel ceases.
CONCLUSIONS

The surface extensometers over Longwalls 4 and 5 performed successfully and provide a detailed picture of how the overburden strata behaves during mining of two different width longwall panels.

Downward movements occurred in the overburden strata within a dome shaped zone above each extracted longwall panel.

The zone of movement extended through the overburden strata to a height of approximately 1.0-1.1 times the panel width. The height of movement was greatest in the centre of the panel decreasing on each side nearer to the chain pillars.

Movements within the overburden strata occurred as downward movements of discrete blocks. Separation was concentrated at horizons 20 m, 50 m, 100 m and 130 m above the coal seam.

A major unit between 50 m and 100 m above the coal seam appeared to influence the behaviour of the overburden strata. This unit may be a factor in the observed periodic loading of longwall face supports. Downward movement of this major unit was concentrated on a vertical fracture that appears to be a pre-existing joint.

The major unit between 50 m and 100 m above the coal seam appears to be a more significant factor in downward movement of the overburden strata for wider longwall panel.

REFERENCE


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Prediction of Strata Caving Characteristics and its Impact on Longwall Operation

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ABSTRACT

Recent advances in computer simulation together with field measurements of caving and microseismic activity about longwall panels, has allowed a much better understanding of the caving process and the variability due to geology. The joint research between SCT Operations and CSIRO Division of Exploration and Mining has initiated new methods of computational modelling predicting various caving patterns and strata failure far ahead of the longwall face. This work was validated by field measurements of caving and microseismic activity at the longwall face.

The rock fracture distribution and the caving characteristics of a range of strata sections have been simulated by computer methods. Validation studies of the method were addressed together with case studies. The interaction of caving with support convergence and face control is presented. The method allows the simulation of longwall support behaviour under various geological conditions. The system also allows a prediction of the monitoring data, which is best suited to give an early warning of weighting events or signal various key caving characteristics.

BACKGROUND

The authors have been undertaking research into strata fracture and caving mechanisms about longwall panels to better understand the extent and geometry of rock fractures about longwall faces.

Recent studies of microseismic activity (Kelly, et al 1998) and abutment stress measurements about longwall panels have demonstrated that previous assumptions of caving mechanisms and stress redistributions were either too simplistic or not suited to certain geologies.

This work is being undertaken in conjunction with CSIRO Division of Exploration and Mining in Brisbane. The general scope of the project is that CSIRO undertakes microseismic monitoring to determine fracture location during mining and the authors undertake computer simulation of longwall extraction to predict fracture geometries, stresses, caving mechanics and fluid flow characteristics about longwall panels. Monitoring of longwall support pressures and convergence is undertaken in association with these investigations to assess the interaction of caving and supports under various geologies.

This project was initiated to;

- predict rock fracture patterns about longwall panels;
- understand caving mechanics in differing geologies;
- optimise gas drainage drilling to intersect gas sources and flow networks;
- predict abutment stresses occurring under various geologies; and
- assess longwall support requirements.

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Key sites for the study presented here were from Gordonstone and South Bulga Mines. Recent microseismic monitoring has been conducted at Gordonstone Mine, and computer simulations have been undertaken of these and other sites.

**COMPUTER MODELLING APPROACH**

The aim of the study is to understand the ground caving mechanics under the geological and mining conditions present and the influence of longwall supports in this process. To achieve this, the model must simulate the dynamic caving process as the longwall retreats.

The progressive mining mechanism is achieved by assuming a two-dimensional longitudinal slice down the central zone of the panel and sequentially excavating 1m “shears” in the model. The advancing longwall supports are used to provide roof support at the longwall face. The stress redistributions, rock failure and ground movement then occur in response to an incrementally changing geometry thus simulating a real longwall face.

The finite difference code FLAC (Itasca, 1995) is used to simulate the incremental excavation. A numerical model has been formulated to simulate development of fractures in the bedded strata using the FLAC “fish” routines. The programmable fish routines allow interrogation of the stress state at any point of the model and the determination of the type of fracture that may develop. Various failure models are used to predict the type of fracture, orientation and its properties. The rock failure routines calculate the likelihood of shear and tensile fractures through intact rock and shear and tensile failure along the bedding. The fractures are simulated by changing the rock and joint properties to model the strain softening behaviour of rock derived from the triaxial rock testing.

The rock failure and permeability routines used in the code have been developed by the authors to realistically simulate actual strata behaviour. In this study emphasis is on the rock failure, caving characteristics of the ground and longwall support behaviour. The computer simulation capability is being refined and validated as an ongoing process in association with CSIRO Division of Exploration and Mining researching longwall caving mechanics and other research projects undertaken by the authors.

Although the computational model is two-dimensional, the third dimension can be approximated by mining the distance equal to one half of the longwall face width from the reflective boundary located on the goaf side. This method allows longwall simulations of subcritical width with the front abutment stresses similar to those measured underground.

The properties of strata used in the model are based on triaxial testing of overburden material. A typical section of the modelled strata is shown in Fig. 1. An enlarged portion of the longwall face is presented in Fig. 2.

The model of the longwall supports is constructed using the grid and support elements. The stiffness of the canopy and base parts is chosen to approximate the properties of the actual longwall supports. The modelled supports have the ability to advance forward and reset each time the coal is cut. The set loads are gradually increased to the yield value in response to the support convergence. The support loads are monitored and can be compared with the leg pressures measured underground.

The goaf behind the supports is allowed to free fall a nominated distance to reach the zone where a convergence induced vertical load is applied to the goaf roof. The vertical load is gradually increased until the full goaf load is experienced at a nominated convergence above floor level.

The progressive excavation of the longwall panel and associated ground response can be captured in a "movie" file, which allows visualisation of caving cycles and stress changes as the longwall retreats.

**MODELLING OF STRATA OF VARIABLE STRENGTH**

Vastly different caving styles have been defined in this project. Two examples are presented to demonstrate the variability in caving as a result of rock strength properties and stressfield.
Fig. 1 Typical section of the modelled strata

Fig. 2 Enlarged section of the Longwall face
Example 1 - Weak strength ground - forward ground failure

The properties of strata used in this model are based on the overburden rock at Gordonstone Mine.

The longwall fracture distribution presented in Fig. 3 indicates rock failure well in advance of the face. This style of behaviour has been verified by microseismic monitoring but would not have been predicted by traditional approaches. In this caving style, no large caving blocks are formed and periodic fractures only occur on the small scale as the ground is heavily fractured in front of the face. The peak stress concentrations are located well ahead of the longwall face, while the ground is de-stressed in the vicinity of the longwall face. The roof failure mechanism is characterised by the formation of frequent subvertical fractures and sheared bedding planes which develop after each shear has been cut. On a large scale, the roof failure in weak strata can be described as non-periodic.

![Fig. 3 - Longwall fracture distribution - weak ground](image)

Example 2 - Moderate strength ground - cyclic caving

The properties of strata used in this model are based on the overburden rock at South Bulga Mine.

A very different caving and fracture mechanism is presented in Fig. 4. The absence of weak bedding planes in the upper roof and moderate strength of rock prevents frequent formation of fractures in the roof. Major sub-vertical fractures develop at less frequent intervals forming large caving blocks above the longwall face. The geometry of these blocks is defined by;

(i) failure along a weak layer in the roof above or ahead of the face followed by; and

(ii) a fracture network forming at this zone and extending down to meet the longwall face.
Face guttering, rib spall and convergence of supports is anticipated to be most severe where fracture systems intersect the face. In these geologies, significant fracturing above the longwall face and supports may only occur every 10-20m. The caving mechanism observed in the model was compared to the overburden movement measured by an extensometer extending from the surface down to the coal seam. The extensometer was located at the centre of the longwall panel. The surface extensometer results presented in Fig. 5 indicate a good correlation of strata movement between the model and the in situ measurements.

Underground monitoring of the longwall support pressures and convergence shown in Fig. 6 were used to study the frequency of periodic weighting. The results were directly compared with the leg pressures and the convergence of the modelled support. The underground data and the modelled results were showing similar trends.
Fig. 5(a) – Comparison of modelled and measured extensometer results in the 7 – 15m range behind the face area

Fig. 5(b) – Comparison of modelled and measured extensometer results 20m behind the face area
SELECTION AND OPERATION OF LONGWALL SUPPORTS UNDER VARIABLE ROOF STRATA

Prediction of premature caving at canopy rear (case study)

A computer model was constructed to simulate weak strata with roof consisting of bedded mudstone and siltstone. The geometry of the model and the strength of intact strata is shown in Fig. 7. The rock properties typical of weak mudstone and siltstone strata are presented in Table 1.

The strength reduction factor of 0.58 was used in the model to approximate the in situ rock strength.

The aim of the model was to simulate the premature roof caving to study the response of the longwall supports to strata movement. The coal was sequentially mined and the exposed roof supported by advancing 4-leg powered supports rated at 650 tonnes.

The study showed that the 4-leg supports failed to provide adequate support to the broken rock at the canopy tip. When the rear legs were set at 80% of their design capacity, premature yielding of the fractured roof at the rear of the support caused the canopy to rotate slightly and lower the canopy tip away from the roof. Low confining stress zones at the canopy tip can be seen in Fig. 8a where contact between the canopy tip and the roof was lost. When the rear leg pressures were reduced, the stress distribution above the canopy improved with better roof confinement at the canopy tip.
When trialing the 2-leg supports rated at 650 tonnes, the premature caving at the canopy rear appeared to have little influence on stress distribution above the canopy. The geometry of the 2-leg supports and the stress distribution in strata showing higher confining stresses at the canopy tip is shown in Fig. 8b. The model indicated that the 2-leg supports provided better roof support and desirable confining stress at the canopy tip.
Table 1. Rock Properties used in the Model

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<tr>
<th>Rock Type</th>
<th>Mudstone</th>
<th>Coal</th>
<th>Siltstone</th>
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<tr>
<td>Maximum tension (MPa)</td>
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<table>
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<th>Residual Stress (MPa)</th>
<th>Intact Stress Range (MPa)</th>
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<td>58</td>
<td>100-140</td>
<td>75</td>
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</table>

Fig. 8 - Distribution of maximum principal stress indicating the level of roof confinement at the longwall face for 2 and 4 leg supports set at 650 tonnes
In summary, the model showed that premature caving at the canopy rear may reduce the overall support capacity, contribute to the canopy tilt and cause unwanted reduction of roof support at the longwall face potentially leading to roof failure.

The model further indicated that for strong overhanging immediate roof, 4-leg supports can provide a more favourable reaction to the roof. If the immediate roof caves readily or premature caving at the canopy rear is anticipated, then 2-leg supports will provide better roof control.

**COMPARISON OF METHODS TO SELECT POWERED SUPPORTS**

**Role of the powered support**

The underground observations, computational modelling and microseismic survey provide evidence that:

Fractures in the strata are induced by stress concentrations ahead of the longwall face. Longwall supports have little influence on the formation of fractures within the roof. The role of longwall supports is to assist already broken roof strata to remain reasonably intact until caving occurs behind the canopy. If the support canopies do not exert enough force onto the immediate roof, opening and displacement along the mining induced fractures can occur reducing the integrity of strata and affecting stability of the longwall face.

**Longwall support capacity requirements**

Historically, the increase in load capacity of the powered supports in Australia coincided with better production rates and fewer strata problems at the longwall face. In search of further increase in production, the mine management urged the longwall manufacturers to gradually increase the capacities of the shield supports. This ongoing trend is restricted by the cost of the supports and the potential wastage if the support geometry or load capacities are not suitable for the strata conditions.

Selection of the longwall supports has traditionally been determined on the basis of:

- Minimum force required to support the immediate roof;
- Additional loading of the roof;
- Unexpected loads when negotiating geologically disturbed zones;
- Increased loading during the longwall recovery;
- A margin to cover shortfalls in the hydraulic system health; and
- Previous experience in the mine.

As demonstrated, current development in computer simulation techniques can provide more appropriate selection of the powered supports. The computational model is also more suitable to indicate whether the further increase in the support capacity would provide any benefits at the longwall face.

**USING THE COMPUTATIONAL MODEL AS THE LONGWALL SUPPORT DESIGN TOOL**

The traditional method of longwall selection allowed for basic designs only while the computational models backed by the underground measurements can provide a wider range of solutions to select appropriate powered supports.

The computer model has the unique ability to compare how different powered supports cope with identical mining conditions. Repeating the study for all possible conditions that can be encountered in the mine allows the best support type to be selected to suit the prevailing underground conditions.

To show the diversity of the problem that can be solved using the model, some are summarised below:
The computational models can:

- Provide information to assist with the selection of new powered supports;
- Evaluate the effectiveness of existing powered supports;
- Estimate the minimum safe site specific support loads to control roof strata;
- Assist in selecting the best mining method;
- Estimate the maximum safe roof exposure at the face;
- Estimate roof stability when using one web back system;
- Predict likelihood of premature roof caving at the canopy rear and indicate the best operational techniques to minimise the canopy tilt;
- Show the influence of the longwall panel width on strata behaviour;
- Indicate strata behaviour when change in geology occurs;
- Assist with prediction of periodic weighting;
- Indicate the possibility of major roof failure;
- Help to predict strata behaviour when negotiating faults;
- Assist with roof reinforcement design at the longwall finish line;
- Predict the extent of fracture propagation in the roof at the longwall face; and
- Study the behaviour of failed roof.

CONCLUSION

This work is “breaking new ground” in understanding and demonstrating techniques to predict ground behaviour about longwall panels. The results of this work are being applied to support design at a number of mines, and demonstrate the benefit of diverse skills within the collaborative research team. The up-to-date results from the model have provided a new understanding of how the longwall supports respond to different strata and stress environments. This would not be possible using the more conventional approaches to longwall design. The study demonstrates that the computational modelling can provide accurate predictions of strata behaviour at the longwall face leading to greater safety, improved costs and higher production levels. The longwall model has proven to be of significant value and further development is envisaged to provide better service to the mining industry.

REFERENCES


Bearing Plates: New Developments in the Unsung Heroes of Ground Support

P Gray

ABSTRACT

The trend in the mining industry over the past ten years has been to use higher capacity ground support systems. Much of the engineering design and development for these high capacity ground support systems has focussed on new, more effective, and stronger rock bolts and cable bolts. However, higher capacity rock bolts or cable bolts are only part of the solution to ground support problems, and it is now apparent that a total systems solution is required which addresses such issues as: speed of installation; safety; cost; rib support as well as roof support; and, the interaction of different ground support products. This last factor has received very little attention in the past, and applies particularly to the interaction of bolts, mesh or straps, and, bearing plates. Bearing plates are a fundamental and integral part of any rock support system. The capacity of bearing plates should be matched to the capacity of the rock bolts or cable bolts that they are used with, and they should be designed not to damage other support systems such as mesh or straps. This paper gives examples of both good and bad bearing plate installations, and indicates trends for the future.

BACKGROUND

The original function of bearing plates was to retain and support the rock immediately underneath the bearing plate, and to prevent the nut or forged head on the end of the rock bolt from being “pulled” into the rock bolt hole. Early bolts used in the underground coal industry were point anchored “slot and wedge” bolts, and these were used with basic bearing plates which were simply flat, square pieces of steel with a central hole to accommodate the rock bolt. Subsequently, a wide variety of different types of bearing plates have been developed for rock bolts and cable bolts for the mining and tunnelling industries. These have included square and round plates, and even triangular plates have been developed. Today, most bearing plates also accommodate uneven roof conditions by allowing angular movement between the bolt and the plate, either by:

- using a domed ball which will rotate around the central hole in the bearing plate; or by,
- using two hemispherical plates which will rotate over each other; or by,
- using a deformable plate that will distort to accommodate such angular movement.

Most bearing plates are pressed plates, but thicker plates (>15mm thick) are normally machined and/or fabricated. The majority of bearing plates used in the coal industry are square domed plates which use a domed ball to allow angular movement, and are typically 8mm, 10mm or 12mm thick. Square domed plates are simple to manufacture, have a minimal steel wastage, and their bearing capacity can be increased by making them thicker.

Recent trends in ground support

The trend in the mining industry over the past ten years has been to use higher capacity ground support systems including rock bolts and cable bolts. The reasons for this are firstly, that ground conditions have become more difficult in many mines; secondly, that the “materials cost” of bolts is much less than the “total installed cost” of bolts and therefore there is an advantage in using fewer, but stronger bolts; and lastly, that mining methods and systems have resulted in higher stresses in many mines necessitating higher capacity support systems.

1 BHP Engineering Pty. Ltd. Wollongong
In addition, the trend for long tendon support using cable bolts, and the increase in pre-tensioning for both cable bolts and rock bolts, has resulted in greater loads being generated at the collar of the bolt and hence onto the bearing plate. However, much of the engineering design and development for the ground support industry has focussed on new, more effective and stronger rock bolts or cable bolts but there has been less engineering design or analysis undertaken for bearing plates, and yet they are a fundamental and integral part of any rock support system.

It is also now becoming clear that new rock bolts or cable bolts are only part of the solution to ground support problems, and it is apparent that a total systems solution is required which addresses such issues as: speed of installation; automation; safety; cost; rib support as well as roof support; and, the interaction of different ground support products. This last factor has received very little attention in the past, and applies particularly to the interaction of bolts, mesh or straps, and, bearing plates.

This paper is concerned with the performance of bearing plates, and outlines some design criteria that should be used for bearing plates.

**THE TOTAL INSTALLED COST OF ROCK BOLTS**

The cost of ground support is a significant investment for any mining operation. This investment is much greater than the “total materials cost” of rock bolts alone, and is closer to the “total installed cost” of rock bolts. However, if roof falls occur, then the total cost of ground support could be much higher than the total installed cost of rock bolts alone (ie costs to clean-up the fall, costs of delays etc). Galvin and Pallas (1997) estimated that there are 10 major falls in the underground coal industry each year in Australia with a total cost approximately $10 million for each fall. In addition, there are costs associated with injuries sustained during roof falls or when cleaning up roof falls. It is therefore vital that the investment in ground support be optimised, such that the lowest overall ground support cost can be achieved.

It is important to recognise what the total installed cost of rock bolts actually is. Firstly, a basic rock bolt itself costs only about $8 to $12 depending upon length and type (this excludes cable bolts and specialist high strength bolts). Secondly, the costs of the other materials required to go with a rock bolt are approximately another $11, eg a bearing plate, a resin cartridge, and a proportional cost of: mesh or straps; drill rods; and drill bits. This brings the total materials cost to a minimum of $19 per bolt.

In addition, there are the costs of labour to install the bolt (approximately $6 per bolt, but can be much higher particularly for cable bolts). Moreover there are the costs of the equipment to install the bolt such as depreciation and maintenance costs (estimated at a minimum of $2 per bolt), but excluding running costs. Finally, there are the costs associated with the delays caused by rock bolting (ie the cost of the lost production caused by rock bolting and this will vary between different mines but is approximately $75 per bolt). These costs are shown in Fig. 1 and gives the total installed cost as $102 per bolt!

![Fig. 1 - Pie Chart showing the Total Installed Costs of Rock Bolting](image)
Note: The assumptions made above in relation to the Total Installed Cost of Rock Bolts are conservative. Different assumptions made in relation to the number of bolts and the total time taken to install bolts, can increase the total installed cost significantly.

This basic model was re-run several times for different mines with different assumptions and the range in costs was from $80 per bolt to nearly $300 per bolt. Therefore the total installed cost for a $8 to $12 rock bolt is approximately $100 each (ie 10 times the cost of the bolt itself and 5 times the total materials cost). The obvious conclusions are that:

- the cost of the bolt itself is insignificant compared to the total installed cost;
- installing fewer, stronger (but more expensive) bolts would save significant costs;
- installing the same number of cheaper bolts would have minimal cost savings;
- installing more (but less expensive) bolts would dramatically increase costs;
- it is essential to achieve the maximum performance from each bolt installed by using the correct installation procedure and with the use of matched components.

This last point applies to all components used for the installation of rock bolts including resin, straps or mesh and, importantly, bearing plates. Bearing plates are particularly important because they have the potential to damage other components of the rock bolt system, and ultimately cause failure.

THE DESIGN AND PERFORMANCE OF BEARING PLATES

The performance of bearing plates should be considered in terms of their:

- Ultimate load capacity;
- Load/Deformation characteristics;
- Interaction with other support components (bolts, cables, straps, mesh etc);
- Interaction with the roof rock mass; and,
- Ease of installation.

In addition, the design of bearing plates should consider the cost of manufacture.

Ideally, bearing plates should provide stiff confinement to the rock mass immediately under the collar of the rock bolt, as well as between rock bolts. This would require bearing plates to be very thick, very large, and be linked together with each other. In reality, this is impractical but the use of mesh and straps can help to support the rock mass over a larger area than is possible with bearing plates alone. Therefore the design of the bearing plate needs to be considered in relation to the other support systems it is used with (eg mesh, straps, etc).

Matching bolts and plates

In addition, bearing plates themselves should be designed to have the same ultimate load capacity as the bolts they are used with. For example, a common high strength rock bolt may have an ultimate tensile capacity of approximately 30-33 tonnes, but these are almost always used with bearing plates with an ultimate bearing capacity of only 18-20 tonnes. This is equivalent to using rock bolts which have half of their section machined away at the collar of the bolt (such a bolt is shown
in Fig. 2). It would be unthinkable to use the rock bolt that is shown below, but many local mines do exactly that by using high strength rock bolts with bearing plates that only have half the capacity of the rock bolt.

![Rock Bolt Image]

**Fig. 2 - Most mines use rock bolts that have the strength equivalent to the bolt shown above.**

This is analogous to using a chain with one weak link. From an engineering and safety aspect one would never use a chain with one weak link, so why do we do it in the ground support industry? One argument that supports this point of view is that for fully encapsulated bolts, high collar loads are not generated and therefore high capacity bearing plates are not required. However, geological and stress conditions can never be absolutely determined. In addition, the installation of bolts and the mixing of resin anchors is never 100% reliable. Therefore it is possible to have high collar loads on bolts even with fully encapsulated bolts in mines with good roof conditions.

Fig. 3 shows a common square bearing plate that has failed completely by splitting and turning inside out even though the bolt itself has not failed.

![Bearing Plate Image]

**Fig. - 3 Bearing plate failed and turned inside out**

Finally, the capacity of the bearing plate should not only be matched to the capacity of the rock bolt, but they should also be designed to fail progressively. They should not fail catastrophically as shown in Figs. 3 and 4.
Consequently, from both an engineering and safety point of view, it is no longer acceptable to use bearing plates which only have half the capacity of the rock bolts they are used with.

The importance of a “soft footprint”

Moreover the bearing plate should be designed not to damage mesh or straps or other support services. Square bearing plates often have sharp corners or edges, and when roof movement causes straps or mesh to be bent over this sharp corner or edge, then this can trigger strap or mesh failure. This sharp edge acts like a guillotine. This is a fundamental design flaw with most current square bearing plates. An example of a square plate cutting through a strap is shown in Figs. 5 and 6.

Fig. 5 shows that although the square bearing plate has cut through the W Strap, the plate itself is not deformed indicating that the load on the bearing plate was low when failure of the W Strap occurred. This clearly illustrates that square bearing plates with sharp edges act like a guillotine and are a bad design.
Fig. 6 - Square bearing plate has cut through and failed the W strap and then failed completely itself and fallen off
the rock bolt.

Fig. 6 shows complete failure of the W Strap and the bearing plate, and in this case there is virtually no restraint to the roof
at the collar of the rock bolt. The difference between Figs. 5 and 6 is that Fig. 6 has a higher collar load than Fig. 5, and has
consequently caused more damage to the roof support system.

Bearing area

Furthermore, with higher and higher collar loads being generated on bolts and bearing plates due to point anchoring or long
tendon support, it is important to ensure that the bearing plate can withstand these loads and that the bearing plate does not
 crush or punch through the rock underneath it. This often occurs where there is a weak rock in the roof (such as a coal
roof). This is shown in Fig. 7.

Fig. 7 - Thick, square bearing plate has punched through the immediate coal roof.

A massive, thick bearing plate used with a high strength cable bolt, is shown in Fig. 7. In this case, the bearing plate is very
strong, but is of little use because this massive bearing plate will not deform along its edges and thus it can cut through
anything underneath it including mesh or W Straps or even the roof itself. This is particularly important with high strength
cables where high collar loads are often generated.
For example, if we take the case of a weak coal roof with an intact unconfined compressive strength (UCS) of 10MPa, this translates to a rock mass strength of only approximately 4 MPa depending upon what assumptions are made regarding the rock mass. If we further assume a stiff bearing plate of dimensions 150mm x 150mm on the end of a point anchored 60 tonne cable bolt, then it can be shown that the bearing plate could punch through the coal roof at a theoretical load on the plate of only 31 tonnes. In practice the collar load to cause the plate to punch through the roof would be less than this because the edges of the plate would cause the roof to break up around the plate (as shown in Fig. 7).

One way to avoid plates punching through the roof is to make the bearing area of the plate larger. However, there is a practical limit to the size of standard thick plates due to weight problems (maximum size typically 200mm x 200mm x 20mm). Therefore to increase the bearing area, the use of large, lightweight plates should be used in conjunction with standard plates. Also the use of mesh and straps can be used to increase the bearing area of the plate and thus reduce the contact pressure on the rock.

Mesh and W straps

W Straps and mesh help to support the rock between rock bolts, and as mentioned previously can help to increase the bearing area of the bearing plate. In the case of mesh, the initial contact area of individual wires is very small, hence the bearing pressure on the base of each wire is very high, and therefore the mesh underneath a bearing plate can be pushed into a weak roof relatively easily. This assists to lock the mesh wires in place underneath the bearing plate. It is also important to note that failure of straps or mesh nearly always occurs at the bolt or plate contact point. In the case of straps, failure often occurs by the plate tearing into the strap (see Figs. 5 and 6). In the case of mesh, failure often occurs by the plate cutting into the wires, thus weakening them and initiating failure. Failure of straps or mesh due to steel tensile failure rarely occurs.

Large, lightweight plates

These plates have been around for many years, and are made from thin pressed or rolled steel. They are a useful adjunct to standard bearing plates because they help to spread the load over a large surface area, but at the same time they have “soft” edges which are relatively easily deformable under load. Hence they are less likely to cut or damage mesh or straps or the roof.

The performance of square bearing plates

As mentioned previously, the most common plates used in the mining industry are square domed plates used with rock bolts, and square flat plates used with cable bolts. The square domed bearing plate is a simple design which is cheap to manufacture. These plates are square which minimises steel wastage during manufacture, but also produces sharp edges and corners which can damage other support services such as mesh or straps. In addition, the thicker the standard square plate becomes, then the edges and corners also become stiffer, and are more likely to cut straps or mesh than thinner plates. Consequently, making a square plate thicker in order to make it stronger and stiffer, is self defeating because the corners and edges are more likely to damage other support services under high load conditions.

Moreover, the hemispherical washer which contacts the standard bearing plate around the domed central hole, has a bearing area which is only a tangent point, and this generates very high contact stresses as the load is increased. The plate will deform under load by the central dome being flattened, and the corners of the plate being upturned away from the roof. At ultimate failure the hemispherical washer can pull through the central hole causing complete failure, and or, the plate can split into two pieces. (see Figs. 3 and 8).
Fig. 8 - Square, domed bearing plate showing how the domed ball punches through the central hole and the corners turn up at failure

The ultimate load capacity of square, domed bearing plates depends upon the plate thickness and the grade of steel used. Approximate capacities are as follows:

- 8mm thick plates: 15-17 tonnes
- 10mm thick plates: 25-28 tonnes

However, the ultimate load capacity of a bearing plate does not indicate its performance, and other plate characteristics also need to be considered, as mentioned previously. Nevertheless the load deformation characteristics for an 8mm thick square, domed bearing plate are shown in Fig. 9.

Fig. 9 - Load / deformation characteristics for an 8mm thick square, domed bearing plate

20mm thick flat bearing plates

In contrast to square domed plates, very thick square plates are too thick to press out in a conventional stamping operation and hence the plate is not “domed” to accommodate a hemispherical washer. They are simply made from flat square steel plate and a central hole is then punched or drilled through the plate to form a bearing plate. These plates are very stiff and have a high ultimate load capacity, but they are also very heavy and expensive. They not only have sharp corners and edges, but these edges are very stiff, and therefore form an excellent cutting edge for anything under the plate. (See Figs. 10 and 11).
Fig. 10 - 20mm thick square bearing plate showing that there is very little deformation even up to very high loads

Fig. 11 - Load / deformation characteristics for an 20mm thick and square bearing plate

Very thick square bearing plates are normally used with high capacity cable bolts, and these tend to create high collar loads on the bearing plate.

Bearing plate design criteria

The preferred bearing plate design criteria should avoid the problems described above with square bearing plates. Design criteria are as follows:

- the compressive strength of the bearing plate should match the tensile strength of the bolt that it is used with;
- the bearing plate should fail progressively and it should not crack or split and fail suddenly;
- the bearing plate should not damage other support services such as mesh or straps.

One bearing plate which fulfils the above design criteria is the Cup and Saucer bearing plate. It has a round design with no sharp edges or corners, and has a "soft" footprint. The footprint is initially small, but increases under load as the plate flattens out (see Fig. 12).
Fig. 12 - Cup and saucer bearing plate showing that the plate flattens out under high load.

The load deformation curves for the Cup and Saucer plate are shown in Fig. 13. It can be seen that the plate fails progressively, and even at extreme failure, the plate simply flattens out completely and would then begin to re-load up again. It will not fail catastrophically.

Fig. 13 - Load / deformation characteristics for a cup and saucer bearing plate.

The Cup and Saucer bearing plate also interacts very well with other ground support components, such as W Straps and mesh. In the case of W Straps, the plate sits neatly between the ridges of the strap. If movement of the roof causes the strap to bend over the edge of the plate, the "soft" edge and footprint of the plate will not cut into the strap.

In the case of mesh, the small, soft footprint of the Cup and Saucer will pinch the line wires of the mesh without cutting them, and will lock them in place (In contrast, standard square plates allow the wires to slip under the plate until the rock bolt cuts into the wires).
FUTURE TRENDS

Bearing plates are now recognised as playing an important role in ground support, and there is a wide variety of bearing plates available on the market. Bearing plates range in size from approximately 90 mm diameter to rectangular plates nearly 500mm x 400mm. They range in thickness from 1.5mm thick steel to over 20mm thick steel and even to fabricated plates with a welded structure.

Plates are made from steel, hardwood, plywood, and plastic, with these three last materials commonly being used for rib bolting where the plate is eventually mined through by the continuous miner. Some bearing plates for roof bolting now include attachments for cable slings, and others include attachments to hang services from the plate directly. Various load indicator mechanisms are also incorporated in some plates, but at present, are all relatively inaccurate. In addition, steel mesh is now available that is specifically designed with a reinforced section to accommodate bearing plates.

The trend is clear that there are an increasing number of bearing plates available on the market, and this trend is likely to continue. Although the mining industry recognises that it must reduce its costs, it is also becoming aware that bearing plates must work effectively to be of any value. Given the very high "total installed cost" of rock bolts (approximately $100 each), bearing plates must maximise the performance of every rock bolt, and that square plates with sharp edges and corners may no longer be good viable or cost effective.

CONCLUSIONS

The actual performance of a support system will be different to that as tested in a laboratory, and the in-situ performance of any support system should always be closely monitored. Nevertheless, it can be concluded that:

1. The effectiveness of the ground support system will depend upon the interaction and performance of all components of the ground support system, including bolts; resins; straps or mesh; and bearing plates.

2. The total installed cost of rock bolts for most underground coal mines is at least $100 each. Therefore it is important that
   - all the components of the rock bolt support system work well together;
   - that they are matched components; and,
   - that they should not damage other components of the support system.

3. Bearing plates which use a domed ball to accommodate angular movement, concentrate the load around the central hole, and often fail catastrophically by the domed ball pulling through the central hole. This applies to most plates which use the domed ball principle.

4. Bearing plates which have sharp edges and corners or have serrated edges, have a greater potential to damage mesh or straps than plates without these features. In addition they can cut through W Straps with only a small collar load on the bearing plate. Very thick, square plates are even worse than thinner square plates because their edges are stiffer, and they form a very effective guillotine.

5. Even for fully grouted bolts, it cannot be guaranteed that high collar loads will be not be generated once roof movement occurs. Therefore the bearing plate capacity must be matched to the capacity of the rock bolt or cable bolt. From both an engineering and safety point of view, it is no longer acceptable to use bearing plates which only have half the capacity of the rock bolts they are used with.

6. Bearing plates should ideally have a "soft" footprint with no sharp edges or corners (ie be round), and have a capacity which is matched to the other components of the support system.
REFERENCES


Experience with Stress Control Methods as a Planning Tool - Elouera Colliery

G Tarrant¹, W Huuskes² and P Quinn¹

ABSTRACT

Continued extraction of the No. 3 Seam at Elouera Colliery required the driveage of a sequence of roadways within an area of significantly elevated horizontal stress. Initial roof conditions varied from excellent to very severe as the panel entered the high stress area. The variation was attributed to the extraction sequence, with different roadways “leading” the panel into the high horizontal stressfield.

This paper summarises field and analytical work designed to evaluate the use of stress control methods as an effective planning tool. The “leading” roadway of the panel would be required to remain stable whilst experiencing moderate to severe deformation whilst the neighbouring roadways (stress relieved) could be driven at more acceptable advance rates using less reinforcement.

The planning implications of developing the panel with stress control methods are discussed.

INTRODUCTION

The use of stress control methods to optimise extraction and reinforcement systems was evaluated for a panel to be driven through a high horizontal stress area at Elouera Colliery.

The investigation program included a routine monitoring program using tell-tales, supplemented with extensometry at key locations. The analytical work included the use of a numerical model developed to simulate roadway and reinforcement behaviour over a range of conditions.

This paper outlines the approach adopted and incorporates the practical implications of using stress control provided by colliery personnel.

BACKGROUND GEOTECHNICAL INFORMATION

Panel layout

The panel layout shown in Fig 1 illustrates the extent of deformation within Thompson and Hume Panels prior to the investigation, the area over which field and analytical work was conducted and future roadway driveage.

Geological setting

The Colliery mines the No. 3 Seam under a depth of cover of approximately 350 m over Thompson Panel. A roof core obtained from the investigation area is shown in Fig 2. The roof up to 6 m is composed of a sequence of coal and carbonaceous mudstone with various sandstone and clay (tuff) units present.

A sandstone unit approximately 1 m thick is located between 2.4 m and 3.4 m into the roof. This unit overlies a weak claystone material, locally known as the “puggy band”. A further weak claystone unit is also present at the top of the seam. The weak claystone interfaces play an important role in the mechanism of stress redistribution about the roadways.

A major fault forms the western boundary of the panel as shown in Fig

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Fig. 1 - Elouera Colliery panel layout roadway deformation, layout and inferred stress direction

Fig. 2 - Elouera Colliery Geological profile – Thompson Panel
Stressfield

Initial driveage of Thompson and Hume Panels were inspected to determine the orientation of the major horizontal stress on the basis of the deformation mechanisms observed. The relative magnitude of the horizontal stress could also be gauged by the level of deformation evident.

The results of the mapping are shown in Fig 1, which illustrates a strong correlation between the increased horizontal stress levels with proximity to the major fault. The major horizontal stress direction inferred from mapping was west northwest - east southwest, approximately perpendicular to the fault.

The future panel geometry would require a panel to be driven in more unfavourable circumstances than that already experienced (headings aligned almost perpendicular to the major horizontal stress direction).

General concept of stress control

The general concept of stress control methods is well understood and has been previously outlined by Gale et al (1990). In summary, the extraction sequence for a panel of roadways comprises a leading roadway with respect to the horizontal stressfield. This roadway is subjected to the pre-mining horizontal stress and experiences the most deformation. An area of reduced horizontal stress is formed about the leading roadway, providing more favourable driveage conditions. The general concept is illustrated in Fig 3. Stress control is most appropriate where roadways are aligned perpendicular to the major horizontal stress.

![Diagram](image)

**Fig. 3 - Slow build up of vertical stress in the pillar where slip occurs and confinement is reduced**

**MONITORING PROGRAM**

The layout of instrumentation is shown in Fig 4.

Tell-tales are used on a routine basis by Elouera Colliery to ensure that the roadway deformation levels are within acceptable design limits. In areas where the roof movement exceeds a predetermined level, further action, usually in the form of long tendon secondary reinforcement is used.
The monitoring program supplemented the routine tell-tales with more detailed extensometry at key locations.

**General style of roof deformation**

Fig 5 is an extensometer which shows the typical style of deformation at Elouera Colliery. Initial movement is restricted to the immediate roof below the competent sandstone unit at 2.4 m to 3.4 m into the roof. Deformation occurs as both slip along weak interfaces, particularly the “puggy band” and shear failure of the intact material itself through overstressing.

The propagation of deformation is characterised by softening of material above the sandstone, followed by shear failure of the sandstone itself. At this stage in the deformation pathway, softening has occurred up to 6 m into the roof and secondary reinforcement is typically required to maintain roadway stability.

**Roof monitoring results**

‘D’ Heading was the first driven roadway in this sequence. It was driven with an ABM20 and supported with mesh modules and 6 x 2.7 m bolts at 1 m centres. Whilst the miner achieved good driveage rates of 11 m per shift with good (visible) initial roof conditions, the roadway eventually deformed over 300 mm and required cable bolts along its entire length.

Flanking roadways, ‘C’ and ‘E’ Headings were driven without problems, experiencing less than 40 mm of roof movement. ‘C’ Heading has experienced approximately 13 mm of movement to date.

A summary of the total roof movement for each heading within the sequence for Thompson and Hume Panels is shown in Fig 6. The comparative roof conditions between ‘D’ Heading and ‘C’ Heading is also shown in the photograph taken outbye 24 Cut-through (Fig 7). Note the extensive use of cable bolting in the leading roadway.
Driveage was continued inbye 24 line with a heliminer with ‘E’ Heading the leading roadway. After reaching 45 m the miner was taken out and the heading was chocked due to the severe roof deformation. ‘D’ Heading was then driven approximately 25 m in advance of ‘E’ Heading at which point the miner was again taken out due to severe roof conditions.

Extent of stress relief

A contour diagram of inferred stress relief is presented in Fig 8. The stress relief zone can be visualised as a bow wave about a boat. The leading roadway itself is always pushing into an area of increased horizontal stress, leaving a wake of reduced stress.

One of the major contributing factors to the mechanism of stress relief is considered to be the slip along the weak interfaces such as the “puggy band”. The weak interface can be visualised as a surface on which a series of ball bearings rest. Removal of confinement (mining the roadway) allows the material below the “puggy band” to mobilise towards the opening, effectively moving along the ball bearing surface. In a frictionless world, the movement would propagate forever. At Elouera, the movement propagates at least 50 m.
PLANNING IMPLICATIONS

It was decided to review the future layout and support design for the area as it was considered too unsafe to mine the planned layout. The roadways were essential to the viability of the mine, being the main development leading to the last remaining longwall blocks. Concerns about longwall continuity meant that they had to be driven quickly.

The main planning implications included:

- Driveage sequence;
- Reinforcement system required to manage the deformation of the leading roadway; and
- Implementation of new support systems.

Driveage sequence

In determining the new panel layout, the purpose of the leading roadway had to be considered. Allowing total collapse of the leading roadway (sacrificial heading) was considered but would have altered the layout too much and would have been difficult to drive safely.

A four heading layout with flanking returns was decided. Since 'D' Heading was a return and the belt road ('C' Heading) required the most protection, it was decided to drive 'D' Heading 20 m to 30 m in advance of 'C' Heading.

If the new support system failed in 'D' Heading, then timber support could be used and the heading still serve as a return, although less efficiently if timbered. It was expected that 'C' Heading would then be driven under much more favourable conditions with only primary reinforcement.

The sequence of driveage required to maintain 'D' Heading as the leading roadway was not the optimum driveage sequence for the panel. Driveage of the belt road first would have allowed more time to construct the necessary infrastructure for the next longwall.
Fig. 7 - Elouera Colliery comparative roof conditions between leading roadway (D) heading and flanking roadway (C heading)
Leading roadway stability

Analytical study

To determine the reinforcement required to maintain stability of the leading roadway, the numerical model developed to simulate roadway behaviour and reinforcement performance was used. The model is regularly used to evaluate various reinforcement options over a range of geological and loading environments.

The panel driveage coincided with the development of two new products, the “Megabolt” and “High Performance Capacity” (HPC) bolt. The Megabolt is a 65 tonne “caged” wire product which is marketed as an alternative to cable bolting. Its main advantage is seen to be the ability to be installed at the face. The HPC bolt is a 50 tonne solid bar product marketed as an alternative to the 30 tonne “X” grade rebar.

Whilst a complete description of the modelling work is beyond the scope of this paper, in summary, an interim design of three megabolts per linear meter was presented to the colliery, to be installed at the face in addition to the existing primary bolting pattern. The roof would still be expected to soften to the top of the seam (6 m) and between 100 and 150 mm of roof displacement was anticipated. Whilst this level of deformation is high, it represented a significant improvement in roadway conditions, providing a high level of confidence in the safe driveage of the leading roadway whilst still allowing stress relief to adjacent roadways.

Field experience

Mining was carried out with a heliminer and all support work, including megabolting was carried out with hand held machines. The face was advanced on a 1 m cut/support cycle, including installation of the megabolts at the face.

The reinforcement system comprised the standard 6 x 2.7 m bolts per row plus three megabolts per meter. The megabolts were 8 m long with the top 2 m encapsulated. Initial pull tests indicated that the 2 m chemical anchorage was sufficient to hold their rated 65 tonnes.

The initial megabolts were difficult to install as the roof was fairly broken. Post grouting of the megabolts was required to arrest roof movement. This was conducted on a weekly basis.

Fig. 8 – Elouera Colliery stress relief zone formed about a leading roadway – Thompson Panel
At 30 m inbye the dogleg (see Fig 1) 2.7 m long HPC bolts with cups and saucers were used instead of ‘X’ grade bolts. Megabolting with weekly grouting was continued.

A specific cycle of cutting the right hand side (the guttered side) was maintained to contain the guttering on that side. If the left side was cut first, guttering would have occurred on the right hand side of the initial cut which would become the centre of the roadway on widening.

The conditions in the leading roadway at the time of writing were characterised by a level, intact roof. Elevated stresses were still evident from the monitoring (roof displacement approaching 100 mm) and high bolt loads evident, however the roadway conditions are considered to be manageable.

Whilst time dependent movement is still occurring in the leading roadway at the time of writing, the level of roof deformation has decreased from over 300 mm outbye 24 line, to 130 mm where megabolts were installed at the face and post grouted to less than 100 mm where HPC bolts have replaced AX bolts.

The improved roadway conditions is considered to be a combination of:

- Cutting sequence-cutting the stressed side first and minimising opening width
- Installation of secondary support (megabolts) on first pass
- Post grouting (weekly) of the megabolts
- The introduction of the higher capacity HPC bolts with cup and saucer plates
- Close monitoring of roof movement and quick adjustments made to improve support
- The persistence and dedication of the crews whom have had to cope with fairly difficult conditions and have had to learn new support techniques very quickly.

CONCLUSIONS

Stress control methods were successfully used at Elouera Colliery to manage potentially problematic roadway conditions within a high horizontal stress environment.

Routine monitoring of roof displacement provided an effective means to establish the extent of stress relief achievable and to ensure that the magnitude of roadway deformation was within design expectations.

The successful implementation of stress control methods has allowed driveage in an area which may previously have been considered unmineable. Use of new support products (megabolts and HPC bolts) has allowed more manageable driveage conditions of the leading roadway whilst still allowing enough roof softening to provide stress relief to adjacent roadways.

The cost of additional resources required to drive the panel in a specific sequence is expected to be recouped with improved driveage rates in the flanking roadways and in increased recoverable reserves.

REFERENCES

A Safe Way to Reduce Roof Support Costs and Improve Safety and Productivity

M Smith' and R Seedsman'

ABSTRACT

The fundamentals concepts of roof support are similar between coal mining, metal mining and tunnelling and yet there are different design approaches, different hardware, and different levels of site investigation. This should not be the case and this paper discusses the key geotechnical issues that should underlie all roof support design - the need to reinforce defects. A design method to build a beam in laminated rock under high horizontal stresses is presented. The method allows for changing rock strengths, defect properties and varying horizontal stresses. At one level the design can be implemented with design charts and tables. Lessons learnt during the implementation of the designs at 5 different mine sites are discussed.

MINING ISN'T DIFFERENT

If mining is thought to be different - then coal mining must be different again! Why is it that underground metalliferous miners cannot believe the amount of roof support that coal miners install? Why is coal mine roof support hardware different from that sold into metalliferous mines? What do civil engineers mean when they talk about dowels and bolts? The aim of this paper is to tackle some of these questions while, at the same time, putting forward the case for a new (and at the same time a very old) approach to specifying roof support in coal mines.

Coal mine engineers use the same theories of soil and rock mechanics as their colleagues in civil engineering - be they foundation, slope stability, or tunnelling engineers. The materials in which they design are similar, the laws of physics are the same and the demands for safe and cost-efficient outcomes are the same. However there is a difference between the way coal mine engineers and civil engineers practise their professions. The differences do not relate to the different geological materials but appear to be steeped in tradition and history. Civil engineers are taught soil mechanics with its strong focus on elastic theory and the use of failure models and factors of safety (limit equilibrium), mining engineers are taught rock mechanics with a focus on empirical methods and back analysis.

Civil structures are capital intensive, and typically of a scale of tens to several hundreds of metres. The designs lack flexibility (restricted to a specific site) and have extremely tight specifications regarding stability, settlement, and serviceability. Even a domestic dwelling which may cost $100,000 to build requires a geotechnical assessment which costs in the order of $500. A major city development can cost $50-$100 million and require $30,000-$450,000 in

1 Partners International Pty. Ltd. Unanderra
geotechnical design. In the civil arena, the owners/operators are rarely the builder so they rely on project managers, consultants, and contractors.

As a design exercise, a modern longwall mine is no different. It is certainly capital intensive (say greater than $250 million), and has tight specifications - but this time on productivity and rate of return on capital. It does have a greater amount of flexibility in the design which allows it to develop without the areal density (holes/km²) of site investigation. The areal density of the site investigations can also be less as a result of the homogenisation that rock diagenesis tends to put over the complexity of sediments and soils. In contrast to the civil venture, the mining company is typically the designer, builder, operator and owner. There is no reason why the same relative level of expenditure on geotechnical design should not apply to longwall mines - how many new mining ventures spent $150,000-$1 million specifically on geotechnical issues? And if they did, did they get good value?

One of the considerations that appear to be lacking in coal mine geomechanics is the appreciation of the role of defects in the behaviour of rocks and rock masses. Defects are the natural weaknesses in rocks; in coal measures the defects are bedding surfaces, joints, coal cleat, greasy-backs etc. Given the relatively high horizontal stresses that characterise mining excavations in Australia and elsewhere, the most important defects are bedding. For example, mudstones, sandstones, and conglomerates all have a similar range of unconfined compressive strengths but it is known that sandstones give better roof than mudstones, and conglomerates can span longwall panels and delay caving - the difference is in the frequency of bedding defects.

In most cases, rock failure near an underground opening or even on a highwall is expressed as movement on existing defects. Until recently, coal mine geotechnical engineers did not have access to design tools to assess the role of bedding defects in controlling roof reinforcement. The previous design tools based on finite element and finite difference computer codes (such as FLAC) assumed the rock is a continuum (no open defects and the measured rock strengths reduced to account for most of the closed defects) and as a result the focus was on estimating rock strengths from laboratory testing and geophysical logs such as the sonic tool. Rock defects are modelled explicitly in discrete elements codes such as UDEC and these codes allow the defects to open and close. Discrete element codes are extremely numerically intensive and have not yet become commonplace in the industry. Barton (1996) gives a very good example of how UDEC and FLAC differ in the results of analysis of jointed rock masses. If defects are to be included in a design method, the key parameter is their location in the rock mass and their shear strength.

Since 1993 Coffey Partners International Pty Ltd has approached the design of roof reinforcement in bedded strata from an almost traditional civil engineering viewpoint (Seedsman and Logan, 1996). Building on the results of a number of excavations in the Hawkesbury Sandstone (Pells, Poulos & Best, 1991), a design approach based on beam building has been developed. The method recognises the role of bedding defects and seeks to install roof bolts so as to prevent the onset of movements along the defects. In this way the impact of the defects on the rock mass performance is negated. Analytical techniques, instead of numerical techniques, have been used so that the design focus can remain on the critical role of variability of the rock strata - the analytical techniques allow rapid redesigns and sensitivity studies. It is these design tools that have allowed new insights into coal mine roof reinforcement and underwritten the safety, productivity and cost improvements that are being achieved.
BUILDING A BEAM IN COAL MEASURE ROCKS

Introduction

There are specific steps in the application of beam theory to specify rock reinforcement in bedded strata. The general concept is not new but the ability to include the consideration of horizontal stresses has not been readily available in the past.

Field of application

The method is applicable to the building of a structural beam that can span a given opening. It is stressed that in thinly bedded roof there may be failure of scats between the roof bolts - this may mean that a strap or mesh is needed. The method does not specify strap or mesh requirements - this is done through a qualitative assessment of the immediate roof skin.

Note that horizontal stresses in the immediate roof may approach or exceed the compressive strength of the roof beam. In such a circumstance, beam building theory does not have application. The possible onset of failure can be recognised by comparing the measured unconfined compressive strength with the presumed or measured horizontal stress acting across the roadway. The formation of a development roadway can increase the horizontal stress acting in the immediate roof by 20% to 30%. There is an additional concentration of stress on retreat of a longwall face (Mathews, Nemcik, & Gale, 1992). If compressive failure is not indicated, then the design approach is to maintain the pre-failure (ie elastic) behaviour of the roof as long as possible by preventing delamination.

Key steps

There are 4 key steps.

1. In Step 1, the required thickness of the beam to be formed is determined. Well established linear arch or jointed rock beam theory (Brady and Brown, 1983) is used to determine the required thickness - input parameters include rock strength, rock modulus, span, and horizontal stress. Note that the horizontal stress can be the development stress or the abutment stress. Note that for most Australian situations, rock beams of about 0.5m to 0.7m thickness are indicated.

2. In Step 2, the forces that drive the delamination of bedded strata are considered. The assumption is that a bedded unit will delaminate if shear movements are allowed to develop along the bedding defects. Three dimensional elastic theory is used to calculate the shear forces and in particular the excess horizontal shear stresses along any defect after a roof bolt has been installed. Input parameters include the span, friction angle of the defect, and the horizontal stress. It has been shown that the shear stresses increase with height into the roof for about 2m - as a result the shear forces at a height into the roof equivalent to the required beam thickness become the design stresses and the design proceeds based on the assumption that a defect exists at this location (the design defect). The shear stresses are always less than the shear strength of intact rock as such a condition is in fact identified in step 1 and the design does not proceed to this step.
3. In Step 3 the shear resistance of bolts installed across defects is estimated. It is here that the significant difference between bolts and dowels is highlighted. Bolts are tensioned - they must be point anchored prior to tensioning and then they may be column grouted. The maximum shear resistance of a correctly tensioned bolt is the tensile strength of the bar multiplied by the tangent of the friction angle of the defect (for example 17 tonnes for X bar in sandstone). Dowels are untensioned and hence need to be full column grouted. If the defects are closed, dowels can provide the same maximum shear resistance as bolts. However the installation of a dowel cannot ensure that the defects are closed - in stressed, thinly bedded ground the defects can be open before the tendons are installed - in such a case the shear resistance can fall to as low as 4 tonne for the same X bar in sandstone of 20 MPa compressive strength. The difference is that, in the case of the open defect, the shear resistance of the dowel is limited by the bearing failure of the rock just ahead of the bolt.

4. The bolting pattern is optimised if the defects are closed. This can be achieved if the bolts are anchored in strata above the design defect. In step 4, ground anchorage concepts (Littlejohn and Bruce, 1975) are used to determine the required grouting length for the design bolt loads.

Limit state concepts are used to factor in uncertainties for the various input parameters. Typical factors are given in Table 1 but they vary on a site by site basis depending on the confidence in the available data. The equivalent factors of safety are typically in excess of 1.5.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Design Factor</th>
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</thead>
<tbody>
<tr>
<td>Unconfined compressive strength</td>
<td>Typically 0.85</td>
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<tr>
<td>Horizontal stress (modified as in text)</td>
<td>1.05</td>
</tr>
<tr>
<td>Shear demand</td>
<td>1.2</td>
</tr>
<tr>
<td>Shear resistance</td>
<td>1.1</td>
</tr>
<tr>
<td>Anchorage</td>
<td>2.0</td>
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</tbody>
</table>

**IMPLEMENTATION**

Implementing change is difficult but perhaps it is even more so in an industry steeped in tradition with a reluctance to change roof support strategies for fear of failure. There is an expectation that there is only one solution to a ground support problem - how often has been heard the complaint “All you geotechnical engineers disagree with each other - which one is right?”. It is possible that in the context that the request for advice is made, each solution is correct? No one solution will work in all conditions.

There is also the situation that one can go to a mine and suggest the use of 500mm UC steel sets to hold up a 5m wide roadway. There will be no arguments except for the practicality of carrying out such an installation - there is no request for the design calculations. But if creating a 500mm thick beam cheaply out of the roof strata is suggested, there are all kinds of arguments and requests for justifications and design reviews on a subject that most mining engineers have not been trained or educated to understand. And yet the rock beam is in many cases is more stable than steel sets.

The science of rock mechanics and its application of geotechnical engineering has a relatively low appreciation in coal mines compared to metalliferous mining and tunnelling. Working with a number of mining companies aims to manage a
change in their roadway support by the application of a significantly different package of design and in-mine services. The discipline that results is challenging for both the support designers and the mining companies but rewards are being seen in reduced costs, improved development rates, and more stable roof.

The first steps in implementation relate to the confirmation of some of the design assumptions. Roof rock strengths are confirmed by the conduct of short-encapsulation pull out tests and a back analysis based on ground anchorage concepts (Littlejohn and Bruce, 1975). An audit of current installation techniques reveals any short-comings in drill rigs and bolting hardware. Load cells are used to determine the level of pre-tension that can be achieved. Geological mapping seeks to define any changes in rock strength or structure that may require a revision of the geological model.

The support design is presented to the company management and the workforce for discussion and risk analysis. Experience is that the face workers are usually more prepared to accept the changes than management.

A trial driveage is then undertaken with geotechnical engineers confirming that the pattern is installed to design and monitoring the performance of the excavation. Poor workmanship will render the design invalid. Monitoring includes the following in addition to the usual extensometry:

1. Bolt installation techniques;
2. Standards of grouting;
3. Correct location of bolts;
4. Correct pre-tension on installation; and
5. Loss of pre-tension.

A back analysis of the results is conducted to check the design assumptions and to identify how the pattern can be further improved.

CASE STUDIES

The first application of the design method was conducted in December 1996 at Cumnock Colliery and the support and encouragement of all involved (management, underground workforce, and the Inspectorate) is acknowledged for what was seen by many at the time as a radical departure from standard practice. The objective was to eliminate roof failures in roadway development while maintaining development targets and controlling costs as dictated by the business plan. This was achieved in a trial area of 100m of roadway development by use of short bolts in place of long dowels and at a significantly lower support density. A major contributory factor to improved advance rates was the implementation of single pass drilling with miner mounted rigs where clearance is restricted by low seam height. Further trials are continuing prior to full implementation throughout the pit.

There were two requests from mines to reduce bolting intensity as a way of increasing development rates. In both mines it was demonstrated that a 50% reduction in the amount of steel in the roof was possible by the use of shorter pretensioned bolts in place of dowels. Development rates improved during one of the trials even with the disruption of Cofley
engineers to monitor the operation. In the deeper mine it still remains to reduce the pattern to 4 bolts/strap compared to the previous 8 bolts/strap - recommendations on how to achieve this have been made to the client.

At another mine the request was to assist in the redesign of support after a fall of ground associated with a fault zone. The recommendation made was to convert the 6 dowels/m to pre-tensioned bolts and to increase their length by 300mm to achieve better anchorage in the low strength ground. The trial driveage and 3 subsequent drives through the fault zone have been successful - the latter drives indicating that the first fall did not create a stress shadow.

In Mine E a stable coal beam is being formed in 1.2m of top coal. The geology is such that above the coal is a layer of very weak material that washes out during drilling and prevents anchorage of any longer bolts. Above this weak layer is a very strong unit that cannot be drilled with the available drill rigs, and even if it could be, a long-tendon based support system could not deliver the advance rates required.

The salient points of these 5 case studies are summarised in Table 2

<table>
<thead>
<tr>
<th>MINE</th>
<th>PREVIOUS PATTERN</th>
<th>COFFEY PATTERN</th>
<th>REMARKS</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>4 x 2.1m dowels</td>
<td>4 x 1.2m bolts</td>
<td>allowed single pass drilling and hence faster development, reduced consumable costs</td>
</tr>
<tr>
<td>B</td>
<td>6 x 2.1m dowels</td>
<td>6 x 2.4m bolts</td>
<td>recovered from fall, longer bolts needed for anchorage in weak ground</td>
</tr>
<tr>
<td>C</td>
<td>6 x 2.1m dowels</td>
<td>4 x 1.6m bolts</td>
<td>faster development rate during trial with less steel, crews reported significantly easier work</td>
</tr>
<tr>
<td>D</td>
<td>8 x 2.4m dowels</td>
<td>6 x 1.8m bolts</td>
<td>reduced costs, now aiming for a 4 bolt pattern on an ABM20 to get advance rate improvement</td>
</tr>
<tr>
<td>E</td>
<td>new mine</td>
<td>4 x 1.2m bolts</td>
<td>soft puggy band above seam then very hard sill</td>
</tr>
</tbody>
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CONCLUSIONS

By using design approaches more typical of civil engineering rock mechanics so that specific attention is paid to the bedding defects in the rock mass, the efficiency of roof bolting in coal mines can be improved. This can lead to improved advance rates and reduced costs. In the five projects discussed, the benefits are quite evident.

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Some Aspects of Rock Mechanics Applicable to Underground Coal Mining

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**ABSTRACT**

Three aspects of rock mechanics, namely, in-situ stress estimation by acoustic emission (AE) method, strength of rock mass and role of chemicals to reduce the strength are covered. It is possible to detect the previously applied maximum stress by stressing a rock specimen to the point where there is a substantial increase in AE activity. This is known as Kaiser effect. From the AE signatures in the second and subsequent loadings, AE take-off point was identified more easily than in the first loading. In determining the compressive strength of rock mass, two factors have to be considered, namely, the size effect on the compressive strength of intact rock and the effect of discontinuities on the compressive strength of rock mass. Although a modified Bieniawski criterion gives best agreement with the triaxial test data, modifications have been suggested to Hoek-Brown criterion due to its popularity. It is possible to reduce the tensile strength of sandstone by saturating it with weak chemical solutions made with dodecyltrimethyl ammonium bromide, polyethylene oxide and aluminium chloride by up to 30%. In the case of compressive strength, there is no appreciable effect. The possible explanation is that the chemical solutions produce an effect on the strength of sandstone only when the failure mechanism is dominated by tensile mode.

**INTRODUCTION**

The design for underground excavations begins with investigations to determine characteristics of rock and rock mass. Chief among these is Structural Geology because, in underground excavations, instability is usually caused by discontinuities (faults, joints and bedding planes). Next is the estimation of in-situ stress. The third is testing of the rock and rock mass to determine Mechanical Properties such as compressive and shear strengths. The fourth activity concerns Groundwater which is significant to underground operations and to excavation stability in particular.

This paper concentrates on acoustic emission technique for estimation of in-situ stress, strength of rock mass and the effect of chemical solutions on the strength of sandstone.

**ESTIMATION OF IN-SITU STRESS**

Although various techniques have been proposed and developed to determine in-situ stress, the determination of in-situ stress is not an easy task and all suffer from deficiencies and limitations. The main deficiency of established techniques such as over coring method or hydraulic fracturing method is that they are usually expensive and time-consuming. Other shortcomings of the techniques are that they are deficient for measuring in-situ stress at depth in remote regions which are hard to access from boreholes or mine workings. An alternative method for determining the stress state at depth and in remote regions is to take advantage of the acoustic emission (AE) method.

*AE method of determining in-situ rock stress*

The "Kaiser effect" of AE suggests that previously applied maximum stress might be detected by stressing a rock specimen to the point where there is a substantial increase in AE activity. The AE technique has been developed and tried by various researchers in the past (Kanagawa et al., 1976; Kurita and Fujii, 1979; Houghton and Crawford, 1987; Seto et al., 1989a, b, 1992a, b, 1996; Holocomb, 1993; Utagawa et al., 1995) with the aim of providing a practical technique for retrieving the

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Kaiser effect. That is a recollection of the maximum previous stress to which a rock had been subjected to its in-situ environment.

Fig. 1 shows a typical example that indicates the existence of Kaiser effect in a sandstone specimen. Data for the specimen tested 5 minutes after the previous cyclic loading up to 10 MPa are shown. An arrow indicates the previous maximum stress. The previous stress level was within elastic range. In all experiments conducted within the short delay time, the existence of Kaiser effect in rock specimens could be clearly observed and the assigned stress from the take-off point of AE signature was within 5%.

Fig. 2 shows the AE signatures in the repeated loading-unloading of a coal core specimen “A” taken from the depth of 356 m 44 days before the test. The maximum previous stress was recognised by clear indication of AE increase, which is indicated by the arrow in the figure. In both the first and second loadings, AE increase can be recognised clearly at the same stress level. In the second loading the emissions below the stress level were significantly reduced, and an AE take-off point was identified more easily than in the first loading. The estimated stress from the AE signatures was 9.2 MPa, which was very close to the overburden pressure (8.5 MPa) estimated from the depth of 356 m.

In the same area, CSIRO have conducted a number of in-situ stress measurements using hollow cells and hydraulic fracturing technique (Enever and Doyle, 1996). When compared the result of vertical stress at the same depth with that from AE method, 8.9 MPa was from hydraulic fracturing technique and 9.2 MPa from AE method, which are also well consistent.

Fig. 3 shows the AE signatures of a rock core specimen “B” taken from the depth of 310 m nearly two years ago. Although the take-off point of AE event rate was not clear in the first loading, the previous stress could be estimated from AE signature in the second loading. The estimated stress (7.1 MPa) was also well consistent with the overburden pressure (6.4 MPa). The time lag of two years did not deter the evaluation of the critical in-situ stress condition. Rock cores could recollect the in-situ vertical stresses reasonably well within 10% even in case of two years time lag. The estimated vertical stresses from the AE method suggested in this paper, which utilises the AE signature in cyclic loading, agree well with the overburden pressure.

**Strength of rock mass**

The discussion is limited to compressive strength. To properly assess the compressive strength of rock mass, two factors have to be taken into account. The first factor is the effect of size on the compressive strength of intact rock (in between the discontinuities) of the size under consideration and the second factor is the effect of discontinuities (number and orientation with respect to stress field) on the compressive strength estimated taking into account the size effect.

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**Fig. 1 - A typical example of Kaiser effect in sandstone**

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**Fig. 2**

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**Fig. 3**
The influence of joints on the compressive strength of a rock mass was studied by a number of investigators using models. Lama (1974) studied the effect of horizontal, vertical and orthogonal joints on uniaxial compressive strength in models. The drop in uniaxial compressive strength is small after the number of joints exceeds 6 and is in accordance with the results of Goldstein et al. (1966) and Walker (1971). On a percentage basis, the decrease in the uniaxial compressive strength of the model with horizontal or vertical joints is about 30%.

According to Goldstein et al. (1966), the results can be represented by the following relationship:

\[ \frac{\rho_m}{\rho_e} = a + b \left( \frac{l}{L} \right) \]  

where \( \rho_m \) = uniaxial compressive strength of the model (composite block);
\( \rho_e \) = uniaxial compressive strength of the element constituting the block;
\( L \) = length of the model;
\( l \) = length of each element;
a, b and c are constants, where c < 1 and b = (1 - a) (Fig. 4).
Fig. 3 - AE signatures in the 1st and 2nd loading of a rock core taken from the depth of 310m nearly 2 years ago

Fig. 4 - Variation of uniaxial comprehensive strength ratio, $\sigma_{cm}/\sigma_{ce}$ with joint frequency, $L/l$, of rock mass

For models with orthogonal joints, the results of Lama (1974) indicate that the uniaxial compressive strength reduces as the number of elements increases. It reduces to more or less constant value when the joint density (number of elements) reaches about 150. The relationship between uniaxial compressive strength and joint density can be represented by the following equation:

$$p_e = k + n' \quad (2)$$

4 After Goldstein et al., 1966
where $P_c$ = uniaxial compressive strength of a model with less than 150 elements;
$k$ = uniaxial compressive strength of model containing more than about 150 elements (real strength of the system);
$n$ = number of elements; and
$d$ = constant.

The value of $d$ is higher for models of stronger material and comparatively lower for models of weaker material.

The configuration of the joint system with respect to the stress field also influences the compressive strength. The influence of the inclination of the weakness plane on the compressive strength for various rocks was studied by a number of authors (McLamore and Gray, 1967; Donath, 1972).

**Strength criteria for rock and rock mass**

The theoretical strength criteria based on the actual mechanism of fracture do not fit the experimental results properly and to overcome this problem, many empirical criteria were formulated for rocks and rock masses. The strength criteria can be written in terms of either (1) principal stresses, $p_1$ and $p_3$ at fracture or normalised principal stresses at fracture obtained by dividing the principal stresses, $p_1$ and $p_3$ at fracture by the relevant uniaxial compressive strength, $P_c$ or (2) shear and normal stresses at fracture or normalised shear and normal stresses at fracture with respect to uniaxial compressive strength. A typical relationship between $p_1$, $p_3$, or $p_1/p_s$ and $p_3/p_s$ at fracture for rocks is a nonlinear one.

Four empirical strength criteria for rock and rock mass proposed by Bieniawski (1974a) - Yudhbir et al. (1983), Hoek and Brown (1980a, b), Johnston (1985) - Sheorey et al. (1989) and Ramamurthy (1986) - Arora (1988) were assessed regarding their applicability for coal (Vutukuri and Hossaini, 1992a) and jointed plaster of Paris (Vutukuri and Hossaini, 1992b). The following modified Bieniawski criterion gave the best agreement with the triaxial test data:-

$$\frac{p_1}{p_m} = 1 + B_n \left( \frac{p}{p_m} \right)^{a_n}$$

where $p_1$ and $p_3$ = principal stresses at fracture;
$p_3$ = uniaxial compressive strength of rock mass;
$B_n$ and $a_n$ = rock mass parameters.

The important conclusions are that $a_n$ is more or less constant but $B_n$ is a function of $p_m$.

Due to popularity of Hoek-Brown criterion, the following modification has been suggested:-

$$\frac{p_1}{p_m} = \frac{p_1}{p_m} + (1 + m_n \frac{p_1}{p_m})^{0.5}$$

where $m_n$ = rock mass constant.

To use the equation, $p_m$ and $m_n$ are required. From the original Hoek and Brown criterion for rock mass, the following equations have been derived:-

For undisturbed rock mass:

$$\frac{p_m}{p_i} = [ \exp ((RMR - 100)/9)]^{0.5}$$

$$m_n/m = 1/[\left( \frac{p_m}{p_i} \right)^{0.3598}]$$

where RMR = Rock Mass Rating (Bieniawski, 1974b);
$p_i$ = uniaxial compressive strength of intact rock comprising the rock mass and $m = constant for the intact rock.

Fig. 5 depicts the relationship given in Equation (6).

The critical parameter in the modified criterion is the ratio between $p_m$ and $p_i$. According to Hoek and Brown (1988), the ratio depends upon Rock Mass Rating (RMR) as well as condition of the rock mass i.e. undisturbed or disturbed. The
relationship for undisturbed rock mass is given in Equation (5). Aydan et al. (1997) reviewed the topic of rock mass strength in some detail.

From the results obtained in laboratory experiments on discontinuous models of plaster and sandstone, the following relationships have been determined:

For plaster:

\[ m_y/m = 1/[(\rho_m/\rho_s)^{0.842}] \]  \hspace{1cm} (7)

For sandstone:

\[ m_y/m = 1/[(\rho_m/\rho_s)^{0.499}] \]  \hspace{1cm} (8)

Fig. (6) depicts the relationship given in Equation (8).

![Graph showing \( \sigma_{cm}/\sigma_c \) versus \( \frac{m_y}{m} \) for the undisturbed rock mass for the modified Hoek and brown criterion (after Vutukuri and Hossaini, 1995)]
Effect of chemical solutions on the strength of sandstone

Chemical alteration of the strength was investigated to establish the fundamental knowledge for chemically assisted fracturing. If the rock strength can be chemically lowered, this technology would be useful to raise the fracturing efficiency. Brazilian tests and multi-stage triaxial compression tests were performed on Gosford sandstone saturated with solutions of dodecyltrimethyl ammonium bromide (DTAB), polyethylene oxide (PEO) and aluminium chloride (AlCl₃). The tensile strength varied with the concentration of chemical additives, became the lowest at certain concentrations which are consistent with the zero zeta-potential concentrations (Fig. 7). AE activity was the most active in dry specimen, and the least AE activity was found in the specimen saturated with chemical solution (Fig. 8). The uniaxial and triaxial compressive strengths did not vary significantly with concentration of the chemical solutions. In these tests, the failure mode was dominated by shear failure.

In some coal mines, hard massive sandstone formations are encountered in the immediate vicinity of the coal seam in the roof. In such situations, delayed caving of the roof creates a number of problems including air blast. Four alternatives can be thought of to deal with such roof formations difficult to cave.

1. Packing of goaf with filling material
2. Using very high capacity supports
3. Using supports with high flow relief valves
4. Reducing the strength properties of roof formations by water injection under high pressure or blasting

The results of this part can be applied to reduce the strength properties of roof formations by injecting water mixed with appropriate chemicals.

\[ y = \frac{1}{x^a} \]
\[ a = 0.4949 \]
\[ R^2 = 0.99 \]

Fig. 6 - \( \frac{\sigma_{cm}}{\sigma_c} \) versus \( \frac{mm}{m} \) for sandstone models tested in the laboratory for the modified Hoek and brown criterion.\(^5\)

\(^5\) After Vutukuri and Hossaini, 1995
CONCLUSIONS

In-situ stresses were estimated using AE signatures (Kaiser effect) in repeated loadings on rock core specimens. The estimated values were comparable to the ones obtained from other techniques such as overcoring method and hydraulic fracturing method. The time lag of up to two years for studying the Kaiser effect on cores has not influenced the results. The modified Bieniawski criterion appears to be the most appropriate one for coal and jointed plaster of Paris. The important conclusions are that $a_m$ is more or less constant but $B_m$ is a function of $P_{cm}$. A modification has been suggested to the original Hoek-Brown criterion for rock mass. The relationship between $m_/m$ and $P_{cm}/P_c$ has been given for undisturbed rock as per original Hoek-Brown data. From laboratory studies, relationships have been suggested for plaster and sandstone. The tensile strength of sandstone varies significantly with concentration of chemicals and can be markedly influenced by the zeta potential at the rock-liquid interface. The compressive strength did not vary to any significant extent with the concentration of chemical. The possible explanation for the experimental results is that the chemical solution affects the strength only when the failure mechanism is dominated by tensile mode.

Fig. 7 - Tensile strength variation and zeta potential versus DTAB concentration in water for Gosford sandstone
Fig. 8 - AE behaviors of dry sandstone, water-saturated sandstone and sandstone saturated with DTAB solution (10^{-3} \text{ mol/l}) during Brazilian test

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Stress Corrosion Cracking of Rock Bolts

P Gray

ABSTRACT

This paper outlines the mechanism of Stress Corrosion Cracking (SCC) and how it can cause premature failure of rock bolts. SCC occurs due to the progressive development and growth of cracks in the surface of a metal, and it is caused by the combined effects of stress and a corrosive environment affecting susceptible metals and alloys. It can occur in most metals including stainless steel. It is not a new phenomenon, but its effect on rock bolts has only been recognised recently.

Rock bolt failures caused by SCC appear to be brittle failures which occur at less than the ultimate tensile strength of the bolt. In the mining industry, SCC failures of rock bolts can occur a few months, or after many years of exposure to stress and a corrosive environment. If SCC of rock bolts is suspected, then the rock bolts should be examined by a metallurgist, and steps should be taken to reduce the problem.

Considerable research in the past has shown that SCC is a complex phenomenon and no one mechanism can explain all cases of SCC failures. It is therefore unrealistic to expect research to find a complete solution to SCC failures of rock bolts, but simple measures can be taken immediately to reduce or minimise the SCC problem. These measures include:

- to ensure that rock bolts are fully encapsulated with resin; and,
- to use rock bolts that are less susceptible to SCC.

Background

Rock bolts were used for the first time in the underground coal industry in Australia in the late 1940's at BHP's Elrington Colliery, near Cessnock in NSW. Today, rock bolts are in widespread use throughout the underground mining industry, as well as being used in the civil engineering and tunnelling industries. Rock bolts have played a major part in the improvement of roof conditions for the Australian mining industry, and hence have improved productivity and efficiency.

Nevertheless rock bolts are used in the harsh mining environment where there are often difficult ground conditions, and rock bolts can be subjected to high stresses, and sometimes also with corrosive groundwater conditions. When the stresses on the rock bolt exceeds the strength of the bolt, then rock bolt failure does occur.

Rock bolt failure can occur either by tensile failure, or by shear failure, or by failure of the bolt anchor system (resin or mechanical anchor). Geotechnical engineers attempt to minimise rock bolt failures by increasing the number and the capacity of the bolting systems used, and by optimising the mining stress conditions (by changing pillar size, mining direction etc). However, failure of rock bolts is something which does occur in the mining industry.

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Stress Corrosion Cracking

Some mines have reported unusual rock bolt failures. These rock bolt failures appear to be brittle failure of the bolt, which occurs at less than the ultimate tensile strength of the bolt. BHP have examined these failed bolts and the failure mode has been identified as "Stress Corrosion Cracking".

Stress Corrosion Cracking (SCC) is a progressive fracture mechanism which can occur in virtually all metals. It was first identified by the British Army in India in the late 19th century when cracks appeared in brass cartridge cases from ammunition. The cartridge case developed high tensile hoop stresses when the bullet was inserted and this combined with high temperatures, high humidity and traces of ammonia in the air, caused SCC to occur. SCC has also been found in Bronze Age swords, and it is therefore not a new phenomenon, but its effect on rock bolts has only been recognised recently.

SCC occurs due to the development and growth of cracks caused by the combined effects of stress and environmental conditions affecting susceptible metals and alloys. A sustained tensile stress and a corrosive environment (not necessarily acidic) can cause the development of small cracks which propagate into the susceptible metal (in this case a rock bolt). These small cracks initiate tensile failure which can be sudden and unpredictable.

In the mining industry, SCC failure of rock bolts can occur a few months or after many years of exposure to stress and a corrosive environment.

![SCC growth rate vs tensile strength for high strength steels (after Atrens & Wang, Ref.1)](image)

Fig. 1 - SCC growth rate vs tensile strength for high strength steels (after Atrens & Wang, Ref.1)

SCC is a complex phenomenon and no one mechanism can explain all cases of SCC failures, but it is known to occur in many metals and alloys including Aluminium, Titanium, Brass, Steel and Stainless Steel (see Figure 2).
One form of SCC is often called Hydrogen Embrittlement (HE). When high strength steels are stressed and are also exposed to environments that release hydrogen, HE cracking can occur. Cross Section (Figure 2) see cracks in Stainless Steel.

Figure 1 indicates that SCC can occur in a wide range of steel types (after Atrens, Ref.1). However, Figure 1 also shows that higher strength steels are generally more susceptible to SCC than lower strength steels. It can be seen that the crack growth rate is approximately $10^{-6}$ m/sec for steels with an ultimate tensile strength (UTS) of 1400 MPa, compared to a crack growth rate of only $10^{-11}$ m/sec for steels with a UTS of 600 MPa (Ref 1).

Nevertheless, SCC only occurs within a narrow band of conditions involving the stress, the environment (the corrosive conditions), and the susceptible alloy.

Stress corrosion cracking failures of rock bolts

An SCC failure of a rock bolt typically appears to be a brittle failure. An SCC failure surface is normally a fracture plane at right angles to the axis of the bolt (the direction of the axial force), and there is no necking of bolt adjacent to the fracture surface. The fracture surface is frequently located at the end of the exposed free length of the bolt immediately below the resin encapsulation.

Fig. 3 - Typical SCC rock bolt failures
The surface condition of the bolt often does not show significant surface corrosion, and the end bearing plate commonly indicates that the maximum axial force on the bolt was less than the yield strength of the bolt.

A typical SCC failure of a rock bolt is shown in Figure 3.

Detailed investigation of SCC failures using Magnetic Particle Impregnation (MPI) techniques reveals that many small SCC cracks develop in the bolt adjacent to the ultimate fracture surface. These small cracks frequently develop at the base of the rib profile where there is a sharp change in angle on the longitudinal section of the bolt, and this creates a “stress raiser” in the bolt (see Figure 4).

![Fig. 4 - SCC at the base of the ribs on a rock bolt](image)

SCC cracks are usually less than 1mm deep, and are dendritic in section (see Figure 5). The crack which initiates SCC failure can be located by the fracture lines which radiate out from it as shown in Figure 6, and this crack is less than 1mm deep.

![Fig. 5 - Dendritic nature of SCC at the base of the rib profile](image)

SCC bolt failures have occurred in mines with “wet” roof conditions, i.e. where water has been dripping from the roof, and in mines where the roof conditions appear relatively dry.
Observed SCC bolt failures have occurred in mines between approximately 12 months and 24 months from the time of installation.

![Fracture face of an SCC failure. Arrow shows the crack which initiated failure](image)

SCC bolt failures are not restricted to one particular steel grade or one steel supplier, but have only been observed in high strength bolts with a minimum yield strength of 600 MPa.

Currently, based on the samples provided for testing, rock bolt failures caused by SCC have occurred in the Western and Southern Coalfields of New South. However it is possible that the problem could be more widespread than this.

In summary, SCC failed rock bolts are often characterised by:

- a sharp, brittle type fracture surface with no necking of the bolt adjacent to the fracture surface;
- failure at less than the yield strength of the bolt with no significant load on the bearing plate;
- a fracture surface located immediately below the resin encapsulation;
- SCC cracks located at the base of the rib profile.

Some indicative signs of SCC of rock bolts in a mine are:

- loose bearing plates despite obvious roof sag;
- loose rock bolts;
- broken bolts fallen out of the roof;
- unusual or heavy roof conditions.

**Solutions to the SCC failure of rock bolts**

Although SCC is a fundamental characteristic of metals and alloys, it only occurs within a narrow band of conditions, and is therefore a fugacious and difficult problem to solve completely. However any solution to the SCC problem needs to take account of three factors vis: stress, corrosive conditions, and metallurgy.
Stress

The tensile stress level in a bolt cannot be reduced cost effectively, since SCC failure occurs at much less than the yield strength of the steel. Very large diameter bolts would reduce the tensile stress in the bolt but would incur a significant weight penalty. Lower strength steel grades are less susceptible to SCC, but would also incur a similar weight penalty since they would by necessity have to be larger diameter bolts.

It should be noted that SCC cannot occur in an area of compressive stress and a surface compressive stress can be developed by either shot peening or by quenching (Tempcore process).

Corrosive conditions

There are a wide range of possible corrosive conditions that could cause SCC. These range from brass exposed to ammonia, mild steel exposed to caustic conditions, stainless steels exposed to chloride conditions, and high strength steels exposed to environments that can release hydrogen.

If SCC is suspected, a chemical analysis of the mine water can indicate if there is the potential for SCC to occur.

One way to reduce SCC is to prevent the corrosive conditions from affecting the steel rock bolt. This could be done with some form of coating of the bolt. An anti-corrosive coating may help, but work needs to be done to determine its effectiveness against SCC, since even stainless steel can be affected by SCC. In addition, a zinc coating (galvanising) may in some instances actually exacerbate the problem of SCC. Finally, heavy corrosion protection systems such as plastic sleeves over the bolt may significantly reduce the load transfer capability of the rock bolt.

A relatively simple measure to reduce SCC is to ensure that all bolts are fully encapsulated with resin. This will not prevent SCC completely since the resin can crack under high load and the bolt could then still be exposed to corrosive conditions.

Metallurgy

There are some steps that can be taken with the metallurgy and the bolt design to reduce SCC. Firstly, lower strength steels are less susceptible to SCC than higher strength steels. Therefore a rock bolt with an ultimate tensile strength of 900 MPa would be less susceptible to SCC than a cable bolt with an ultimate tensile strength of 1750 MPa.

Secondly, steel bars with a high surface compressive stress are also less susceptible to SCC than bars without a compressive stress. This surface compressive stress can be achieved by, for example, shot peening (as in the case of drill rods), or by quenching and tempering (as in the case of Tempcore bars). Finally, the design of the ribs on the rock bolt has some influence on its susceptibility to SCC. On a conventional rock bolt, the rib projects from the core size of the rock bolt at a sharp angle (approximately between 60 and 90 degrees). This sharp angle creates a "stress raiser" in the bar when the bar is subjected to a tensile force, and consequently SCC cracks frequently occur at this point (see Figure 4). Rock bolts which therefore minimise this sharp angle and reduce the "stress raiser effect", are therefore preferable to ordinary rock bolts. In addition, all stress raisers in a rock bolt or cable bolt are to be avoided (eg cuts, grooves etc).

In summary, the stress conditions, the corrosive environment, the steel properties, the steel micro-structure and the bar design, are all factors which influence the potential for SCC to occur.

Recommendations

If SCC failure of rock bolts is suspected as indicated by the characteristics as outlined above, then the following is recommended:
• Inspect roadways and tunnels and collect samples of broken bolts, water samples if possible, and collect installation data (eg bolt type, steel grade, installation date etc) on SCC failure of rock bolts. Have these bolts and water samples analysed to confirm SCC or not;

• Where SCC failures are suspected, examine roof conditions with a geotechnical engineer to determine if additional support is required;

• Ensure all bolts installed in the future are fully encapsulated with resin right to the collar of the hole. Point anchored bolts with even a short free length are not recommended.

• Use rock bolts which reduce the potential for SCC to occur (ie. Bolts which have been shot-peened or Tempcored and which have a radius in the base of the rib profile).

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A Comparison Between Hoek-Brown and Bieniawski Criteria for Coal and Rocks

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ABSTRACT

The applicability of Bieniawski and Hoek-Brown empirical strength criteria has been assessed for different groups of coal and various types of intact rocks by using a vast number of published triaxial test data from various places. Analysis of individual data sets revealed that the traditional forms of the criteria do not have a perfect agreement with the data. A strong negative correlation has been observed between $B$ in Bieniawski's criterion and $m$ in Hoek and Brown's criterion with uniaxial compressive strength of materials. Both criteria have been modified and empirical relationships have been introduced for coal as follows:

$$m = 62.903 - 34.213 \left( \log \rho_c \right)^{0.9772}$$
$$B = 10.152 - 4.709 \left( \log \rho_c \right)^{0.8889}$$

Similar relationships have been developed for different rock types as well.

A comparison between the applicability of each of the above approaches with the conventional criteria reveals a very significant advantage for new approaches and a supremacy for the Bieniawski criterion in all cases particularly in the case of coal.

The modified Bieniawski criterion fits coal as well as different types of rocks with excellent accuracy. The modified Hoek-Brown criterion gives a good result for rocks but does not fit coal data quite well. In other words, Hoek-Brown criterion is not an suitable one for coal.

INTRODUCTION

To estimate the strength of rock and rock mass a failure criterion is required. The theoretical triaxial strength criteria based on the actual mechanism of fracture do not fit the experimental results properly and to overcome this problem, many empirical criteria have been formulated for rocks.

Laboratory strength data values are the starting points for estimating the strength of rock and rock mass. If a criterion fails to fit the laboratory strength data properly, then its applicability to real field cases would certainly be doubtful. Out of many strength criteria developed so far, none of them has been accepted to be a global formula capable of simply describing the strength of geomechanical materials in general.

Coal is one of the most important energy resources. As it is widely mined around the world, and receiving attention on the current mining and energy scene, developing an appropriate strength criterion for use in coal seams to be of considerable value.

Amongst all available strength criteria proposed for the estimation of the strength of rocks and geomechanical materials, a very few of them have been extended to coal. One criterion proposed for coal is that suggested by Sheorey, Biswas and Choubey(1989). Using this criterion in practice requires a very long complicated procedure, as it is understood from the

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original paper. None of the two failure criteria examined in this investigation (i.e., Hoek-Brown, 1980a & b and Bieniawski, 1974) have taken coal into consideration and the appropriate values for constants are not available for the same.

**STRENGTH CRITERIA**

The general form of a strength criterion is:

\[ P_1 = f(P_2, P_3) \]

where \( P_1, P_2 \) and \( P_3 \) are the principal stresses at failure.

Because the available data indicate that the intermediate principal stress, \( P_2 \), has very little influence on strength than the minor principal stress, \( P_3 \), all of the criteria used in practice are reduced to the form:

\[ P_1 = f(P_3) \]

or in its normalised form:

\[ \frac{P_1}{S_c} = f\left(\frac{P_3}{S_c}\right) \]

**Hoek-Brown's criterion for intact rocks**

\[ \frac{\sigma_1}{\sigma_c} = \frac{\sigma_3}{\sigma_c} + (1+m)\left(\frac{\sigma_3}{\sigma_c}\right)^{0.5} \]  

\[ m = \text{a constant value for each rock type} \]

**Bieniawski's criterion for intact rocks**

\[ \frac{\sigma_1}{\sigma_c} = 1 + B\left(\frac{\sigma_3}{\sigma_c}\right)^{\alpha} \]  

\[ B = \text{a constant value for each rock type and} \]

\[ \rho = 0.65 \text{ or } 0.75 \text{ for all rock type} \]

**DATA SELECTED FOR ANALYSIS**

Coal data representing twenty six seams and collieries are from two publications (Hobbs, 1964 and Das and Sheorey, 1986). These data are homogeneous and on specimens of almost the same size. Thus, they are to a reasonable extent free from the effects of specimen size.

Intact rocks data include various types of geomechanical materials of diverse lithological and mechanical characteristics from weak over-consolidated clays of unconfined compressive strength of 24 MPa to strong hard granite and granodiorite of unconfined compressive strength of 427 MPa.

**ANALYSING THE APPLICABILITY OF THE CRITERIA FOR INTACT COAL**

Analysis of individual data sets revealed that none of the existing criteria shows perfect agreement with experimental values of coal strength. Although unique values of the constants in both criteria have been determined with good coefficients of determination for overall data, a wide variation has been noticed in the values of the constants when individual data sets have been analysed.
Hoek and Brown’s criterion

The value for $m$ has been determined to be 25.132 for the combination of all the 180 pairs of the data. The coefficient of determination has been found to be 0.9105. Plot of all data along with regression curve is shown in Fig. 1. When the data groups are analysed individually, the outcomes differ widely from what is achieved by mixing the whole data of all groups. In general, the best correlation amongst all single constant values assigned to $m$ was due to $m = 10$ (suggested for mudstone, siltstone, shale and slate by Hoek and Brown). Fig. 2 shows six examples of those cases for which by applying this criterion the lowest correlation with exact data has been obtained.

![Graph showing Hoek-Brown and Bieniawski criteria](image)

Fig. 1 - Plot of $S_1/S_C$ versus $S_3/S_C$ for all data along with regression lines according to the two criteria.

Analysis of individual data sets has given a range of values from 5.3795 to 50.190 for $m$. Analysis of these values along with $p_c$ has indicated that there is a significant correlation between them (Fig. 3). The relationship between $m$ and $p_c$ has been found to be as follows:

$$m = 62.903 - 34.213 \cdot (\log p_c)^{0.9772} \quad (3)$$

Bieniawski’s criterion

A plot of $S_1/S_C$ versus $S_1/S_C$ for all data along with the regression curve is shown in Fig. 1. The values for $B$ and $c$ have been determined to be 3.7062 and 0.9225 respectively. The coefficient of determination has been found to be 0.9551.
Fig. 2 - Examples of discrepancy in the traditional Hoek-Brown criterion for coal.
The best result amongst all various constant amounts of $B$ was obtained for the case in which $p = 0.65$ and $B = 3.0$. Fig. 4 demonstrates six examples of the cases in which this criterion has given the lowest coincidence with real data (for $p = 0.65$ and $B = 3.0$).

Analysis of individual data sets has given a range of values for best fitting $B$ from 2.22 to 6.8326 and for best fitting $p$ from 0.4517 to 0.7423.

The best statistical average for $p$ was found to be 0.6. Taking $p$ as 0.6, the values for parameter $B$ have been recalculated for all the individual data sets. From this analysis, $B$ has been found to be between 2.0663 and 7.7150 and the relationship between $B$ and $p_c$ has been found to be as follows:

$$B = 10.152 - 4.709 \cdot (\log p_c)^{0.8889} \quad (4)$$

The coefficient of determination for this regression has been found to be 0.9164 (Fig. 5). Fig. 6 depicts the correlation between modified Bieniawski criterion with 6 groups of experimental data for which the lowest coefficient of determination was observed. Comparison of Figs. 2 and 4 with Fig. 6 reveals the supremacy of the new version of the Bieniawski criterion.

**ANALYSING THE APPLICABILITY OF THE CRITERIA FOR INTACT ROCKS**

The same analysis as conducted for coal was carried out for different rock types, namely, limestone, granite, granodiorite, shale, sandstone, claystone and liparite. Although the data were from various sources with differences in techniques, size and shape of specimens, the results indicate that the parameters cannot be regarded as constant values. For each particular rock type there found to be a correlation between $B$ in the Bieniawski criterion and $m$ in the Hoek-Brown criterion with $p_c - p$ in the Bieniawski criterion takes different values for different types of rocks. Limestone and granite are taken as two examples.
Fig. 4 - Examples of discrepancy in the traditional Bieniawski criterion for coal
The values of $\alpha$ and the relationships between $B$ and $m$ with $\rho_C$ for these two cases are as follows:

for limestone:
$\alpha = 0.76$

$$B = -3.3538 + 10.883 \log(\rho_C)^{0.8131} \quad (5)$$
$$m = -1.6115 + 62.05 \log(\rho_C)^{2.7421} \quad (6)$$

for granite:
$\alpha = 0.65$

$$B = 3.4452 + 21.617 \log(\rho_C)^{2.757} \quad (7)$$
$$m = 971 + 1055.3 \log(\rho_C)^{-0.06} \quad (8)$$

Fig. 5 Plot of $B$ versus $\rho_C$ in the Bieniawski's criterion for coal($\alpha = 0.6$)
Fig. 6 The lowest correlative cases of the modified Bieniawski criterion.

Figs. 7 and 8 give the relationship between $B$ in the Bieniawski criterion and $\rho_c$ for limestone and granite as 2 examples.
Fig. 7 Plot of $B$ versus $\rho_c$ for limestone and marble in the Bieniawski criterion

Fig. 8 Plot of $B$ versus $\rho_c$ for granite and granodiorite in the Bieniawski criterion

$^2 \alpha = 0.76$

$^3 \alpha = 0.65$
SUMMARY AND CONCLUSIONS

- Although eventual modifications to the selected criteria for intact rocks requires more investigations in which more proper data groups must be analysed within any rock type, the first conclusion coming out from the assessments done in this paper implies that treating criteria parameters with constant values would result in considerable inaccuracy, even for intact materials.

- Tables 1 and 2 summarise the statistical analysis of applying the two modified criteria to the laboratory data of coal and intact rocks ($r^2$ is the coefficient of determination). As shown in these tables, a significant accuracy is obtained by applying the modified Bieniawski criterion for both coal and intact rocks. This criterion, therefore, satisfies all the requirements for an desirable empirical strength criterion provided that it is amended with the modifications suggested in this investigation.

The modified Hoek and Brown criterion gives good level of accuracy for rocks but is not a suitable criterion for coal as shows low correlation with the coal data (Table 1).

Table 1 Summary of comparison of 2 modified criterion for coal.

<table>
<thead>
<tr>
<th>criterion</th>
<th>Range of $r^2$ with real data</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>cases with $r^2 \geq 0.95$</td>
</tr>
<tr>
<td>modified Bieniawski</td>
<td>69%</td>
</tr>
<tr>
<td>modified Hoek-Brown</td>
<td>38%</td>
</tr>
</tbody>
</table>

Table 2 Summary of comparison of 2 modified criterion for intact rocks

<table>
<thead>
<tr>
<th>criterion</th>
<th>Range of $r^2$ with real data</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>cases with $r^2 \geq 0.95$</td>
</tr>
<tr>
<td>modified Bieniawski</td>
<td>81%</td>
</tr>
<tr>
<td>modified Hoek-Brown</td>
<td>72%</td>
</tr>
</tbody>
</table>

- An estimate of the triaxial strength can be made by means of the Bieniawski criterion with a variable $B$ dependent upon $P_c$ and a certain constant $\rho$ for each particular material. The only parameter required for this criterion is the unconfined compressive strength which can be determined simply.

- A strength criterion must be capable to deal with different conditions of a certain type of rock having different properties. Such a criterion may or may not provide the best estimation for a large number of mixed data from various collieries and seams around the world.

- In practice, a design engineer is faced with a certain type of rock with its particular properties. The characteristics of any rock type may change from place to place or even from one part to another part of the same seam or block. A criterion must be flexible enough to fit the various conditions of rock properties.
ACKNOWLEDGMENT

The author wishes to thank Teheran University for supporting his further research on the failure criteria. Because the author first started his work on this topic as a part of his study at the University of New South Wales, he wishes to thank Dr V. S. Vutukuri for his supervision during the author’s PhD study.

REFERENCES


Dartbrook Mine - A Case Study

J Hayward

ABSTRACT

Any project carries a number of challenges and risks. Inappropriate design, poor project management, time, industrial disputation and capital over runs are all areas which can impact on a project cash flows and return on investment. In the resource Industry you have the added complexity and risk of geology which is difficult to predict and sometimes unforgiving. The Dartbrook Mine was designed and constructed as a high output longwall mine and has over come a number of hurdles to produce over 2.3 million tonnes for the first year of operation. A number of problems were encountered during construction which resulted in the in seam development being delayed for six months. Dartbrook people have demonstrated they can manage adversity and produce world class results. The mine has been through a very steep learning curve during 1997 and with this experience behind the mine is now producing at an annualised rate of 3.4 MTPA.

INTRODUCTION

Dartbrook Mine was designed and constructed as a new underground longwall coal mine with a planned capacity of around 3.5 million tonnes per year. The mine is located in the upper Hunter Valley 10 kilometres north of Muswellbrook NSW. Construction of the mine commenced in June 1993 and in seam development commenced in October 1994 from the bottom of the 1200 metres 1 in 8 grade men and materials drift. The longwall was installed in September 1996 and when completed in mid October 1997 2.24 million tonnes had been mined by the longwall. Results from the first longwall block were exceptional considering the challenges of steep grade, new technology and managing abnormal quantities of gas.

MINE CONSTRUCTION - LESSONS LEARNT

The Shell Board approved the project in April 1993 and when construction commenced in June 1993 the project was already six months behind schedule. Project Managers were appointed to manage the construction of the mine. The contract was written for the Engineering, Procurement, Construction and Management (EPCM) of the project. The Dartbrook management team were employed before the project commenced which provided the opportunity for input into the mine design, contracts and equipment selection. Having a Project Manager also provided the opportunity for the management team to develop operating parameters for the mine, design systems, negotiate labour agreements and recruit people. However, Project Managers are expensive at 5% of the total Capital value of the Project. The Project Managers at Dartbrook were good at building offices, workshops and coal handling facilities but did not have the experience on underground construction work. We spent a lot of time arguing with our Project Managers and also employed contractors to "keep the Contractors honest". The end result was well engineered and constructed infrastructure with an good safety, Industrial and environmental record, but the project exceeded capital estimates by around 8% and underground development commenced 6 months behind schedule.

The experience gained indicated that an in house project team could reliably manage the construction for a new mine. There is sufficient expertise available in terms of experienced mining and engineering people now in consulting roles to assemble a well balanced team to design and construct a mine.
MINE DESIGN

The concept of a thick coal seams and the opportunity to develop 4 metre roadways sounds exciting and has a number of obvious advantages but also presents a number of challenges. Larger roadways assist in keeping ventilation pressures low and permit larger equipment to be transported and installed underground but presents difficulties in the installation of roadway supports and hanging the services at roof height.

Equipment to mine at 4.0 to 4.5 metres was not readily available “off the shelf” and Dartbrook has had to design and install some very innovative development, longwall and support equipment to meet the challenges of mining thick seams.

With a coal seam that has never been worked before we realised that we needed a comprehensive data base of information to be able to plan with high confidence levels. The mine plan was designed for the best recovery of the resource and orientation of mains and longwall blocks with due consideration given to geological features, grades and the management of gas, water and spontaneous combustion. The exploration program and mine design have been shown to be the best for Dartbrook with no major shocks after 3 years of operation. The coal seam is as good, or better than anticipated with only minor faulting and dykes. We always knew that gas and spontaneous combustion were issues that were unique in a 22 to 24 metres coal section and that we would have to come up with some unique solutions.

PEOPLE - RECRIDTMENT, TRAINING AND EBA’S

The quality of people at Dartbrook has demonstrated the importance of the recruitment process. The process was time consuming, costly and involved a significant number of people.

Dartbrook personnel completed a seven week training program before entering the workplace. The value of this is bearing fruit, not only in terms of knowledge and skills but in the development of a unique safety and team oriented culture. This has led to a desire for training and qualifications, which has to be managed. The mine has been in operation for over three years and the energy and enthusiasm is still evident in the entire workforce.

During 1998 Dartbrook Mine will negotiate the third Enterprise Agreement with its workforce. The first EA was negotiated with the District Officials of the CFMEU and the majority of employees were recruited under this agreement. At the time the agreement contained some very innovative and ground breaking changes which have carried through to the second EA which was settled for a three year term. The basis of the first EA was fixed salaries, all inclusive of allowances, overtime and production bonuses. In return employees were required to work rotating shifts of 8.5 hours duration five days per week and commit to work allocated overtime. The only changes to the second EA is a step change to level 4 and 6 of the Industry work model, greater use of contractors and the overtime component reduced to allow a separate payment for weekend overtime when worked. Of interest, the second EA included the option for all employees to salary sacrifice for a fully maintained company provided car. Approximately 60% of the workforce have availed themselves of this opportunity. It is worthy of note that the concept of production bonus payments has lost its appeal and Dartbrook people are pressing good performances because of personal pride in doing a good job and in their mine. Along with the rest of the Coal Industry we are paying our people higher salaries than other Industries however I believe time and technology will develop realistic labour cost. Industrial relations at Dartbrook remain to be very strong and robust.

Although Dartbrook has maintained excellent management / labour relationships over the three years of operation, I believe that it could be enhanced by talking direct with our people without District Union intervention on critical issues. We have built up a strong relationship with our people based on two way trust which could explore mutually agreed new opportunities outside the traditional union structure.

SAFETY

Another greenfields opportunity is the setting of high standards of safety which can be an integral part of the induction training program and be written into the mines operating procedures. From the design and construction stage, Dartbrook set very high safety standards with hazops, risk assessments, procedures and Risk Management Plans (RMP’s) being
utilised as tools for the mine construction and operation. Only Contractors with good safety performances were considered for construction work on the site. Contractors have to go through a rigorous pre-qualification process to work at Dartbrook and Contractors with poor safety history are not considered for the tendering process.

External and internal safety audits are an ongoing activity at Dartbrook and maintain the safety focus. Extensive induction and training programs have assisted in setting a unique safety culture at Dartbrook.

Safety performance has been excellent for the first 2 years with LTIFR's less than 10. However 1997 was disastrous in terms of safety performance with 12 LTI's and one fatality. The fatal accident to a young contractor in January 1997 was a very sobering reminder that too much time cannot be spent on safety. The accident was a major shock to all employees who did not believe that such a tragedy could occur in a mine with such high standards and a strong safety culture.

Initially contractor safety performance during the first year of construction was very poor with an LTIFR of around 25. We have now formed long term partnerships with contracting organisations who have accepted the mine's high safety standards. The two major Contractors have retained a regular experienced workforce and employed fulltime safety and training persons. The Unions initially did not like to see regular contractor organisations on site because of the fear of Contractors taking potential jobs away from permanent workers. After the accident everyone realised that you cannot have new and unknown contractors on site and good safety is having well trained and experienced people who you can trust to maintain the standards.

RESOURCE

After an extensive exploration program, the coal seam is almost as predicted with excellent mining conditions, low water make and some localised steep grades. On the downside we have experienced higher than predicted gas levels.

The Wynn seam is Permian in age and is a bituminous coal of thermal quality. It is low in rank and has little to no swelling characteristics necessary for coking coal. Coal quality is generally as predicted with low sulphur and low ash but has high levels of Calcium Oxide (CaO) in the ash content. The higher than expected CaO provided the impetus to review the need to bring forward the construction of the coal preparation plant from year 10 of the project. The recently commissioned coal preparation plant enables Dartbrook to compete in the premium Pacific Rim markets and also to recover an additional 500mm to 700mm of the coal seam which partially offsets the washery capital cost and helps the economics of the project to look more respectable.

Exploration

At Dartbrook early drilling by the Department of Mineral Resources (DMR) and other organisations identified the presence of abundant coal reserves. Further drilling detailed the potential open-cut reserves. Prior to the commitment to longwall mining at Dartbrook, some 4 bores intersected the Wynn Seam every square kilometre. The Wynn Seam does not outcrop anywhere, nor has it been mined previously. All information about the seam came from boreholes or remote sensing.

With Shell's decision to commit to mining in April 1993 it was decided that we needed more information on an area to cover the first four longwall blocks. A further exploration program involving 24 extra boreholes was undertaken during 1993/94 to improve the knowledge of the area. Drilling across the lease and exploration areas still continues in an effort to better identify the risks associated with mining. Currently, the mine has a coverage of 18 bores per square kilometre, composed of 94 open holes and 86 cored holes.

Stress

Initial stress determinations at Dartbrook in sedimentary strata above the coal seams realised a high degree of variation of horizontal stress. The Bayswater seam, which is the immediate roof of the Wynn seam was targetted for follow up stress work. This work highlighted the relatively benign stress environment enveloping the target Wynn seam. Ultimately, this meant that the effect of stress at Dartbrook was not a critical factor to be taken into account with the mine layout.
Stress measurements were undertaken both from exploration bores and underground. In general there is an active east-west, maximum principal stress in the horizontal direction, averaging 110 degrees. However, in the environment of the Wynn seam envelop between overlying and underlying coal, the effect of the horizontal stress is minimised to such an extent that the vertical stress is predominant. The combined effects of the vertical and horizontal stresses have resulted in only minor guttering and rib crush occurring in gateroads. It appears that the vertical stress is nearly perpendicular to the seam itself which is dipping to the west.

Structural geology

To assist in determining the layout of a mine, the accurate identification of geological structures is critical. Depending on thickness, hardness and consistency. Igneous intrusions can have a devastating impact on mining as has been demonstrated at a number of longwall mines with dramatic consequences. Identification of these structures during the exploration stage, allows the opportunity to plan for them, rather than deal with them as an emergency situation.

Techniques such as surface and aeromagnetic surveys, have a proven track record of accurately locating structures. At Dartbrook several surface and aeromagnetic surveys found intrusive anomalies were orientated in a north-east direction. Further investigations involving costeaming, helped to identify these structures. Intersecting dykes near parallel to a longwall face could cause difficulties with extraction. Intrusions oriented at an oblique angle to the faceline will minimise longwall delays.

Dartbrook Mine has a nominal borehole spacing of 250m. Uncertainty surrounding possible faulting (at the time) resulted in limited input into the mine design. Clearly, faulting could cause difficulties with mining. However, the extensive coal overlying and underlying the mining horizon would minimise these difficulties. The issue becomes one of coal quality rather than structural constraint associated with mines with stone roof and floor.

Evaluation and examination of borecores identified the presence of extensive jointing at Dartbrook. Determination of orientation became a priority during exploration. The RaaX photography method accurately identified the orientation of the joints. The jointing is ubiquitous and trends relatively consistently at 110 degrees. Underground measurements in the first workings confirmed the exploration data. The orientation of the longwall blocks took the jointing into consideration. With the thick seam and the jointing in mind, design of the longwall supports incorporated face spags (flippers). The flippers support the top of the seam to reduce spalling, and protect the operators from injury.

Coal quality

The mainroad development is in a westerly direction along the length of the southern lease boundary. These roadways experience relatively higher ash levels and lower seam height to facilitate the longwall extracting the premium quality coal.

Roof and floor geology

In the current longwall mining area at Dartbrook, the typical roof comprises of about 14m of coal. While loading from the overlying sediments is unlikely, a ‘risk averse’ policy has selected longwall shields rated at 913 Tonne yield. The roof support density is 118 tonnes per square metre.

The mining floor comprises of 300mm of tuff which is quite competent with a compressive strength of between 14 and 28 Mpa. The Mine Technik Australia (MTA) longwall face supports at Dartbrook have a base lifting capacity to assist in keeping the face on the appropriate mining horizon. The Coal is very strong and both longwall one and two advanced around 15 to 20 metres past the installation roadway before the coal roof started to fail. No problems were experienced with wind blasts of gas inrushes on both longwall startups, although precautions were taken to minimise and manage these risks.
Gas

All coal seams in the Dartbrook mining lease contain a seam gas mixture of Methane (CH4) and Carbon Dioxide (CO2). In-situ gas contents range from 6.5 to 11 cubic metres per tonne in the Wynn seam. Gas contents for the upper seams are generally less. Carbon dioxide is the predominate gas with CO2/CH4 ratios ranging between 90:10 and 60:40.

Prior to the commitment to mine at Dartbrook, the gas data obtained from boreholes indicated the necessity for gas drainage. A variation between the surface exploration standard and the underground 'quick crush technique gave a discrepancy of between 1 to 2m³/tonne. Absorption of CO2 into the acidified brine caused this error. To make a more satisfactory comparison between the exploration and the underground data, the mine employs the quick crush technique in current exploration work.

Reservoir permeability, diffusivity, porosity and sorption isotherms have been determined by laboratory testing. In situ gas pressure has been directly measured. This work indicated that the coal has a fairly low permeability for its depth and is under-saturated with gas.

Gas reservoir modelling has been undertaken to determine gas emission rates upon development and longwall extraction of the Wynn seam. Development gas emission was modelled using the SIMED simulator from the Commonwealth Scientific and Industrial Research Organisation (CSIRO) which uses a two phase 3D multi-component simulator. The indicated rib emission rates range from 20 L/s to 50 L/s per 100 metres and would be markedly influenced by seam permeability and gas content. Based on predictions for a 4 kilometre roadway, two heading gateroad with 8.6 cubic metres per tonne disoarable gas of 80% CO2 and a permeability of 0.44 mD, a ventilation requirement of 55 cubic metres per second is required.

Longwall gas emission has been predicted using various empirical techniques. The modelling is limited due to most techniques being particular to methane gas. However, gas reservoir estimates indicate between 40 and 55 cubic metres of gas per tonne of coal to be mined, is contained in the immediate floor, working section and roof of the Wynn seam. It was estimated that without gas drainage on longwall one a ventilation requirement of 232 cubic metres per second across the longwall face would have been required to meet the statutory limit of 1.25% CO2.

The actual seam gas content was 1 to 2 cubic metres per tonne higher than predicted from the borehole data, however the experience to date has demonstrated that the permeability is higher than expected and the gas drains relatively easy. In fact a little too easy as the cumulative rib emissions for a four kilometre gate road (in 2 klms and out 2 klms) culminated in general body readings at the outbye end of the returns of around 1 to 1.20% of CO2.

Two weeks of development was lost due to gas levels running around the legal limit of 25% CO2 Extensive rib drilling was carried out to reduce the rib emissions.

Initial block drainage was trialed using directional drilling to drill 450 metre holes across the block parallel with the gateroads. When gateroads were advanced 450 metres cross holes were drilled across the block at initial spacing of 20 metres which reduced to 10 metres approaching the face installation roadway.

The cross holes were drilled in the seam section, downholes into the 4 metre section below the 300mm tuff floor band and up holes into the 12 to 14 metre coal roof section. Roof holes were determined to be of poor value with only 30% capture after 6 months due to impermeable carbonaceous bands. Holes were branched off the up holes perpendicular to the stratification which only marginally improved the capture rate. All holes drilled were by directional drilling using a number of drill rigs including LM35, LM55, Boyles and Diamec 262 and Diamec 252. As at November 1997 approximately 300 kilometres of gas drainage holes have been drilled.

With the pre-drainage of the block modelling demonstrated that production would be limited to around 50,000 to 55,000 tonnes per week of production. The Department of Mineral Resources provided special dispensation to allow a special fenced off toxic return up to 3% carbon dioxide. This was later raised to 3.5% CO2 as the mine production had plateaued at around 60,000 tonnes per week with an average of 20 hours per week of lost production due to high CO2. As soon as the exemption was given to mine up to 3.5% CO2 the longwall production levels increased to around 75,000 tonnes per week.
The ventilation system adopted for Longwall one was three intakes and one toxic return with two intakes on the maingate side and one intake up the blockside tailgate and a back return system. To ensure the back return remained open, initially two rows of 6 metre flexi bolts were installed. A number of 900mm auger holes were drilled through the tailgate chain pillars with minimal impact. The last 500 metres of the tailgate of Longwall 1 were supported with 900mm aerated concrete cans to maintain the back return. The cans were easily and safely set and maintained an excellent back return.

The longwall goaf was producing CO₂ and methane at the rate of 5 cubic metres per second and it was decided from the early modelling that post drainage of gas would be required if Dartbrook was ever to be able to meet the statutory limit of 1.25% CO₂. Vertical goaf wells were drilled 70 metres from the installation roadway and approximately 70 metres from the maingate side of the block and connected to a high pressure exhaust fan. Research found there was no experience with goaf wells extracting carbon dioxide. Goaf holes have worked well at a number of mines in Australia and the USA extracting methane but greater suction was required to overcome the buoyancy effect of CO₂. The holes eventually proved to be very successful particularly on the tailgate side of the block at 200 metre intervals exhausting at the rate of 1 to 1.5 cubic metres per second of CO₂ and methane per hole.

With two pumps operating, up to 2 cubic metres per second of gas was exhausted which dropped the gas levels in the toxic return by around 0.3% CO₂. Longwall 2 has adopted a homotropal ventilation system which is a mirror image of Longwall 1. The Department of Mineral Resources (DMR) has given approval for longwall 2 based on additional post drainage facilities being adopted to reduce the 3% CO₂ limit in the toxic return to 1.25% by the completion of Longwall 2. As well as post drainage from vertical holes post drainage from seals behind the face line have proved successful. The Longwall 1 face was ventilated with 95 to 100 cubic metres per second which is 3 to 4 times the ventilation quantity on the average longwall face. Longwall 2 is currently ventilated with around 70 cubic metres per second.

The four kilometre Hunter Tunnel construction experienced high methane emissions and water ingress associated with a synclinal structure. With the exception of two major dykes (2m and 9m) mining conditions in the tunnel were generally good. Water flows peaked at around 60 litres per second and produced uncomfortable physical working conditions. Water flows are now down to an easily manageable 12 to 14 litres per second. The gas content in seam was as predicted at around 4 to 5 cubic metres per tonne, however the fractured ground associated with the synclinal structure provided a conduit for the high gas emissions. The Hunter Tunnel was holed on the 3rd January 1996 and the 1800mm conveyor was commissioned during Easter 1996.

**Spontaneous combustion**

Subbituminous thermal coals of the Hunter Valley have a history of spontaneous combustion events in both longwall and bord and pillar mining operations. In order to determine the propensity of the coal seams at Dartbrook to spontaneous combustion, a series of laboratory tests were undertaken and expert advice sought.

Based on the four tests

- Relative ignition temperature;
- R 70 index;
- Initial rate of heating; and
- Total temperature rise.

It was decided that coal seams in the Dartbrook lease had a medium to high propensity for spontaneous combustion and this would have to be a major factor in the mine design. The risk rating used indicated that the thickness of the seam and the amount of coal left in the goaf to be of particular importance.

The longwall ventilation system was designed as a relatively simple “U” type system with no goaf bleeders. It was also recognised that effective seals and an accurate and reliable monitoring system would be a pre-requisite for safe longwall mining. When one million tonnes of coal was mined from Longwall 1 approximately 3.5 million tonnes of coal remained.
in the goaf. With the normal oxidation rate of coal we were running at 120 to 150 litres per minute CO make after 5 months of longwall mining. The Dartbrook conditions are very unique and cannot be compared to any conditions anywhere in Australia or overseas. Expert advice suggested that CO make could not be utilised as an accurate indicator of spontaneous combustion. The racking of CO levels and Graham's ratio (GR) and action response plans were incorporated in the Dartbrook Underground Environmental Management plan (DUEMP).

During April 1997 the CO levels were stable at around 400 ppm at the tailgate intersection of the installation roadway approximately 500 metres behind the faceline. On April the 15th the CO level rose from 450 ppm to 600ppm within 6 hours. At this time the Graham's ratio had risen to 0.62. All persons were withdrawn from the mine at 3.30 pm in accordance with the Spontaneous Combustion Management Plan (Action response plans) and a series of bag samples were taken and analysed by chromatography in Muswellbrook, Maitland and the Mine Rescue Mobile Laboratory to confirm results. The results indicated 80 to 100ppm of hydrogen from the tube bundle point at the edge of the installation roadway. At around 11.30 PM the CO at the installation roadway was 690 ppm CO with a GR of 0.72. Ventilation pressures were reduced across the longwall face by regulating the longwall return. The CO readings started to immediately decline and after 24 hours had settled back to normal. It was shown later than hydrogen was found in a number of vertical boreholes drilled from the surface for exploration and for hydofrac trials. Helium gas was also discounted from the readings to show a true hydrogen reading. It was later suggested that hydrogen may be a seam gas given off at a lower oxidation temperature.

A similar incident occurred during June where people were again withdrawn from the mine. The cause was believed to be the absence of stoppings in the toxic return inbye the last ventilation cut-through inbye the tailgate. The DMR would not allow these seals to be installed by Mines Rescue teams under breathing apparatus at concentrations above the statutory working limits. After the second incident drop doors were erected in the toxic return every 200 metres which maintained the ventilation fringe shallow behind the longwall face line. No further problems have been encountered since.

Hydrology

Hydrology studies were undertaken to determine water make on development and from longwall goaf caving. Although these studies produced a wide range of results Dartbrook adopted a risk averse strategy to install sufficient pumping to withstand the worst case scenario. Despite the high quantities of water experienced during construction of the Hunter Tunnel, the current working area and longwall goaf is only producing around 1.5 litre per second. The only water created in the mine workings is typical nuisance water, which we create during the mining process.

TECHNOLOGY

Working at 3.9 metres height and longwalling at 4.5 metres presented some interesting challenges and to some extent was underestimated. Unfortunately you cannot procure standard equipment off the shelf to operate at this working height. The question was asked by many people, "why don't you mine at around 2.8 to 3 metres height to match the mining equipment available and ramp up from the gate roads for the longwall like the US mines do". We took the view that higher is better for ventilation efficiency, and to get larger pieces of equipment underground. We saw opportunities to design equipment and systems to mine and install services at this height and our people have designed and developed some very innovative solutions to problems. The 3.9 metre high roadways allowed conveyor belts to be installed against the roof, which allows for machines to move under the belts and makes belt cleaning less labour intensive. It also allowed for gas drainage drilling of the longwall block.

In the current working area of the lease the seam section comprises of three working sections, the Wynn upper A, B and C sections. The Bayswater B seam section also joins the top of the Wynn section and results in a combined coal section of 18 to 24 metres. Dartbrook typically mines 3.9 to 4.0m on development and mines up to 4.5 metres with the longwall to optimise coal quality. Working at heights greater than 3 metres presents a whole range of problems. How to install conveyors, pipes, cables, etc at roof height. A number of very innovative designs were developed to manage the tasks. A platform 1.5 metres of the ground was installed on the Voest Alpine ABM20's to provide a safe working area for operators and ease of installation of supports and ventilation mono-rails. Purpose designed platforms were manufactured for the
installation of pipes and the 1800mm main conveyors. A double crawler mounted tailpiece was designed with a conveyor installation platform to dispense with the development coal and install the 1500mm gate road conveyors. The 3.9 metre height in the development units allowed the opportunity to install a specially designed ventilation and cable management system.

DEVELOPMENT SYSTEMS

Dartbrook’s operating structure is based on 5 days per week and 3 by 8.5 hour shifts per day. Initially, our plans were to have 2 production shifts per day and one service shift per day. The idea of the service shift was to not only complete the equipment maintenance orders, it also completed all the mining related activities such as belt extensions, DCB moves and installs 60 metres of supplies on the continuous miner ready for the production shift to have a press button start.

Equipment was designed to enable the conveyor and services to be extended by short increments on the service shift. Unfortunately the development equipment available on the market did not suit the operational aspects of the 2 production and 1 service per day concept as well as the 3.9 metre working height. The equipment size and complexity of the tasks did not provide the 10 quality shifts of production per week desired, so a cyclic system was introduced to allow maintenance activities to fit with the completion of a development cycle.

A new mine has the opportunity to install the latest technology available to the Industry. Dartbrook selected three Voest Alpine ABM 20 Continuous miners with four hydraulic roof bolters and two rib bolters. The ABM20’s have an installed horsepower of 542 KW with a 270 KW cutter motor, 2 by 36 KW conveyor motors and 2 by 100 KW pump motors. The TRS canopy provides a footprint force of 2 by 200 KN. The ABM20 loads at the rate of 25 tonnes per minute. Roof support is via four by 4000 series semi-automatic Hydramatic Engineering roof bolters and rib support is provided by 2 by series 5000 hydraulic rib bolters. The support pattern is four 2.1 metre roof bolts at 1 metre centres with 6 bolts through gate road intersections and 4 to 6 rib bolts per metre of advance. Typical mining conditions are shown in Fig. 1

During the evaluation process for suitable development machines for Dartbrook, the Voest Alpine ABM20 Miners was seen as the only tried and proven machine with the capability of installing roof and rib support contiguous with the cutting activity.

Considerable redesign of the machine was undertaken with Voest Alpine to permit the machine to cut to 4.3 metres height and to install an on board sizer. Although the Wynn Seam at Dartbrook has relatively strong coal we did not want operators working between large single pass machines and 3.9 metre high ribs. A work platform was designed for the operators to stand 1.5 metres off the ground, be able to touch the roof and have all tools and supplies at their finger tips. The excavation of an overcast in shown in Fig. 2.

The platform was designed for operator convenience with a bolt box on either side of the machine located immediately behind the operator. Baskets were designed for chemicals, mono- rail fittings etc. Racks were fitted along the sides of the platform rails for spare drill steels and mono-rails. The ABM20 can carry sufficient supplies for 60 metres of roadway drivage. What was considered as one of the hardest and most hazardous jobs in a coal mine is the easiest job at Dartbrook.

Horizon control is managed by setting the working height on the Penpeck radio control system and referencing from the tuffaceous floor. Although the machine is very close to 90 tonne in weight it has low ground pressures and does not break up the floor on intersections. The ABM20’s from day one have averaged approximately 5 metres per cutting hour and at times have achieved over 7 metres per hour.

Joy 15SC-32 shuttle cars were chosen basically because there were no other shuttle cars on the market and after mixed results with mobile conveyors at Capcoal in Central Queensland we opted for tried and proven coal clearance systems. Joy Manufacturing were contracted to supply shuttle cars that hold 15 tonnes of coal and after considerable modifications we now have a 15 tonne car which permit us to cycle two cars per metre of advance. The seat on the 15SC’s were raised by 500mm which significantly improves the visibility of the driver.
With a new project and a "clean sheet of paper" a systems approach was used to design supply systems, ventilation and development systems. Materials handling is one of the most labour intensive activities and has the greatest potential for accidents in coal mines. Dartbrook designed a supply system of delivery from the manufacturer to the Continuous miner which involves the supplier loading and delivering boxes full of roof and rib bolts and trays that hold 60 straps. These are unloaded onto the ground or direct onto heavy duty trailers on the surface. Eimco EJC 130 LHD’s tow the heavy duty trailer to the development panel were bolt boxes and strap trays are transferred to the ABM20 by a QDS hyab crane or by Eimco platforms. A panel support vehicle (PSV) was designed for this purpose and is currently away for modifications to install the bolt boxes onto the miner. The PSV is a crawler mounted machine with a flat deck and a hiab crane arrangement.

Face ventilation is provided by a 720mm diameter monorail mounted ventilation system which is attached to, and advanced by the ABM20 continuous miner and retreated by using a crawler mounted fan. The vent system also carries the
miner cable and 50mm water and compressed air hoses which has resulted in substantial time savings in cable and hose movements. The vent system is a combination of fibreglass and flexible ducting which runs on 1.5 metre lengths of round mono rail.

To enable the 1500 mm gate road conveyor to be installed at roof height and in short increments on the service shift a Development Tail End (DTE) was designed jointly by Dartbrook / ACE and MTA. It is double crawler mounted and has two MBS bolters mounted on the front above the tailpiece to install roof bolts for the belt structure. The unit has side shift and levelling facilities for belt alignment and the unit is aligned by lasers and perspex site boards mounted on the side of the machine. Two lifting platforms are on the outbye end of the unit to enable the belt structure to be installed at roof height. Initially a DCB mounted on a sled was towed behind the DTE with a reticulation cable in a basket behind it. We now have the DCB maintained back at the section transformer and a mono-rail mounted cable and ventilation management system, which advances and retreats with the miner. The ventilation fan is crawler mounted and is used to advance and retreat the system. An outbye compressed air driver moves the reticulation cables outbye the fan. A 30 metres structure pod is positioned behind the DCB sled under the belt ready for the belt extensions on the service shifts.

**EXPERIENCE OF THE FIRST LONGWALL BLOCK**

The Dartbrook longwall is currently mining the second longwall block. With 18 to 24 metres of coal we have a very large reservoir of gas which have been extensively drilled but experience on Longwall 1 showed that gas capture above the mining horizon was only around 30%. Within the goaf envelope there is a total of three seams with a combined gas content of around 50 cubic metres per tonne extracted. It is difficult to accurately model how much gas is liberated from the goaf but experience from Longwall 1 shows that around 5 litres per second of gas has to be managed by the ventilation system and post drainage from vertical goaf wells and from seals behind the longwall face.

The Wynn seam at Dartbrook is longwalled at between 4.0 and 4.3 metres in height. The first longwall block was 2.2 kilometres in length and Longwall 2 is 2.4 kilometres. The longwall's retreat to the rise with goaf water pumped by a borehole pump at the back of the blocks. Longwall 2 is ventilated by a simple "U" type homotropal ventilation system with no bleed system due to the risk of spontaneous combustion. The first longwall face was installed on a grade of 1 in 5 for 80 metres of the 200 metre face line. The steep grade was managed without problems. We took the view that steep grades will be experienced in several areas and we need to learn how to manage them from the first longwall. The initial faceline is shown in Fig. 3.

![Fig. 3 - Longwall 1 faceline on startup](image-url)

The longwall was supplied by Mine Technick Australia (MTA) and is 200 metres in length with sufficient horsepower to extend to 250 metres in the future. The Maingate equipment is installed on three self propelled crawler mounted trailers, which are retreated under the roof, mounted belt outbye of the Longwall Tail End (LTE). The LTE is a MTA/ ACE / Dartbrook designed unit which is skid mounted under the BSL and Crawler mounted on the outbye end and elevates the coal to roof mounted conveyors. The LTE has structure recovery platforms on either side at the outbye end where the
structure is recovered and installed in 30 metre structure pods ready for installation in the gate road panels. The specifications of the longwall equipment is shown in Appendix 1. Fig. 4 shows Dartbrook roof supports and AFC.

Fig. 4 - Dartbrook roof supports and AFC

**Longwall changeover**

The first longwall changeover was planned for 20 days and this was not achieved due to delays with the tailgate drive and crusher overhauls and a number of electrical problems. The longwall equipment move was actually completed in 15 days but we did not start cutting coal until 28 days after we started to bolt up the face. The Geogrid mesh, 14 metres in width was used in conjunction with one can and one timber crib per support during shield recovery. This process saved at least 4 days on the recovery and provided a safety barrier to prevent the goaf from flushing into the faceline during the recovery of shields.

**CONVEYORS**

For a high capacity longwall system the mine required a reliable high capacity coal clearance system. Like most modern underground mines in Australia and the USA, Dartbrook elected to not include a surge bin or bunker in the coal clearance system.

The main conveyor system is 1800mm in width operating at 4 metres per second speed. This gives a designed capacity of 4,200 tonnes per hour or a spill capacity of 5,300 tonnes per hour. The Hunter Tunnel conveyor hauls coal from the mining area to the west of the Hunter River 4 kilometres underground to the coal handling facilities on the eastern side of the New England Highway and main Northern Railway line. The first conveyor from the surface (HT01) is powered by a 2.2 MW drive and hauls coal up a 600 metre long 1 in 5 drift. The second in line conveyor (HT02) is 960 KW and roughly half way along has a 960 KW tripper drive. The drive units were designed by Dartbrook and manufactured by ACE Conveyors with the winches provided by Nepean Engineering. The gearboxes were supplied by Flender and CST drives by Dodge.

The maingate conveyors are also powered by the same drives as the mainroad conveyors (3 X 320KW power packs and CST drives. The gate road conveyors also have a tripper drive installed (960KW) to permit the haulage of longwall coal upgrade for 2.4 km. The maingate conveyors are 1500mm wide and have a designed capacity of 3200 tonnes per hour. After some compatibility problems with the CST software and minor winch programming problems the belts are settling down well. Coal on the surface is delivered to a 1100 tonne bin or can be diverted by a lifting boom to a 50,000 tonne
emergency stockpile. The coal from the bin is fed through sizers and Syntron feeders, through to stackout on to two 200,000 tonne stockpiles. Coal can also report direct to the 1000 tonnes per hour washery.

PLANT CONTROL SYSTEM

To understand and manage equipment and the mining environment a comprehensive and responsive control and monitoring system is a must. After evaluating a number of systems a decision was made to install the Windows based Citect Plant Control System. The system is working extremely well and is very "user friendly". Dartbrook has a control centre manned 7 days per week to monitor the Citect system and acknowledge and respond to alarms.

Underground monitoring:

- Maihak Tube Bundle gas monitoring system;
- AMR environmental telemetry system;
- Continuous miners;
- Conveyor belts;
- Fans;
- Power distribution 66/11 KV;
- Gas drainage plant;
- Longwall;
- Vertical goaf gas drainage pumps; and
- Dewatering pumps.

Surface monitoring

- Compressors;
- Water reticulation;
- Sewerage plant and water treatment plant;
- Irrigation system
- Office security and fire alarms;
- East site coal handling plant;
- East site bins and conveyors;
- Stackers and reclaimers;
- Train loading bin and system;
• Weather station

The system is PC based with Allen Bradley Programmable Logic Controller (PLC) and Small Logic Controller (SLC) equipment to provide data on a regular basis, which is collected by the Citect software for display to the control room operators and a number of site offices including the mine managers office.

FUTURE PLANS

Dartbrook is currently undertaking exploration work to the west and north of the current mining lease to extend the reserve base. Dartbrook needs to produce in the vicinity of 3.5 million tonnes per annum to provide the shareholders with an acceptable return on capital employed. Gas drainage is a large cost impost on any mine but at Dartbrook it represented around $6.00 per tonne in 1997 which has to be reduced for the mine to survive. Gas drainage is time dependent and we now have pre-drained for the next two longwall’s beyond Longwall 2 and have reduced our gas drainage effort by one third. A lot was learnt on Longwall 1 which has allowed us to reduce the amount of pre-drainage with optimum drilling patterns and with the experience of successful goaf drainage holes. We believe that we now understand the parameters for effective pre and post gas drainage for longwall mining and understand the fundamentals to manage the risk of spontaneous combustion.

SUMMARY

Dartbrook Mine has not reached world class as yet but the basic elements are in place to provide a safe, efficient and profitable business.

The same level of enthusiasm and energy is still evident at Dartbrook that existed when the mine started 3 years ago. A high level of trust exists between all parties at Dartbrook and this helped through the early operational problems of learning to work at heights and the development of suitable equipment and systems to mine a thick coal seam full of gas and achieve high advance rates on development and planned tonnages of the longwall.

In summary the advice to any new Project Manager would be to allow sufficient time to gather and interpret an extensive data base of knowledge on the resource and to establish the best mine plan and cost structure. Invest money and time to employ the best people, train them well and have the best possible labour agreements in place. This is the best chance you will ever get to set the mine up on a stable foundation and take all the advantages presented by a greenfields opportunity.
APPENDIX 1

LONGWALL SPECIFICATIONS

MTA:

- Shearer Initiation;
- 116 X Two leg 913 tonne shields;
- 1.75 metres wide shields;
  Range 2.2 metres to 4.8 metres;
- Face support mass 26.56 tonnes;
- Gate road support mass 27 tonnes;
- High strength steels 700 UTS;
- Support density 118 tonnes per sq metre;
- Full automation and data transmission to surface through Citect system;
- PM4 electro hydraulic support control units (SCU's);
- Leg pressure and push ram displacement transducers;
- Water sprays in canopy's; and
- Rear walkway in canopy down to 3.3 metres operating height

Shearer

- Long Airdox (Electra 1000);
- Cutting range 2.5 m to 4.5 m;
- Installed power 1332 KW, 3.3KV;
- Cutting drum diameters 1.9m to 2.5 m;
- 2 X 500 KW cutter motors;
- 2 X 56 KW DC haulage motors;
- 200 KW lump breaker;
- 20 KW hydraulic pump motor; and
- Hiab type crane on MG end.
Pumping system

- MTA;
- 3 X 275 l/min @ 350 bar Hauhinco hydraulic pump sets mounted on a crawler mounter trailer;
- 250 KW motors on each pump;
- 10,000 litre stainless steel tank with 1,500 litre mixing tank;
- High pressure shearer water pump mounted on crawler mounted trailer with tank.

AFC

- MTA
- Width 1150mm:
- Capacity 3200 tonnes per hour;
- Twin 42 mm Compac link chain;
- Automatic chain tensioning at Tailgate;
- 2 X 800 KW motors driving CST’s at Maingate and Tailgate;
- Slow chain speed running; and
- Provision for fitting of pan tilt cylinders.

BSL

- MTA;
- 1500mm nominal width;
- Twin 34mm link chain;
- 350KW drive motor;
- Slow running device fitted; and
- Full length dust suppression system

Crusher

- MTA (Westfalia Becorit);
- 1800 mm width;
- High inertia impact roller; and
- Size coal to minus 150mm.
Oaky Creek Coal - Improving Productivity

B Nicholls\(^1\) and P Lynch\(^2\)

LOCATION

Oaky Creek Coal Pty Ltd. currently mines high quality coking coal from both underground and open cut operations at its Oaky Creek Mine. Coal is transported by rail to the Dalrymple Bay Coal Terminal south of Mackay and to the Gladstone Coal Terminal. Oaky No.1 Underground Mine, Open Cut and Highwall Mine have concentrated on seams from the German Creek Formation, one of the major productive coal measure formations in the Bowen Basin Coalfields. (Fig. 1) location map.

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\(^1\) General Manager, Oaky Creek Coal Pty. Ltd.
\(^2\) Oaky North Manager, Oaky Creek Coal Pty. Ltd.
BACKGROUND

Oaky Creek Coal is located 100km from Emerald in central Queensland. It mines the German Creek, Corvus, Aquila and Pleiades seams to produce high quality coking blends. Mining started in 1982 with coal being exported to steel mills in Japan, India, SE Asia, Europe, North Africa and South America. The open cut mine was established to produce 2.5 Mt a year and is currently producing approximately 1.5 Mt of product coal. Increasing stripping ratios has driven the requirement to go to underground mining operations. Highwall mining has been practised from several abandoned highwalls during the past three years.

The N1 underground mine was started in 1989 off an open cut highwall to expand the total coal capacity from the site and improve coal quality when mixed with coal from the open cut operation. The underground has suffered from a chequered history, with the longwall being development constrained at the end of the first block after commencing early in an effort to maintain cash flow. Flooding from surface water inflow in 1996, a lack of underground experience in the initial project team coupled with a staff turnover around 30% pa have contributed to a history of poor performance.

The mining conditions to date in the N1 underground are good, with good roof, low in-seam gas make, minimal faulting and consistent (±3m) working section. Gas make within the mine is low but pockets of H2S restrict mining rates when longwall mining in the affected areas. Annual tonnage has varied since the commencement of longwall mining due to a number of factors, however it is currently running at three million tonnes per year increasing to almost four million from mid 1998.

Following lengthy negotiations and final acceptance of an Enterprise Bargaining Agreement (EBA), production records for a shift, 24hr, weekly and monthly periods, were broken. In September 1997 production was 390,000 ROMt coal from the No. 1 Underground mine. A contract is to be awarded for longwall gateroad drivage (approx 30,000m) for the North East section of the mine to ensure continuity of longwall production and cash flow.

Oaky Creek Coal (OCC) has recently signed an EBA for both the N1 Mine, Oaky North and Surface Operations in which it is stated "... that the company may employ outside contractors and in whatever work it determines at its sole discretion ..." with minimal qualifiers.

This reflects the OCC operating philosophy and what the authors believe should be a standard management prerogative throughout the industry.

ROM coal is sourced from six locations. Three open cut pits and three underground mines, two of which currently operate longwalls.

Four seams varying in thickness from 800mm to 4.5m. are mined from the open cuts. The seams and thickness are set out below and a stratigraphical section is shown in Fig. 3.

<table>
<thead>
<tr>
<th>SEAM</th>
<th>THICKNESS</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pleiades</td>
<td>800mm</td>
</tr>
<tr>
<td>Aquila</td>
<td>800mm - 1,800mm</td>
</tr>
<tr>
<td>Corvus</td>
<td>600mm</td>
</tr>
<tr>
<td>German Creek</td>
<td>1,500mm - 4,600mm</td>
</tr>
</tbody>
</table>

The main thrust in the open cut operations is the development of increased Aquila seam quantities required for product blending.

German Creek seam open cut reserves will be exhausted by September 1998 when the first dragline will be shutdown.
Increasing volumes of German Creek coal will be mined from the underground mines predominantly from three longwalls. The Oaky North project will commence longwalling early in 1999.

Fig. 3 - Typical stratigraphic section
The projected Run of Mine (ROM) production profile is shown on (Fig. 4) ROM Production Graph

![Graph showing ROM production profile]

**THE TURN AROUND STRATEGY**

Operational constraints - were many varying from increasing strip ratios in the open cuts with reducing equipment availabilities to very short (<1,000m) longwall blocks which can be extracted in less than three months.

Lack of confidence by the shareholders in the ability of Oaky creek Coal Pty Ltd to show any return on the investment had created a cost constraint downwards spiral with increasing downtimes on major equipment.

No real commitment had been made as to the future direction of the business by either management or employees until late in 1996 when the value of the resources was realised. The enormous task of removing industrial constraints brokers was recognised, assessed and seen as being achievable albeit at potentially horrendous cost.

Industrial constraints - revolved around an extremely militant group who basically controlled all the workforce and were effectively the de-facto managers of the business. When challenged on their attitudes and the continued industrial losses (46 days in 18 months to November 1996) the custom and practice argument reigned supreme.

The strategy was put in place to try to negotiate an EBA for the site but differentiating between underground, open cut and CPP operations.

Some 362 customs/practices and union policies were tabled almost all of which had to be removed if the business was to have a future. Given the quality of some of the people in the workforce and the value of the resource plus the level of the investment, the challenge had to be met up front. Introduction of the Federal Workplace Relations Act on 31 December
1996 enhanced our resolve to turn the tables, remove the industrial constraints and put the business into a profitable position. Financial year 1995/96 had resulted in a $30 million loss. Financial year 1996/97 resulted in a $300,000 profit.

On top of these two major constraints we had a planning dilemma which, simply stated, was there was no integrated planning in place across the site. Each production area basically did its own thing and the hope was it would all come together in the end. That philosophy cannot work and will not work and so enormous amounts of time, money and energy were being wasted by not having an integrated planning procedure for the six mining operations and the four raw coal products. This had to be addressed urgently and a Planning and Engineering Manager was appointed in January 1997.

THE WAY FORWARD

The two immediate things to fix were the industrial climate and the lack of co-ordinated planning. The quality of the OCC coking product is such that marketing was not a problem.

Integrated Planning

Subsequent to the appointment of the Planning and Engineering Manager the site operational planning was gradually brought under one banner to co-ordinate the site activities, basic stuff but very necessary. A major task which required a complete re-structure of the planning function.

Result - an immediate improvement in mine planning and therefore costs and efficiency.

Mine activities were integrated to enable optimisation of the Coal Preparation Plant to ensure product quality through planned raw feed availability.

Management to manage

The de-facto management by the lodge executive had to be removed and the management team had to take the reins of management and leadership, probably for the first time for a long time at Oaky Creek Coal. This meant entering into serious negotiations to achieve separate EBAs across the site and tackling unacceptable practices head on.

Persuading middle management that we intended to do this was at first difficult - how could it be done? - it can't be done!! were common questions and thoughts from a team who up until early 1997 had almost no control.

To convince people two things happened:

1. the workforce and union were told up front we intended to regain management control and manage, with or without their co-operation and

2. every issue where we had the right to exercise management control and decision making was taken on and settled either at the mine, or usually in the AIRC. During the negotiating period for the EBAs, we were also selective to make sure we took on issues we believed we could not loose.

Correction of the industrial dilemma followed on from the insistence on our part to manage the mine. At no time were we attempting to remove union involvement from the operational activities, conversely, we were embracing participation at all levels to improve the systems in use and give operators more say in how we did things. We publicly acknowledged the fact that there is (and always has been) an enormous amount of talent in the workforce that needs to be harnessed. We are actively promoting, in all facets of the OCC operations, involvement of our production and maintenance teams. The non-negotiable issues taken on in the EBA discussions were:

- no minimum manning;
- no demarcation;
- unfettered use of contractors;
- no overtime limitations; and
- removal of all past customs and practices.

Whilst the EBAs proved extremely difficult to achieve, the results since acceptance of the EBAs, initially at the N1 underground and surface operations and later at Oaky North, has given us confidence that the operations can now be developed into a highly productive and profitable operation. The underground performance (Fig. 5) with the reduced manning level and use of contractors is now amongst the best in Australia. The Open Cut mine and Coal Preparation Plant have their own management structures and performance bonus arrangements and are continually benchmarking against other Australian mines.

Fig. 5 - Underground performance

The workforce and union are actively encouraged to contribute to the improvements in systems, standards and procedures without being given any de-facto management rights.

Importantly, the management team in the operations areas are in continual communication with all the teams informing and involving team members about what is going on and why. We still have a long way to go to be “good” at communications face to face but it is essential that we succeed in this area.

By implementing a renewed safety culture, MIMSafe (MIM safety and health strategy) and NOSA standards, the whole site has improved dramatically in standards and the culture of OCC is changing rapidly to one that will not accept poor standards or work practices. Again the change is welcomed but not fast enough.
Achieving the objective of correcting the industrial dilemma has been difficult, costly and unfortunately cost some 80 jobs. The actions had to be taken to get OCC to where it now is - productive and efficient and providing shareholders the confidence to further invest in the underground operations. The Oaky North Project being approved in December 1997 at a cost of $218m is evidence of that confidence.

ADDITIONAL CASH GENERATION STRATEGIES

Longwall punch mine

After lengthy negotiations during 1995/96, an area of the southern part of the lease was sub-leased to Thiess Namoi Joint Venture to develop a longwall punch mine off the Crinum pit highwall. Longwall operations commenced in October and the mine is designed to produce 1.2mt ROM product for washing in the Oaky Creek Coal Preparation Plant. This provides not only additional volume and cashflow but also the opportunity to market a third coking coal product. This mine at 1.2mtpa has a projected life of 6 years.

Increased Oaky No. 1 Longwall performance

Considerable effort and management time was put into ensuring the main cash generator at the mine, the N"1 mine longwall, was driven as hard as possible. This has been done successfully since the start of 1997 (Tables 2 through 5) Longwall Performance) and this longwall is now one of the top performers in Australia using 9 year old supports on short (200m x 1000m) blocks.

Development into the long North East blocks is currently being done partly by Oaky Creek and partly by contractors. Commencement of the first long panel LW14, in July 1998 will enable the N"1 mine production to rise to 4mtpa.

Three Dragline strategy

The decision to cut back to 2 draglines was reversed in late 1996 and three draglines have been kept swinging in three different open cut pits. Whilst not highly productive or highly cash positive the additional coal has generated extra cash. Improvements in mine scheduling, employee flexibility, operations planning and the acceptance of an EBA in July 1997 has enabled Oaky Creek Coal to continue this strategy until later in 1998 when the first Dragline will be shut down. This was planned to coincide with the commencement of longwall operations at Oaky North when a substantial increase in underground ROM volume is projected. The two remaining draglines in the Aquila seam will continue after mid 1999 only if operating margins are acceptable.

Medium and long term plans

Implementation of EBAs and improved work practices was and is essential to the long term success of the business. People issues are critical and a highly motivated workforce is the most important factor in the whole success equation.

The confidence of the shareholders in the current operators and management is obvious from the new commitment to Oaky North, the upgrade of the Coal Preparation Plant and the potential development of the South Eastern area of the No. 1 mine with an investment in another longwall.

The reserves (Table 1) are considerable, the coal quality high and the people amongst the best in the coal industry.

The transition from open cut to fully underground operations will be dependent on the margin achieved from open cut coal after mid 1999 and the volume of product we can get into the market place at an acceptable price.

With what has been achieved over the past 15 months, the signing of EBA's across the site, removal of industrial constraints and the rights of management to manage, Oaky Creek Coal can, and will be the best underground operation in the Australian industry.

COAL98 Conference Wollongong 18 - 20 February 1998
Table 1. - Coal Reserves

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<tr>
<td>SITE TOTAL</td>
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</table>

As with most things in industrial life, a determination, a sound strategy, an involvement of the workforce and the removal of artificial industrial barriers has all contributed to the successful turn-around of the Oaky Creek Coal business from one of the worst in the industry to one of the best. The success of this strategy is a result of the above factors and a determination by all now at the mine to secure a future for the mine and the Tieri community.

THE OAKY NORTH PROJECT

The Oaky North Project is an underground mining area located in the northern part of the Oaky Creek Coal lease. The area had been drilled previously and was the subject of further drilling in 1992 resulting in a feasibility study to construct a separate new underground longwall mine. Due to poor performance at the existing Oaky No.1 mine and poor rates of return from the Oaky Creek Coal business the proposal was not supported by MIM.

A further drilling program was initiated in 1995 to define structural discontinuities and reserves to the west of the mine plan proposed in the 1992 feasibility study. These reserves while large in size were previously considered too geologically complex and unsuitable for longwall mining. Following a short drilling program a new mine plan was proposed and
support of the MIM Board was gained to initiate a small scale development operation from an existing open cut highwall thereby accessing the main longwall reserves. Fig. 7 shows Oaky North Mine plan with geology.

The development operation was used to win further support for a future longwall operation by proving the viability of the mine and eliminating the major risks perceived by the Board. These were mainly three fold, geology, industrial

relations and the ability to achieve or exceed industry best performance in development. The operation was commenced in October 1995 with a single continuous miner development operation manned 24 hours per day five days per week. A second continuous miner unit was brought on line in January 1997. In December 1997 a third unit was added, to be manned on a part time basis.

A full blown feasibility study was undertaken in April 1996 being internally driven with consultants used selectively to provide input on specific areas which were identified to be their strengths. This study has been used as the baseline for Board approvals and as a measure of our ability to achieve results both in terms of advance rate and capital expenditure. The original proposal of April 1996 has been further enhanced by successive optimisation efforts which are based on doing things smarter, not simply cheaper.

The project was given full approval on 1 December 1997 immediately following the signing of a three year Enterprise Bargaining Agreement. During the process of negotiating the EBA not one single dispute occurred over the eight month period of the discussions. In fact since the mine commenced operation only 24 hours has been lost due to a site issue.

The mine is characterised by a number of innovative approaches to underground mine project development, all of which have proved to be overwhelmingly successful and will ensure its place amongst the world's best.
Human resources

Every single employee at Oaky North was only employed following successful advancement through a rigorous targeted selection process. Each person irrespective of their position was hired on their merit and what they had to offer the organisation. Employees who were successful in achieving employment then received significant amounts of training to ensure they had the necessary skills to ensure the mine's success. A person's ability to receive training efficiently was a principle part of their selection. Wages employees were selected from a mix of experienced and inexperienced sources. The ratio of inexperienced to experienced is considered high by industry standards. This involved commitment to considerably high levels of training but was more than worth it to avoid a mine of industry standard.

Employees are given the trust and respect they deserve and are expected to pro-actively contribute to the improvement of mine performance. They are expected to carry high levels of responsibility and be held accountable. This is achieved through the use of a very flat management structure to ensure decisions are not lost in a waffle of management beaurocracy but retained as much as possible at the shop floor. To date Oaky North has eliminated the entire shift management level and replaced it with a strong team focused group of front line supervisors. This has created a high degree of job satisfaction amongst the work force and reduced the size of the management structure.

Development operations

Development is carried out by conventional continuous miner and shuttle car methods. A development panel incorporates a Joy 12CM12 ‘C’ 5.2m single pass miner. This unique machine incorporates two pivot points to allow interchanging from thick to thin seam configurations. It is equipped with four on board ARO 4000 series rigs and two rib bolters. The machines include a material supply system from Roberts Engineering. Two Joy 15 SC 32 50/50 cars are provided with wide low ground pressure tyres to cope with a soft floor condition. Stammler 14BF breaker feeders are utilised to discharge the coal onto the 1050mm conveyors.

Roof support consists of a staggered 4/6 bolt pattern at 1.0m centres using 2.1m high strength bolts fully encapsulated with a two speed chemical. 1.2m rib bolts are installed on both sides of the roadways at 2.0m centres. 6.1m point anchored flexi-bolts are installed during primary development driveage in areas of high horizontal stress and weak roof conditions. The roof has a tendency to suffer a buckling failure, the flexi-bolts act to support the dead weight of the de-stressed failed roof beam. A continual geotechnical monitoring program is employed throughout the mine.

All development at Oaky North is on 1050mm belt. This allows the belt moves to be quick and efficient and allows development inventory to run ahead of the longwall without tying up large amounts of expensive longwall structure. Ventilation is by 17.5m³/s fans from ABB using 610mm fibreglass tubes. Each panel is powered via a 1MVA IP55 substation feeding via a 150mm² cable to a six outlet FLP gate end box.

Development crew size is dependant on the resources available and the total work load scheduled for the shift, a panel will not stop cutting due to an artificial manning barrier. The basic concept of our development success is to keep the systems simple and repetitive to foster confidence and improvements. Last financial year a total of 12,296m were developed with one miner for a full year and a second machine for five months. Figs. 8 and 9 show the performance for 96/97 and 97/98 financial years, respectively. So far this financial year we have developed 9,868m for the first six months from two machines.
Fig. 8 – 96/97 financial year performance

Fig. 9 – 97/98 financial year performance
Longwall operations

A longwall will be installed in January 1999 at which time about three and a half blocks of inventory will exist. The faces are 260m centre to centre with the extraction height of the first eight panels over 4.5m. The longwall will be the highest capacity commercially available utilising state of the art automation to minimise the need for operators on the face. Due to the nature of our coal and the dryness of the seam, dust will be a problem.

The conveyor system to handle the longwall coal will include 1600mm gatebelts rated at 4,500tph continuously, running at 4.4 m/s. These will discharge onto a 2000mm trunk belt rated at 6,500 tph continuously, running at 4.1m/s. The first trunk belt will be a 2.7km long single flight conveyor servicing the first six panels with the single drivehead located in the fresh air at the highwall portal. The coal will be taken by a similar belt up an open cut ramp and discharged onto a 150,000t single cone ROM stockpile. The coal is then reclaimed and transported to the Coal Preparation Plant by overland conveyor. Fig. 10 shows Coal Clearance route

A production rate of over 4.0mtpa is being targeted with a high degree of confidence that this figure can be exceeded within one year of the longwall starting production.

Construction

The project construction is managed by the operations team with a minimal use of consultants to ensure ownership and a finished product which matches the company's requirements. A conscious decision was made to steer away from EPCM style construction based on the poor track records which prevail in our industry. The East Site Facilities Plan is shown in Fig. 11.

Significant use is made of MIM's buying power and internal commercial expertise. However, the main prerequisite to achieve this was the ability of the persons employed to operate the mine. All have enough industry experience to know how they don't want things to turn out at the end of the day and are prepared to make decisions for which they will be held accountable. Unfortunately too many people in the coal industry have been prepared to relinquish this responsibility to a third party with little or no operational experience in lieu of being held responsible for a possible mistake. This soft management approach will never achieve the competitive edge required to survive.
Fig. 10 – Coal clearance route

Fig. 11 – East site facilities plan
Table 2 - Average production mine days worked (tonnes) - individual Longwalls

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Table 3. Average production mine days worked (tonnes) - Longwall mine total

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<th>Tonnes</th>
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Source: New South Wales Joint Coal Board Australian Longwall Mining Statistics.

3 Australian Longwall Mining Operations, Period 29th December 1996 to 27th September 1997
### Table 4 - Tonnes per man

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### Table 5 - Total Longwall mine production

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Source: New South Wales Joint Coal Board Australian Longwall Mining Statistics.


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Use of Contractors for Mining Operations

P Allonby

ABSTRACT

Contract mining is an integral part of the underground and surface hard rock mining industry in all states of Australia. Contract mining is now making a major contribution to the opencut coal industry in Queensland and New South Wales with Thiess alone producing over 16 million tonnes per annum with safety performances that most coal companies would be proud of. Contract mining is now attracting interest in the underground coal sector.

INTRODUCTION

The use of contractors has been common in the underground coal industry for quite some time but not for coal production. Contractors have carried specialist jobs such as drift and shaft work, fault driveages, excavations and specialist strata control. Contractors have also been used for "dirty" jobs that the mine regarded as unpleasant or dangerous, such as recovering from falls or cleaning out sumps.

Contractors came with a reputation for flaunting safety and for being under-equipped so the mine had to lend roof bolters, pumps, hand tools, etc, that rarely got returned.

The need for mining companies to focus on their core business and the introduction of industrial reforms has increased the scope and use of contractors. Longwall moves, conveyor installations and maintenance are now the domain of the contractor. Competition amongst contracting companies is fierce and they must perform to clients' expectations to secure further work.

Attitudes to mine safety have changed – corporately, mine operators are required to treat contract employees as their own and pre-tender assessments of contractor safety performance and safety management procedures is common. Contracting companies take safety every bit as seriously as the mine companies.

A number of underground coal mining contracts are now in place in Queensland and New South Wales and there is no reason to suggest that the success achieved in surface operations will not be repeated below ground.

WHY CONTRACT MINING?

There is a lot more to being a coal mining company than simply mining coal. Commodity prices, selling volumes, the effect of exchange rates, share price and capital constraints are constant topics of discussion in coal industry circles. Aboriginal land claims and community and environmental issues relating to mining are regularly in the media.

1 Thiess Contractors Pty Limited
These are just samples of the issues facing mining companies that may serve to distract the coal company from mining coal.

The demands on the mine management are far greater than in the past. Safety, statutory processes, planning approvals, servicing customer and marketing requirements, industrial reforms and corporate bureaucracy all reduce the time that mine management have to focus on production and cost.

Other activities such as transport, exploration, long term planning and acquisitions also continue.

In Queensland town administration is another significant component of a mining companies activities.

It is tempting to ask, What is the core business of a coal company?

Many mining companies have attempted to counter these increased mine site activities by making structural changes such as introducing the role of General Mine Manager or by introducing team building and employee participation programmes.

These moves have proven largely unsuccessful because the focus, generally, still remains on so many areas we get lost in standardisation and processes and loose sight of results.

The use of the mining contractor enables the mining company to be successful in fewer areas whilst the contractor focuses on mining efficiently.

The use of a mining contractor can also serve to equip or re-equip mines with restricted capital budgets. Many hard rock mining companies are lease owners with the contractor supplying the mining plant to develop then operate the mine. Thiess replaced Collinsville’s mining plant when it commenced mining operations on behalf of MIM.

Companies such as Thiess who are civil, construction and mining contractors have the ability to design and construct the mine infrastructure as well as mine the coal.

TYPES OF MINING CONTRACT

Contract mining operations range from parts of mines such as Allied Contractors development at Oaky Creek No 1, through stand alone mining operations within a lease such as Alliance Colliery or Newlands Open Cut, to total mine operations such as Mount Owen, Collinsville, Burton and South Walker Creek.

Thiess contracts are generally “fixed price” whereby the mine owner buys the coal from the contractor at a guaranteed price per tonne or price per metre. Contracts may be of fixed duration or life-of-mine. Life-of-mine contracts have agreed performance criteria that must be achieved for contract continuance.

By contracting to a fixed price for the mining operation the mine owner has now removed one of the largest variables in the profit equation. Mine operating costs now become the responsibility of the contractor who must maintain productivity and cost control to make a profit.

THE SUCCESS OF A CONTRACT MINING OPERATION

Relationship

To be successful the client and the contractor must understand each other’s business and trust each other. Both parties exist to make profit and if either party fails the contract will fail.

Understand the resource and risks

Thiess will fully evaluate the information provided by the client to prepare a detailed and accurate mining proposal. The
evaluation will included:

- Mining plans
- Equipment selection
- Personnel requirements
- Skills requirements
- Management Organisation
- Assessment of risks

People

People are a key to any successful project. Thiess’ people management practices include:

- Selection
- Training
- Procedures
- Supervision
- Remuneration
- Teamwork & Involvement
- Communication

Projects are run as accountable businesses with small management teams. This maintains a clear focus on the outcomes of their business. Communication is direct and employees are aware that if the project outcomes are not achieved the client will not extend the contract. Employees are expected to provide input into improving project safety and performance through “tool box talks” and face-to-face communication. Management is expected to consider employee suggestions in their decision making but are accountable for their actions (or inactions).

Plant

Thiess have refined their plant management procedures with over 60 years of earthmoving plant ownership and operation.

Key factors in the Thiess plant system include:

- Equipment selection;
- Life-of-plant costing for ownership and maintenance;
- Project accountability for utilisation;
- Project accountability for operating and maintenance cost; and
- Replacement when due.
Equipment is purchased for an application and is in effect, hired to a project. The rate charged to the project is established by projecting operating and maintenance costs for the life of the equipment and including the cost of ownership (finance and depreciation). The project is therefore accountable for the true cost of operating the plant and it is in the project’s best interests to maximise plant utilisation and reduce surplus plant. Open cut mines are typically achieving plant utilisation of 22 operating hours per day.

The Thiess plant system appears ideally suited to underground coal mining however the industry generally does not have available lifecycle plant histories and has had a mentality of operating “grandfather’s axe”.

Safety

Before each project commences a Safety Management Plan is prepared for the project. The plan follows quality management principles and is based upon proven standard policies and procedures.

The plan is specific to each project and provides input into training program, supervisory procedures and site audits. Because the plan is in place prior to project commencement and induction training emphasises employee involvement in safety, safe work is regarded as the way that business is conducted.

When environmental responsibility is required under a contract a similar Environmental Management Plan is prepared for the project.

Cost control

The potential profit of a project is obviously divided by the mine owner and the contractor. The contractor takes on the mining risks within the contract price. To achieve a satisfactory profit margin the contractor must manage the risks just as any other miner must do. Badly managed risks may negate profit. Similarly costs must clearly be understood, tracked and controlled.

Supervisors are clearly accountable for their expenditure and all means of reducing project costs are explored.

The contract miner will subcontract work if it can be carried out more cost effectively by others. Thiess for example subcontracted some of the initial development of the Alliance Project to Allied. Product coal haulage at Burton is carried out by Brambles.

Continuous improvement is sought in all areas of performance to improve the profitability of a project. Without it the contractor will not retain contracts or win new business.

CONCLUSIONS

The coal industry in Australia is undergoing major reform. Contract coal mining is a major consideration in the development of new mines and the revitalisation of old mines. Contracting companies are enabling mine owners to intensify their focus on fewer business functions whilst the contractor focuses on mining. This recipe is proving effective in open cut coal mines. It will also do so underground.
Managing Mining Contracts

J Luxford

INTRODUCTION

Contractors have penetrated all areas of the Australian mining industry over the last 15 years. The process started in the WA gold and nickel mines and has since spread around the nation to iron ore, base metals and coal mines.

The mine contracting industry has progressed from tentative beginnings in the 80s to the current situation where it is now providing high standards of professional management, safety and workmanship with competent people and high quality plant and equipment.

People are the most important link in the mining contract management chain. This applies equally to both the principals and contractors in the mining industry. The key is competent people on all sides who can manage the projects well and build effective working relationships.

The steps to successfully managing mining contracts include:

1. Starting with an effective contract document;
2. Understanding how the contractor has priced the work;
3. Recruiting competent and experienced personnel;
4. Establishing thorough systems to document and record all aspects of the project;
5. Establishing systems to promptly and fairly deal with:
   • Progress payments,
   • Other monetary claims,
   • Extensions of time claims, and
   • Variations.
6. Establishing a complete mine development program and keeping it up to date; and
7. Both parties understanding the other’s business practices

THE RISE OF CONTRACTING

Mining contractors were practically non existent in the Australian mining industry until the 1950s. Most mining companies sank their own shafts and undertook any other capital development that they required. Major mines started to use contractors for shaft sinking projects from the 1950s onwards. Examples of this in the hard rock sector included Mt Isa, Mt

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1 Principal of Luxford Mine Management Services and Project Manager - Underground Mine Construction at Cannington for BHP Minerals

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Lyell, Leinster and Mt Charlotte. The unions in Broken Hill kept contractors out of all the major shaft sinks there. Allied Constructions were active in the coal industry through the same period sinking shafts in the Newcastle and Wollongong areas.

The WA gold boom transformed mining industry attitudes towards the use of contractors in the mining industry. Many of the small gold mining companies born in the 1980s operated with minimal capital. These companies preserved their capital base by using earth moving contractors to mine their open pits. Intense competition kept contract mining prices to a minimum with the contractors usually mining the ore bodies at less cost than the companies could have themselves. During the 1990s, many of the gold mines have reached the lower limit of their open pits and have commenced underground mining. The use of contractors has continued as these mines have gone underground. As a result, the majority of mines in WA are now using contractors to carry out their mining work.

Based on the success with contractors in the gold sector, mining companies introduced contractors into the newest generation of iron ore mines in the Pilbara region of WA. Some of these mines are producing up to 10 million tonnes per annum.

Other very large scale open pit operations are also using contractors now. Some of the most notable include the Super Pit at Kalgoorlie, Lihir Island in Papua New Guinea, Mt Keith, Boddington and Ernest Henry. WMC provides one of the best examples of the mining industry’s embrace of contractors. They led the way with the introduction of contractors to their new nickel mines in the 1980s. This had the effect of changing the prevailing work practices in these new mines at Kambalda and focussed mine site managements and their workers on productivity and costs.

Following the early successes at Kambalda, WMC used contractors exclusively at the Leinster Nickel Operations and Agnew Gold Operations when these were started in the late 1980s. In 1996, the wheel turned full circle when WMC introduced contractors to all their Kambalda and St Ives mines.

Prior to this change, the Kambalda nickel operations had suffered industrial relations problems as difficult as any in the coal industry. The introduction of contractors has given WMC far greater flexibility to manage their mines more productively.

The eastern Australian coal industry has also grasped the contracting nettle in the 1990’s. The use of contractors in both surface and underground mining is increasing. Thiess have underground mining contracts at Oakey Creek and Newlands and surface coal mining contracts at Collinsville, Burton Downs, South Walker Creek and Mt Owen. There are now several other surface mines using contract mining in the Hunter Valley and in Queensland. In most cases, the mining companies have achieved significant productivity gains and cost reductions.

Other underground coal mines in the Bowen Basin are using contractors to increase their flexibility. Oakey Creek has gone as far as to have contractors operating longwalls and development sections. They, and a number of other mines in the Bowen Basin are also using contractors to carry out a wide range of underground activities such as longwall moves, belt installations and miscellaneous construction activities.

In 1997 Anaconda Nickel introduced BOOT (Build - Own - Operate - Transfer) contracts to the mining industry. They are using BOOT contracts to provide fixed plant for the Murrin Murrin laterite nickel project in WA. Outside of the mining industry, these types of contracts are gaining in popularity with State governments around Australia for the provision of infrastructure.

**GENERAL OBSERVATIONS**

**Professionalism**

The fledgling hard rock mine contracting sector’s professionalism left a lot to be desired in the 1980s. The quality of many hard rock mining development projects was poor. The focus was entirely on speed and profit at the expense of...
workers health, safety and standards of workmanship. Following a strong industry push around Australia this decade, standards have improved dramatically. Australian contractors are now on a par with the best contractors around the world and stand up well in comparison to the better mining companies.

This is not a claim that could have been made 10 years ago. In particular:

- Health and safety of workers is systematically managed;
- Workers are well trained and competent;
- Plant and equipment are of a high standard; and
- Efficient systems for maintenance have been developed.

**Occupational health and safety**

The gradual improvement in standards of occupational health and safety in the contracting industry has mirrored the increasing professionalism of the contractors. The standards of training and safety management among the major contractors is on a par with the major mining companies. This is now reflected in the excellent safety records of the main contractors. Lost time injury frequency rates have come down from rates in the 100s ten years ago to in some cases less than 10. There are some underground mine sites operated by contractors in WA that have operated for several years without an LTI. At least one of the main contractors is adopting leading edge safety management systems such as the Positive Attitude Safety System (PASS) and the ISRS safety system.

**Variation in rates**

Rates appear to be remarkably similar across many of the major hard rock mining projects around the country. Tendered rates will vary significantly on any particular project. However, the winning rates across many projects are quite similar. This reflects the realities of the market place where most of the major contractors have similar cost structures. The variation in tendered rates for any particular project will reflect the varying judgements of risk and desire to win the work from each of bidders. In some cases, inexperienced contractors will underbid work due to omissions in their cost calculations or underestimation of the difficulty of the work. In the coal sector, rates will vary more as there is less experience with major mine development and operations contracting on hard money. Rates should stabilise as the contractors gain more experience in the coal industry.

**ESTABLISHING THE CONTRACT**

**Writing the contract**

One of the greatest causes of problems and disputation on projects is poor contract documentation. Typically, many contracts are ambiguous, have repetitious and contradictory clauses and incomplete or incorrect specifications. As a result, the contract is of little use to the people on site who are managing the work. About the only part they will refer to is the Schedule of Rates to determine the monthly progress claim. This is not a problem if nothing significant goes wrong. However, if a seriously expensive issue arises that is not clearly covered by the contract document, disputes and deteriorating relationships on site often result.

An effective contract provides an easily understood guide book to all the parties on the project. It will clearly define for both parties their powers, entitlements and duties, who does what, how the work is to be done and the standards required.
To achieve the above goals, an effective contract must be:

- Clear;
- Certain;
- Consistent (no ambiguities);
- Conclusive; and
- Complete;
- Capable of being enforced.

Superintendents and project managers must know and understand the contract if they are to represent and protect the interests of their respective employers. It is disturbing to hear a superintendent or project manager say “Oh I don’t worry about the contract. We threw it in the bottom draw at the start of the job. If we have any problems here, then we sort them out as we go along”. One may survive with this attitude so long as nothing goes seriously and expensively wrong. However, in the event that a dispute arises over significant amounts of money and either party hasn’t fulfilled its obligations under the Contract, then that party may well find itself in breach of contract and liable to the other party.

When difficulties arise on the job, both parties need to be able to refer to the contract for guidance in resolving the issues. This will usually come down to who pays. In these situations where the contract has to be interpreted, the fundamental question to be asked is “what does the contract say?”. If it is an effective contract, it will usually provide the guidance needed to resolve the issues.

Having said all of the above, situations will arise that not even the best contract will have anticipated. In these situations, there is no substitute for good will and fair dealing between the parties. The fundamental question to ask in this case is “what is fair?”. When tempted to be “hard nosed”, one should always remember the old adage that “what goes around comes around”.

Role of the lawyers

Contrary to popular opinion (of engineers), there is a role for lawyers in forming effective contracts. Too often, engineers with little or no legal or contractual training will modify standard form general conditions of contract with absolutely no appreciation of the consequences of their actions. In this case, the lawyers are not involved until the project has run into major difficulties and the parties are talking about serious claims.

Engineers should consult experienced construction lawyers for advice when they want to modify standard form general conditions of contract. The ideal lawyer in this case is one with broad construction litigation experience. They will have a sound appreciation of the pitfalls and problems that can be caused when modifying conditions of contract.

Using standard form conditions of contract

Most experienced practitioners in the construction fraternity will advise clients to stick as closely as they can to standard conditions of contract. The AS 2124 series of standard form general conditions of contract have traditionally in the Australian construction and mining industry. The advantages of such standard forms are that:

- All the clauses have been tested in court over the years;
- The document has been refined over the years through several editions to remove ambiguities, inconsistencies and contradictions;
- In the event of disputation, experienced contract professionals advising the parties all know and understand what the clauses mean; and
- They are fair to both parties.
Mine owners who have had unfortunate or unpleasant experiences with a contract gone wrong will often be tempted to heavily modify their next contract in their favour. Such temptations should be strongly resisted. The result more often than not is a clumsy document that is full of inconsistencies, ambiguities and contradictions. The author has had to administer some over these contracts at different times.

The standard form contracts provide an astute superintendent with all the power he needs to effectively manage a contract. Clauses 33.1 and 35.2 in AS 2124 provide the superintendent with all the powers he needs to deal with a contractor who is failing to perform. Those who heavily modify standard form contracts to improve their positions would do better to more effectively manage their contracts rather than to rely on onerous and unfair contracts.

Negotiating effective contracts

An effective contract is one that is fair to both parties and meets the criteria set out above. A contract will be seriously flawed from the start if it fails to meet these criteria.

In the author’s experience, five is the optimum number of contractors to invite to tender after exhaustive preselection evaluations have been conducted. The cost of tendering is so great that it is grossly unfair of mine owners to invite contractors to bid for work when they have no intention of awarding them the work. With five tenders, a wise bid evaluator will normally discard the highest and lowest bids and negotiate with the three tenderers in the middle price range.

The aim of the mine owner should be to select a contractor who has a proven track record in the type of work to be carried out, can bring competent and capable people, suitable equipment and a well resourced management team to the project and has fair prices for the work to be carried out.

Having narrowed the negotiating down to the preferred tenderer who meets the above criteria, the shrewd and wise mine owner then spends as long as is required in discussion with the contractor in order to completely understand how the contractor has priced the work, what is and isn’t included in the rates and how contractual differences will be resolved once the project is underway. It is important to reach an understanding up front as to how the thorny issue of claims will be dealt with. Many people put their heads in the sand and pretend that problems won’t happen on their project. In other words, its vital that the mine owner gains a thorough understanding of how the contractor does business. A “partnering” style of post award workshop can be very useful to help the parties and their staff understand what is driving the other side and to start to build the ever so important personal relationships that will either enhance or plague the project.

Mine owners who automatically chase the lowest bid without regard to all the other issues that may affect the project often finish up with a more expensive project than they would have had they followed the process outlined here.

PERSONNEL

Need for competence

The key to successful contract management is the presence of competent people on both the mine owner’s and the contractor’s teams. When competent people are present on a project, problems will nearly always be resolved; the work will be well planned by the mine owner and well executed by the contractor.

Projects may survive inadequacies on the mine owner’s side but not in the contractor’s team. A competent contractor can often compensate for deficiencies on the other side. Unfortunately, disaster will often strike if the contractor’s team does not know what its doing.

The first casualty when competence is lacking is trust and cooperation between the parties. This is because each party will be blaming the other for all the problems that will inevitably be starting to trouble the project.
Role of superintendent

The superintendent’s role is to manage the work under the contract. The responsibilities placed on the superintendent go to the heart of effective contract management. The role under most forms of construction contract is not only to be the mine owner’s agent, but also to be an independent certifier of the value of work done under the contract.

The mining environment is a particularly difficult one for superintendents to operate in. This is due to all the unknowns in underground mining. Superintendents have to call on all of their experience and knowledge of mining, contracts and people, usually at the same time, when something has gone seriously wrong underground. This is because the superintendent has to determine what went wrong, why it went wrong and who is going to pay to fix it.

Most mining contractors will acknowledge how important it is to have a superintendent who possesses the above experience. Major disputes have arisen when superintendents have not possessed this experience.

With respect to superintendents contract management experience, at least one of the major contractors active in the coal industry has related his extreme frustration to the author of having to deal with superintendents who had no contractual expertise and consequently were oblivious to the effects their decisions were having on the contractor under the contract. Contractors are placed in a very difficult position when an inexperienced superintendent is administering a contract poorly and adversely impacting on the contractor, i.e. costing them money. Most superintendents and mine owners get very agitated when contractors start talking about claims.

In fact the worst tag any contractor can get is be labelled “claims conscious”. Yet if they are losing money due to the superintendent’s poor administration of the contract, they are entitled to compensation. Most contractors will endure this situation until the pain gets too expensive to bear in the interests of maintaining the relationship with the mine owner and his superintendent.

Superintendents must also be aware of how dependent the contractor is on them, as the mine owner’s agent, for the timely supply of drawings, directions etc. required for the efficient planning and execution of the work. Delays in this area are a major source of frustration to most contractors from time to time.

Another trap for young (and not so young) superintendents is in becoming an unrealistic perfectionist. It seems that many superintendents quickly forget the realities and difficulties in getting things done underground. They may demand perfection from the contractor when this is not practical at the time. This can lead to a lot of tension and poor working relationships on the job. An experienced superintendent working with the contractor will achieve far higher standards in the long run.

Importance of relationships

Effective working relationships are the key to effectively managing contracts from both the mine owner’s and contractor’s perspective. These relationships are built on competence and trust at all levels, particularly at the top. It is a very powerful model for the technical and supervisory staff on a project when they can see their bosses committed to working cooperatively to get the job done.

Poor relationships inevitably are a factor in most contractual disputation. Sometimes they can be a result of more fundamental problems on the project. At other times, they may be the root cause of the problems.

Difficulties in retaining a stable workforce

The last few years have been characterised by shortages of experienced staff at all levels in all facets of the mining industry. As a result, the demand for good staff (and their remuneration) has risen significantly, leading to greatly increased turnover.
The result is, that despite the contractor's best intentions, the staff they had committed to a project at the time they were awarded the tender may have left for greener pastures prior to, or during, the job starting. Such turnover may have an adverse effect on productivity.

It is not uncommon for principals to include strongly worded clauses in their contracts to the effect that the contractor cannot change any of his people without permission. The reality is that contractor's staff will often move on regardless of what is written in the contract. One way for mine owners to minimise these risks is, where practical, to select a contractor who has a low turnover of key staff. This can easily be determined from the resumes submitted in the tender schedules.

Contractor's manning levels

The manning levels contractors apply to projects may not be consistent with the outcomes desired by the mine owner in areas such as training, safety, road maintenance or grouting post grouted rock bolts. The requirement for these things will normally be written into the contract, but the contractor may not have the personnel on site to properly carry them out. It is very important that mine owners ensure the contractor has committed sufficient personnel to the project to do all of the things that they want, and are paying for.

This issue often arises in the areas of safety, training, environmental management and quality assurance. Contractors have been known to leave this work up to their Project Manager, Foreman or Site Engineer. In today's commuting environment on isolated projects, these people often do not have the time to adequately discharge these other responsibilities. This may lead to major disputation later on in the project if the contractor is failing his duty of care obligations or cannot verify the quality of the work done.

RECORDS

Advantages of good records

Good records are an essential part of effective contract management. They enable the parties to keep track of what has been directed, agreed, disagreed, gone wrong and been done on the job. The discipline involved in correctly documenting these matters aids clear communication and minimises misunderstandings.

In the event of disputes or differences of opinion, good records will allow differences to be quickly resolved. Where the records are poor, disputes flourish because it becomes one person's word against another. This is particularly significant in the area of variations to the scope, sequence of work, methods, procedures and latent conditions.

There will be turnover of staff on both the mine owner's and contractor's staffs during the course of the project. Good records allow new people joining the project to see what has gone before. In the event of problems, they can refer to the records to see the background to the issue in question.

Form of records

Records may comprise the following forms:

- Diaries
- Correspondence
- Meeting minutes
- Agreements

- Reports
- Plans and designs
- Databases and spreadsheets
- Photographs and videos
Contractual records

Contractual records are important to ensure that all parties are meeting their obligations under the contract. They will include registers and files for:

- Document transmittals;
- Site memos;
- Site instructions;
- Variations;
- Extensions of time; and
- Requests for information.

- Progress payments;
- Dayworks;
- Drawing register;
- Progress reports; and
- Minutes of meetings.

Quality assurance records

Quality Assurance records provide verification that the contractor's work is in accordance with the specifications. This is vital to the contractor if the mine owner later disputes the quality of the work.

QA records will include:

- Work procedures;
- Non conformance reports;
- Inspection and test reports;

Safety and training records

Safety and training records are vital to verify that the contractor's obligations under "duty of care" are being met. They also help to promote compliance with the safety management program in force on the project.

The following records should be kept:

- Personnel resumes and approvals to start;
- Training and assessments;
- Incident and accident investigations;
- Toolbox meetings; and
- Audits and inspections.

MANAGING THE CONTRACT

Meetings

Regular meetings are very important in the efficient management of mining contracts. They provide an excellent method of coordinating activities, and documenting progress, problems, agreements and other issues.
On major mining projects, daily coordination meetings are important. This is especially so when there is a lot of traffic in the mine entry and there are potentially conflicting activities to be coordinated. Key staff from the superintendent’s team and all the contractors will normally be present at these meetings.

Weekly meetings involving all the contractors on the project are important on projects of all sizes. Minutes of these meetings should be written up and signed off by both parties within 24 hours. Provided they are circulated quickly, minutes provide an excellent record of the progress of discussions and decisions on a range of issues. In fact, minutes can be used to record a lot of matters that would otherwise require letters back and forwards between the superintendent and contractor.

**Progress payments**

Delays in paying progress claims can be a source of frustration to contractors. It is important that the superintendent has the resources to process the monthly claim in a timely fashion.

One trap that both parties should avoid is allowing discrepancies to creep into the progress claim between what the contractor has claimed and what the superintendent has approved for payment. Aggressive contractors have been known to push a welter of dubious claims in the hope of being paid for some of them in the final wash up. It is in the interests of both parties to put a stop to this nonsense as soon as it appears. By the same token, it is important that the superintendent settles valid claims quickly. The best policy when there is genuine good will on a project, coupled with an efficient contract document, is to settle all outstanding claims each month and not let them accumulate.

Unfortunately it often will not be possible to promptly settle claims when the contract clauses are open to more than one interpretation as to how the issue at hand should be valued. In this case, protracted negotiations are often required. This is especially so if prolongation costs could be an issue.

**Claims**

Claims are the dirty five letter word in the mining industry that most superintendents and mine owners dread. Fairness dictates that if the contractor has a valid claim under the contract, then the superintendent should pay it. Unfortunately this often does not happen, which then leads to distrust and deteriorating relationships.

The same superintendent who automatically rejects any contractor’s “claim”, irrespective of its merits, will often give the contractor a sympathetic hearing if the contractor asks “for help with a problem”. Contractor’s project managers would do well to follow this line. By the same token, superintendents must accept the contractor’s right to give due notice of a potential claim under most contracts to ensure that time bars in the contract will not rule out future discussion of valid claims.

In the event that the superintendent still does not help, then it becomes a commercial judgement as to the merits of pursuing the claim.

The decision to go to litigation is never taken lightly by contractors in the mining industry. Even when they are in the right, they often run the risk of being tagged as a “claims conscious” or even worse still, a “litigious” contractor. Contractors who have won such reputations have found it difficult to win work in the mining industry.

Litigation in the mining industry to resolve claims has not been common in the past. Although claims often arose during mine construction projects, they were usually settled by negotiation. It was very uncommon for the parties not to resolve their differences. This was due in part to the small size of the mining community and the fact that many of the key players on both sides of the contractual fence had been educated in the same mining schools and received their early professional training in the same mining districts.

However, given the increasing size of mining contracts and the amounts of money at stake, disputes are on the increase. There are several major disputes in the WA mining industry heading for litigation at the time of this conference. In each
case, the contractors have lost a lot of money on the projects involved and they are seeking to recover some of their losses through claims.

Effective management of mining contracts by mine owners, superintendents and contractors is the surest way to avoid disputes and litigation.

Significant claims can arise early in major projects. They have in two out of the three projects that the author has managed in the last 5 years. There is an enormously positive spin off if the issue is resolved to the mutual satisfaction of both the parties. This is that trust is established early in the project, with major benefits to the project in the longer term. The contractor sees that the mine owner will treat him fairly when he has a legitimate problem and the mine owner sees that the contractor is not trying to “rip him off”.

Variations

Poor management of variations is one of the greatest sources of problems on mining contracts. The types of variations likely to be encountered by superintendents include:

- Genuine mistake by the contractor;
- Latent conditions;
- Changing development or production programs requiring new levels of resources;
- Changing scope of work as a result of changing mine designs; and
- Other changes directed by the superintendent causing the contractor to incur extra cost.

It is not uncommon for all of these types of variations to occur on one project. In order to manage them properly, there is no substitute for the superintendent having a thorough knowledge of the contract and how to value the claims that can arise under it.

Thorough documentation and complete records are also vitally important if variations are to be well managed. They will ensure that misunderstandings are kept to a minimum and that all parties know what has and has not been agreed.

As a general rule, variations should be kept to a minimum. In the mining environment, this is often not possible. Changing ground conditions will dictate changes to mining plans and sequences. Thorough mine planning and scheduling by the mine owner will minimise the need for variations.

Careful structuring of the scope of work in the contract documents is vital to minimise variations. Contractors will quite happily tolerate all sorts of changes provided that their ability to work efficiently, cost effectively and profitably is not compromised.

Mine owners must be extremely careful how they handle variations in situations where their contractors are losing money. In this case, an unscrupulous contractor may be able to exploit loopholes in the contract to claw back some of the money they have lost. This is one of the reasons why it is so important to select a contractor who has sensibly priced the work, rather than the low bidder who has gone in expecting to make his money on the inevitable variations that arise in mining contracts.

Extensions of time

Extensions of time are a potential mine field involving Liquidated Damages (LDs), prolongation and acceleration costs. When the contract contains LDs, contractors must claim an extension of time when entitled to, in order to protect their position under the Contract. Otherwise the contractor may not finish the work by the date for practical completion. In this case the mine owner may deduct LDs from the contractor’s payments.
It is rare to see LDs actually deducted in mining contracts. Usually there is enough fault on both sides to deter the mine owner from deducting them. Mine owners should be aware that where LDs are put into a document, the contractor will have made an allowance in the price to pay at least a portion of them. In other words, the mine owner is increasing the cost of the work by including LDs in the contract.

The other drawback of LDs in a contract is that they usually lead to overly conservative mine development schedules from the contractor. It is much easier to get a realistic schedule from a contractor if LDs are not an issue.

Many Superintendents are unaware of their powers under Clauses 33.1 and 35.2 of AS2124 to order the contractor to accelerate if he is behind schedule. Used judiciously, these powers are far more effective than LDs in getting the job finished on time. On the other hand, if the superintendent uses these powers unwisely, then the mine owner may be exposed to a substantial claim.

The real problem for superintendents with extension of time claims is the spectre of prolongation costs lurking in the background. If the contractor is delayed and is granted an extension of time, then there may be an entitlement to payment for the extra time that the contractor will be on the job due to the delay for which an extension of time has been granted. An efficient contract document will contain delay clauses that clearly identify these costs, how they will be paid for and under what circumstances.

A superintendent may spend months resolving major delay claims when the contract document fails to address these issues.

Mine construction program

Managing the mine construction program is one of the superintendent’s most important tasks. Most of the problems that arise on mine construction projects stem from the work falling behind program. One of the most important things a superintendent can do is to start the project with a thorough and realistic program that includes all the important activities.

It is not uncommon for the program in the feasibility study to be lifted straight into the tender documents for the mine development contract. Often these feasibility study programs are too general to be of use as a project and contract management tool. The problems for the superintendent in managing the contract are compounded when the contractor accepts or does not question the mine owner’s unrealistic mine development program.

With the advent of powerful spreadsheet programs and A0 plotters, it is now possible to set up massive development programs with columns for each week of the schedule and a row for each development face. These schedules may contain hundreds of rows. The best way of establishing these programs is to provide a blank copy to the tenderers when the job is being bid. They can then fill in their estimates of advance of each face on a weekly basis. Doing this ensures that the bidders do not miss anything. During final negotiations with the preferred bidder, the weekly advance rates in the mine construction program can be reviewed and amended as required to ensure the contractor’s commitment to the program.

As the project proceeds, the actual advances can be entered into the spreadsheet each week against each face. The whole program is then updated at the end of each month allowing any problems to be identified and addressed. The need to regularly update the program and re-forecast the program is vitally important.

INFRASTRUCTURE STRATEGIES

The provision of infrastructure is an issue for the mine owner rather than the superintendent. The contractor normally provides the mobile plant and people for a mine development contract. With regard to infrastructure, either party may provide:
• Power generation;  • Workshop;
• Power reticulation;  • Bulk fuels storage;
• Pump stations;  • Offices; and
• Main vent fans;  • Camp;
• Magazines

The advantage to the mine owner of the contractor providing all this infrastructure is that it minimises the capital commitment by the mine owner to the project. This can be a significant advantage to some mine owners. However, they must remember that the more the contractor provides, the greater the disruption to the project if the contractor is changed.

TRAPS FOR THE UNWARY

The items in this section list some of the problems that principals, superintendents and contractors can create for themselves during a mine construction project.

Unrealistic program

Both the mine owner’s feasibility construction program and the contractor’s tender program have been known to be quite unrealistic.

When the program is too ambitious, the inevitable delays as the project progresses will cause significant stress to all parties, particularly when the process plant is ready to go and the mine is not yet producing ore or coal.

The opposite problem can also occur when the program is too conservative. This can happen when the contract contains LDs and the mine development program in the contract is very conservative to protect the contractor. The problem then arises where the site people for both parties measure themselves against the conservative program in the contract rather than a realistic one.

Failure of contract to address delays

In the event of major delays, there should be a delay clause in the contract that pre-agrees payment to the contractor for all the fixed costs, direct plant and equipment costs and material costs. Such a clause simplifies the resolution of major delay claims. On the other hand, the absence of such a clause may lead to disputation in the resolution of major delay claims. At the very least, it may take the superintendent and contractor months to resolve how to pay for the delay.

Contradictions and ambiguities in the contract

Poor contract documents are the bane of superintendent’s and contractor’s lives on a major mining contract. If contradictions and ambiguities are present in the contract they may lead to serious disagreement as to the interpretation of the contract. Even with goodwill, a lot of time will be wasted in resolving how to fairly interpret a poorly written contract.

Failure of the parties to control their site representatives

Two not uncommon scenarios are:

• An aggressive project manager, out to make his name, causes unnecessary conflict with the superintendent and is unproductive and uncooperative; or
• Alternatively, the dictatorial superintendent gives instructions that cause the contractor to incur extra cost which leads to claims and disputes.

In either case there will be a break down of site relationships to the detriment of both parties and the project itself. It is therefore important for senior management on both sides to be aware of potential personality conflicts on the project and to control their representative where necessary.

Not resolving claims

Failure to resolve contractor’s claims in a timely manner is one sure way for superintendents to create very real problems for themselves. It is not uncommon for superintendents ignore claims in the hope that they will go away. All that happens in this case is that there is a large backlog of claims to be resolved at the end of the project. Not only that, but key relationships and trust will have been steadily deteriorating as the claims are left unresolved.

It is in the best interest of all parties to ensure that claims are resolved promptly and not left to fester.

Lack of records

Good records are vital to all parties on a mine development project. Particularly when something has gone expensively wrong. Without complete records, it is difficult for a contractor to convince the superintendent that there is a valid case for extra compensation for the problems on the project. By the same token, it is difficult for a superintendent to properly assess a contractor’s claim if there are incomplete or poor records of the events.

Many disputes arise over what has or has not been directed by the superintendent at various times. Complete records of all directions will avoid such disputes and the damaged relationships that result.

With contractual correspondence on site such as memorandums, instructions and document transmittals, the author strongly recommends to all parties that the recipient countersigns the document to acknowledge receipt of the document. This can avoid a lot of problems down the track if documents are misplaced.

No financial incentive in contract to complete certain work

Some types of work are built into the contract rates and incur no financial penalty if they are not completed by a particular time. Classic cases of work that may contain no incentive for completion are grouting post grouted rock bolts (HGB’s) and paving decline roadways in hard rock mines.

Incentives for completion of these tasks can be built into the contract rates. The most effective incentive is to withhold payment until the work in question is completed. This will not cause problems provided that the rules are clearly spelt out in the contract before the work starts.

Without these incentives, it will be a constant battle between the superintendent and contractor to get these jobs completed. This battle can be intensified when the contractor is working on very low margins and is struggling to make a profit. A not uncommon scenario here is for the parties to argue for months about the unfinished work. In the end, the superintendent in frustration, withholds significant amounts of money from the progress payment in an attempt to force the contractor to complete the work. As a result the relationships on the site deteriorate even further.

Contractor’s manning levels

Hard nosed mine owners must remember that the only way most contractors can reduce their costs is to reduce the resources committed to the project. In particular, the pressure will usually be on contractors to reduce their staff on the project. This can impact on the contractor’s ability to achieve the levels of safety and quality demanded by the mine owner.
A popular cost cutting measure in the past was to have the shift supervisor working as a dedicated operator on the crew, usually as the jumbo operator in hard rock situations. Depending on the complexity of the operation, the quality of the supervision and safety on the job can suffer when the supervisors do not have time to ensure that their “duty of care” responsibilities are being properly addressed.

Contractor losing money

Superintendents beware! A contractor losing money on his job will usually lead to all sorts of problems. Contractor’s head offices quickly forget that it was their aggressive bidding for the contract in the first place that probably led inevitably to the losses on the job. Instead, they will put their site managements under immense pressure to increase revenue, which can lead to “corner cutting” or innovative claims.

CONCLUSIONS

The success of the contracting systems can be summarized as a set of recommendations as follows:

To Mine owners:

- Start with an effective contract;
- Carefully select the contractor;
- Select the superintendent with just as much care; and
- Ensure the superintendent establishes and maintains a good working relationship with the contractor.

To contractors:

- Ensure a good working relationship is established and maintained with the superintendent and mine owner; and
- Resource the job to meet all the obligations under the contract.

To superintendents:

- Manage the contract fairly and consistently;
- Be wary of perfectionism; and
- The standard you set is the standard you get.

To all Parties:

- Recruit competent and experienced people;
- Train them in contract management; and
- Consciously develop relationships and trust.
BIBLIOGRAPHY


Improving Relationships

R C Mellows

ABSTRACT

People in the work environment (and often elsewhere) are seen as a commodity (we use them for our purposes).

The general accepted view of a Company is that shareholders are “The Company” and that they employ people to utilise their money in a particular field to generate more money to give a return on investment. To achieve this, capital equipment is purchased to fulfil a task and people are employed to use this equipment. Equipment and people are commodities purchased with money.

The focus of this approach is MONEY. The equipment and people, and the use of them, determines the amount of money that can be generated. In this approach people are motivated by financial concerns. The harder the times the more money becomes the focus.

In more recent times there has been a greater recognition of the role of people, and Human Relations has become an “In” phrase. Recognition of the cost a person and the degree of value of various individuals has prompted action to adjust the approach to people to get the best out of them. This has become a management technique and has led to a great emphasis on in employee selection, job descriptions, training, industrial relations etc. This approach is reflected in our current push for Work Teams. However, this view is still badly flawed as it still looks at people as a commodity that must be managed correctly to gain maximum benefit.

At Cornwall, a different approach is taken that on the surface may not seem so different, but it has a vastly different motivation and outcome.

CORNWALL COAL APPROACH

Management attitude

The foundation and focus of our operation is people. The basic philosophy is that “The Company” is the people who work together to fulfil a task. To do that work they need money. This is supplied by shareholders, who are entitled to a return on their investment.

People are important as individuals. They have feelings, needs and concerns. All of these express themselves in the workplace. Dealing with these is an essential part of living in any form of community, and our workplace is a community. The work we do is a means of addressing these issues and to do so requires helping one another. Of course we must give an adequate return on investment to fulfil our shareholders needs.

Management technique

There are two opposing approaches to management. The first is the historical approach as already outlined. This is authoritarian and legalistic in nature and aims to control behaviour.

Under this approach a hierarchy of control is necessary with detailed instructions and regulations. A system of recording, reporting, written authorisations with an emphasis on disciplinary procedures should any failure occur in fulfilling the requirements. Taken too far, the effect is destruction of openness, relationships, innovation, flexibility and morale, and the development of mistrust. People hide their weaknesses and failures and learn to manipulate, to use the system, and to blame others. This is a destructive environment and leads to failure.

1 Cornwall Coal Company N. L
Cornwall has taken an alternative approach which recognises the inherent value of people. Our aim is to build up and develop people and give them greater freedom in doing their work.

Under this approach we recognise that people are restricted in reaching their full potential as people. They generally carry rejection, emotional pain and many emotional burdens. They are happiest when operating at their full potential and living and working in a friendly environment.

In our endeavours to achieve the right environment we must be aware that just as there are laws dealing with our physical environment, eg The Law of Gravity, so there are laws that deal with the well being of people and the relationship between people. Unless those laws are recognised and complied with the outcome will be people who are emotionally disturbed and who react to many situations in an undesirable manner. Both individual and group achievements will remain well below their potential and damaging conflicts will continue.

The law I am referring to is the Law of Love. It is not because of legalism that Jesus Christ told us to love God and love one another. It was because he knew it was essential to our well being in all aspects of life.

We are dealing with people who have been hurt, abused, rejected and emotionally bruised. Many have been used, taken for granted. They cannot trust. All people to one degree or another utilise an instinct for self preservation. A surprising number of people think little of or despise themselves. All of these emotions reveal themselves in anger, bitterness, jealousy, resentfulness, envy etc and have serious consequences in both Safety and Industrial Relations as well as productivity.

Love is both a feeling and an attitude that expresses itself in practical ways. Some of these ways which are essential to a healthy working environment and which we have tried to cultivate are listed below

- Accept people as they are, warts and all. Do not rubbish people whether openly or behind their back
- Always treat people with respect. Be humble
- Decide to trust
- Forgive – we all need it
- Be open – admit your mistakes and failures
- Build up confidence and self esteem
- Give responsibility, authority, support
- Create a non-threatening environment, give freedom to make mistakes, remove fear of failure and the need for self defensiveness. Give freedom to make foolish suggestions, allow room for people to grow and mature
- A person’s family and family needs are important. Do everything possible to accommodate them. Be interested in their family
- Treat injured people the same, whether compensation or not, wherever possible. Their needs are the same
- Try to place people where they are suited and want to be, if possible

We want people to enjoy work, to have self esteem, to have self confidence without pride, and to have concern and care for themselves and others.

To do all or even part of this is not easy. Our self interest, our weakness, our needs get in the way. Our reactions are often automatic and wrong. Our self preservation instincts are strong. Outside pressures and influences often dictate what can be done.
However by treating people correctly and by working on a persons needs we see growth, confidence, commitment, responsibility, openness and strong relationships. People are free to admit weaknesses and failures and they can work together to overcome them, supporting one another in the process.

This requires the Manager to adopt a leadership role. Their role is to give support and to act as a facilitator. Their emphasis is on developing people as individuals and their relationships with others, because they care about them as people rather than as commodities.

**MANAGEMENT STRUCTURE**

Under this structure our Managers work as a team but have full freedom as individuals to manage their areas of responsibility. Because of close liaison with other Managers, input is given by the others and a consensus of any overlapping issues is easily reached.

Managers have close contact with all their employees with only one intermediate leader operating within each group. This leader does not limit close communication with all individuals. Most decisions are made knowing the opinions of all concerned and are usually effectively, consensus decisions. This does not prevent unpopular decisions being made but because of the relationships that exist, they are generally accepted.

People at all levels can react quickly and confidently in any given situation because of their understanding of others and a confidence in their relationship. The management structure is set out in Fig. 1

<table>
<thead>
<tr>
<th>Accident Frequency Rates</th>
</tr>
</thead>
<tbody>
<tr>
<td>RATES</td>
</tr>
</tbody>
</table>

Fig. 1 - LTI frequency rates 1979 – 1997

**Employee training**

Little training existed before 1991. Training was generally by on the job coaching.

Two areas of training occurred in 1991/92 which together changed relationships

1. Team training which involved managers, staff and crew members in small groups. This was introduced following recognition that despite the best intent of all parties, there was obviously a lack of trust and understanding that needed to be dealt with. As the operation had gone through a period of growth over the previous decade, communication has suffered and with it relationships were becoming strained.

2. Work model training at level “0” for all employees. This again involved small groups comprising a cross section of all employees and was presented by the Managers. This did require policies to be developed on paper, a situation that did not pre-exist. The training was introduced as a result of the development of the Underground Work Model and the realisation that our people had no formal training and a lot of their understanding was based on outdated theories and beliefs. It was decided to start from basics for all employees.
The close liaison that occurred during this training had a major impact on understanding and appreciating one another and in breaking down barriers.

Further training continued in the areas of:

- **Leadership Workshops.** First line supervisors with the Manager and Engineer met under an external facilitator to further understand and appreciate each other and create a team environment at that level.

- **Technical Work Model Training.** This is given by the Managers whenever possible to further develop closer relationships with outside support only when necessary.

- **Deputy Training.** Out of 28 mining employees, 6 are qualified as Deputies and a further 8 are in training. Previously there was an unwillingness to take on the responsibility because of the small community and social separation caused by promotion. There was also a reluctance to take on what was seen as an onerous responsibility. Because the relationships between Deputies and their crews is now based on leadership, promotion does not create social separation. Because Deputies are supported in their jobs and encouraged, the responsibility does not appear as onerous. As a result people are keen to advance and gain the financial benefit.

**Employee involvement**

Encouragement is given to employee involvement in the following ways:

- **Consultation Committee** - Comprises all four Managers, one staff representative drawn from Administration, one Washery representative, two Underground representatives, the four representatives being elected by their workmates.

- **Team Leaders Meeting (at the Mine)** - Comprises the Manager, the Engineer in Charge, the electrical team leader, mechanical team leader, and the four permanent deputies. This meets fortnightly to discuss operational issues including planning, performance and costs.

- **Safety Committees** - Meet six weekly, one at the mine and one at the Washery. Again these comprise a good cross section of employees.

- **Crew Discussions** - These are all held whenever necessary to provide information to employees where notices are inadequate such as market and contract information, and also where discussion is recommended by the Consultation Committee to include all employees. They are addressed by the Manager and are limited to major issues to ensure clear and uniform communication. Daily contact with the sharing of information as appropriate occurs, limiting the need for formal meetings.

- **Others** - All employees are involved where appropriate to develop changes such as machine modifications and operational changes. This includes interstate visits.

**Enterprise agreement**

This has been operating since September 1994. The major changes to the Award and previous practices have been:

- **Introduction of the Work Model - Level four wages** were paid to all employees upon the introduction of the Enterprise Agreement on the achievement of 20% productivity increase. This occurred immediately. There was a commitment to follow up with training as quickly as practicable. Due to operating pressures training has been patchy.

- **Working Hours - Four day week** comprising 4 x 8½ shifts changing at the face, plus one hour accumulated to give 47 hrs/year nominally for training. Ten public holidays paid at 8½ hrs/day. Annual leave and sick leave is
accumulated as per the Award but paid at 8½ hrs/shift absent. Production crews work Monday to Thursday. Maintenance crew work Tuesday to Friday.

- Operating Hours
  
  Production 7.00am Monday to 7.30am Friday
  
  Maintenance 7.00am – 3.30pm Friday
  
  Overtime Production Sunday nightshift and Friday dayshift as required.
  
  Training Friday dayshift each crew 6 days/year.

- Sick Leave - Five days permitted to be taken as personal leave. Proportion of sick leave payout annually and limited by compensation claims.

- Compassionate Leave - Includes Grandparents and excludes Pressing Domestic Leave.

- Allowances - A fixed payment automatically paid on attended days.

The aim has been to simplify provisions to eliminate causes of conflict.

Recognition

Appreciation of effort is shown in the following ways:

- Safety - A dinner including partners occurs on the achievement of 12 months Lost Time Injury free (now 3 occasions) plus a nominal award is given.
  
  1. Production
     
  Those involved receive a Gift Voucher when a record is broken for either a shift, a day, or a 12 shift week.
  
  2. Individual Effort
     
  A point is made of recognising effort by verbal appreciation and thanks.

- Union Representatives - All union representatives at the mine are Level 6 Team Leaders. That is, Operational Leaders are also the natural leaders.

- Evidence of Success
  
  1. Safety – (Fig. 2)
     
  Last underground Lost Time Injury (38 employees), 12 July 1995

  Last surface Lost Time Injury (23 employees and 18 permanent contractors), 16 May 1992

  Our safety standards have been recognised by the Mines Department three times in the last four years as the Safest Underground Mine in Tasmania and this is reflected in our relationship with the Department.
2. Productivity – Underground

Employees 38

Budget Production 1,950 tonnes per day or 354,900 tonnes per annum

Actual Production 12 months to 31 October 1997 406,144 tonnes - equivalent to 10,688 t

3. Industrial Relations

Excellent. Although Cornwall has a long history of no Industrial Lost Time it has often been alleviated by considerable discussion with Union representatives and Union meetings. It is now rare to have formal meetings with Union representatives. Normal working contact deals with most issues.
4. Deputy Positions

Providing all persons currently training complete their course 50% of underground production employees will be qualified Deputies.

5. Rescue Team

Winners of the 1997 Tasmanian Mines Rescue Competition.

6. Profitable Operation

SUMMARY

What has been done should not be seen as another method or technique. It has not come about as a result of any plan but as a result of a gradual developing relationship that is occurring because of a specific attitude towards people. That attitude is reflected in every word and action and gradually builds mutual trust and respect.

Our industry operates in a difficult and dangerous environment. There is a need for standards to be set and complied with. There are many technical aspects that need to be understood and the knowledge applied in new and innovative ways. But the success of anything we attempt remains dependent on people and those who succeed will be those whose people are operating at their maximum potential. That potential can only be reached when we as leaders in this industry have the courage to love our people.

This requires personal risk taking, trust, hope and perseverance, a real commitment to the benefit of others before ourselves. It takes real courage. It's a big ask but with Gods help it can be done and we are rewarded by the release and growth we see in others, as well as success in our business.
IMPROVEMENT PROGRAMS THROUGH SYSTEMS MANAGEMENT

“Every morning in Africa, a gazelle wakes up. It knows it must run faster than the fastest lion or it will be killed. Every morning a lion wakes up. It knows it must outrun the slowest gazelle or it will starve to death. It doesn’t matter whether you’re a lion or a gazelle; when the sun comes up, you had better start running.”

old Ugandan Proverb (anon)

General experience with improvement programs in the mining industry has been that, after some initial success, performance tends to deteriorate back to pre-improvement program level in a relatively short time after the spotlight is switched off.

The “switching off” can take many forms, such as a logical end to the program, change of leadership personnel, change in business emphasis or a severe emotional event (e.g. retrenchment program, partial closure, industrial action).

The thing that appears to be missing from most improvement programs is sustainability. The lack of sustainability is costly as follow up programs tie up valuable resources over and over again. Often, the only benefits accrue to consultants, as it is relatively easy to prove a good return on the investment by showing impressive performance charts during the implementation of the program but not after completion.

The ideal improvement program shows significant early returns and continues through and beyond, i.e. it is sustainable.

There have been some notable improvement programs that have stood the test of time. Examples are Hamersley Iron and Comalco Aluminium in Tasmania. These companies put equal emphasis on the three elements that together ensure that work can be done effectively.

These three elements to ensure effective work are:

1. Technology;
2. Technical processes;
3. Social processes

In the mining industry, the fixation with technology and, to a lesser extent with the technical processes, have driven improvement programs to date.

Along with technology, these are the legs of a three-legged chair; each one necessary but not in itself sufficient to ensure the chair fulfils its function.

The systems management approach is to outline an improvement program that puts equal weight on these three elements and demonstrates the impact that each element has on the other.

1 Hawcroft Consulting Pty. Ltd. Musham, NSW
What makes people work hard? (Work in this context is defined as intellectual work i.e. the exercise of discretion and not physical work.)

It is not because they are made to and it is not because they are given financial incentives. Our own industry is a great example where this has not been successful.

It is the prime duty of a manager to provide the conditions that release the peoples’ full potential and creativeness into their work.

To clarify more fully the understanding of managerial work and to appreciate more fully the difference between the work of a manager and the work of an individual contributor, it is helpful to divide the work of a manager into 3 elements:

- The leadership of people;
- The scheduling of resources; and
- The use of technology

All 3 elements are intimately intertwined in the daily course of managerial work but can be conceptually separated. The work, which is unique to managers, is the leadership of people and this is addressed by:

- creation of environment of change;
- articulation of desired behavior of all employees;
- design and implementation of overarching people systems;
- training team members in leadership and team membership skills; and
- performing regular process audits

As part of the Systems Management approach the team members will have a better understanding of some core ideas and principles that have been successful in other organisations and they will become more aware of:

- The impact of leadership & team member behaviour on the outcome of tasks;
- How to understand & improve systems at work;
- The expectations of their role and work;
- The importance of clarity in assigning, receiving and recognising work & tasks; and
- How to sustain improvements in the workplace

Change philosophy

The business environment is never still. Changes occur in demand, technology and opportunity, which require changes in the way work is organised and carried out. This is necessary just to survive over time. The success or failure of the business is strongly influenced by people; the way they work, the way they are guided and their feelings about being part of a team.

Business is not just about understanding technology and finances, it is also about understanding people; how they feel about their work and what they are asked and want to do.
The Systems Management philosophy is predicated on our belief that people want to be constructive and creative and it is their environment and working relationships which influences how their energies are directed.

An organisation that treats its people like machines, constrained by petty rules and regulations that contradict common sense and a sense of fairness will fail.

Building trust is not a simple matter, what one person regards as fair or honest may not exactly correspond with another person's view. Understanding how people see the world and acting on this understanding is a key element of leadership. Understanding change in organisations requires a consideration of beliefs, a clear articulation of organisation theory and knowledge of what really is happening. Changing systems brings about changes in behaviour and hence the ways in which people work.

Systems changes which are sustainable are not simply technical matters that "can be knocked over in an afternoon". They require a manager to understand the present culture, what behaviour is desired and be able to predict the impact on team members.

Again, the Systems Management approach manages the change process through training the manager in the understanding of culture and leadership.

General propositions

The Systems Management approach is predicated on a number of general propositions about the business, the environment and the people who work in it. These propositions follow:

- The social and business environment is changing continuously and organisations in it need to change in order to survive. To prosper they must develop in every field of their activity and in particular they must continually improve management practices.

- People are social animals. We are dependent for our survival on the coherence and maintenance of social groups.

- People are not inherently resistant to change. They can and do accept and welcome change under circumstances in which the risks they perceive inherent in the change offer them an acceptable probability for improvement. It is understanding the circumstances from the perspective of those affected by the change that is important. Equally important is the need to recognise that for people to support change requires a commitment that is deeply felt.

- People behave rationally through their own eyes. They interpret systems, symbols and behaviour in terms of their beliefs about the world they live in and what they believe to be the underlying intent. Consequently a leader must know these beliefs and perceptions and the interpretations which will be derived from them.

- The environment we live in is chaotic. The longer into the future we can predict with accuracy, they safer we are. Accurate prediction is a significant advantage. The basic premise is that, in order to ensure that improvement is sustained, there needs to be an equal emphasis on both social and technical processes.

A case study

The behaviour of management in the coal industry, when confronted with severe downturns in profitability as is the present case, generally is described by one or more of the following:

- Cut cost, put head down and weather the storm;
- Acquire larger equipment and produce more;
- High grade the resource;
- Do nothing & hope the exchange rate stays down;
- Close the business; and
- Blame everyone and everything else for the situation.

Hence, the focus of survival “strategies” to date has been on external things (third parties, contractors, equipment, geological resource, etc) and not on internal things (people, systems, etc). With the drying up of capital some Companies have targeted technical process improvement alone as their saviour. These Companies generally win some initial improvement, which has not generally been sustainable.

In the world outside of coal, some Companies have put at least equal emphasis on social processes as technical process in their struggle to survive. A good example of this is the Aluminium smelting industry with Companies such as Alcoa and Comalco at the forefront.

In the early nineties, the key performance drivers at Comalco’s Bell Bay smelter showed the trends set out in Fig. 1:

Fig. 1 – Performance record at Bell Bay Smelter 1992 - 1995

All of the indicators in Fig. 1 displayed similar characteristics during the period:
- Poor, erratic performance;
- A step change in improvement; and
Stable performance at new level

The step changes in all cases occurred about the same time which was in mid-1994. The strategy and the implementation of the strategy for improvement started in mid-1990!

Between 1990 and 1994, all of the Company’s systems were overhauled using a systems design methodology that demanded that all existing processes and systems, both technical and social, be understood fully and what beliefs were held by all stakeholders about the existing systems be known before any redesign was contemplated.

The work at Bell Bay and the resultant significant improvement confirm that the changing people’s behaviour in a sustainable fashion takes time and cannot be achieved overnight. It also confirms that success has high rewards.

The big problem, of course, is that it is very rare to have patient and understanding shareholders as they were at Comalco. They usually demand a much tighter timeframe for improvement than a “couple of years”.

To placate the owners and to also to quieten the inevitable critics in the workforce who usually increase in population and voice with time when progress is slow, it is necessary to incorporate a program that ensures early success. This is where early improvement in a technical process has a very important role. It not only gives immediate return on investment but also keeps the critics at bay.

Time is needed to not only understand the organisation’s processes but to develop the competent systems to improve them. The right systems, along with the consistent behaviour of the organisation’s leaders, will change behaviour over time. At Bell Bay, this took around three years. The emphasis needs to be on the three legs of the chair, technology, technical processes and social processes, if improvement is to happen first and then be sustained.

SUMMARY

In summary, the three elements to ensure effective work are:

1. Technology;
2. Technical processes; and
3. Human processes

These elements are the “legs” of a three-legged chair; each one necessary but not in itself sufficient to ensure the chair fulfils its function. The Improvement Programs through Systems Management approach is to outline a program that puts equal weight on these three elements and demonstrates the impact that each element has on the other.

The Systems Management approach is about sustainable improvement brought about through managed change. It is about helping leaders understand and predict behaviour, and helping team members understand their role better.

The intended outcome of this Systems Management approach will be:

- Establishment of improved standardised systems;
- Ownership of the systems;
- Efficient use of the resources (human and technological);
- Improved productivity; and
- Greater participation and job satisfaction;
Introduction of Battery Powered Coal Haulers into Board and Pillar Panel Production

A Myors

ABSTRACT

Powercoal Pty Ltd operates eight underground coal mines in New South Wales and produces coal for both domestic power stations and the export market. Approximately 40% of total production is from continuous miners. Over the past five years, place change mining methods have been introduced to improve productivity from continuous miners. This has involved utilising the existing coal clearance system from the continuous miner to the ratio feeder using trailing cable shuttle cars.

By late 1996 average productivity at Cooranbong Colliery using this system had plateaued at 800 tonnes per eight hour shift. A number of factors were identified to increase average productivity to 1500 tonnes per eight hour shift. An improved coal clearance system was one of the factors identified.

Battery powered coal haulers were selected to improve the rate of coal clearance from the continuous miner to the ratio feeder. This paper details the decision making process in arriving at this selection. Issues discussed include alternate systems considered, advantages and disadvantages of battery haul cars, compatibility with other mining processes, tendering and supply arrangements, and implementation issues including mining and equipment approvals.

The battery powered coal haulers commenced operation in November 1997. The paper details limited operating experience since that time.

INTRODUCTION

Background

Powercoal Pty Ltd operates eight underground coal mines in New South Wales. The company produces approximately eleven million tonnes of coal per annum for both the domestic thermal and export markets. Approximately 40% of Powercoal’s production is from continuous miners, either from first workings or secondary extraction. Much of this coal is won from the Wallarah, Great Northern and Fassifern Seams of the Newcastle Coal Measures where either local geology and/or subsidence constraints preclude the use of longwall methods. Prior to place change mining (PCM), typical “whole of panel” productivity averaged 350 tonnes per seven hour shift or 7.3 tonnes per face man hour. (Note: This measure is calculated using hours at the face and face manning numbers employed in the process.)

In 1989, faced with increasing competition in the domestic thermal market, the PCM system was identified as a means to deliver competitive productivity within the geological and mining constraints commonly encountered at Powercoal’s Lake Macquarie mines. The system was successfully introduced in 1992 at Myuna Colliery in the Fassifern Seam. Productivity increased to 800 tonnes per seven hour shift (12.1 tonnes per face man hour) with peaks of up to 1500 tonnes per seven hour shift (22.7 tonnes per face man hour). The system relied on a coal clearance system of three Joy 15SC shuttle cars of 9.5 tonnes capacity each.

The system was introduced at Cooranbong Colliery in 1994 and modified to suit local geology and mining constraints. Coal clearance typically utilised two Joy 15SC shuttle cars. As improvements to the system were made, average productivity in first workings increased during 1995/96 but has plateaued at 800 tonnes per eight hour shift (12.7 tonnes per face man hour). Peak performances of up to 2000 tonnes per eight hour shift (28.6 tonnes per face man hour) have been achieved.

Benchmarking with United States mines identified scope for considerable improvement. Average productivity from PCM operations in the United States is 20 tonnes per face man hour with peaks of up to 35. One factor for this difference was
the rate of coal clearance from the continuous miner to the ratio feeder. Battery powered coal haulers (BPCHs) were highlighted as providing improved coal clearance rates in PCM units. The introduction of BPCHs combined with a review of pillar dimensions and the introduction of a roadway improvement programme will aim to improve coal clearance rates in PCM units. Target average productivity is 1500 tonnes per eight hour shift (20.0 tonnes per face man hour).

This paper details the selection and introduction of BPCHs at Cooranbong Colliery.

Cooranbong colliery

Cooranbong Colliery is located approximately 50 kilometers south of Newcastle on the south western edge of Lake Macquarie. The mine produces 1.7 million tonnes per annum from the Great Northern Seam at depths ranging from 60 to 140 m. Seam height varies from 2.4 to 3.0 m and the coal is extremely hard by national and international standards. The roof is a hard, undulating conglomerate with support requirements ranging from pattern bolting (1.8 m bolts, 2 m spacing) to more intensive bolting with W- straps (1.2m spacing) and mesh.

The floor (Awaba Tuff) ranges from claystone to siltstone and 300 mm of coal is left as a wheeling base. The seam is naturally damp and bogholes are commonplace. Regional seam dip is less than 1 in 40. Local seam rolls can be up to 1 in 8.

Five heading panels with pillar dimensions of 45.5 x 45.5 m centres are advanced using place changing techniques and are subsequently extracted using Voest Alpine Breaker Line Supports. Prior to battery haul cars the place changing unit consisted of:

1 x Joy 12CM12 Continuous Miner
2 or 3 Joy 15SC Shuttle Cars
1 x Mobile Roof Bolter
1 x 913 Eimco for roof scaling/floor cleanup
1 x Stamler Ratio Feeder

Manning consisted of one team leader, one engineering technician and seven production employees.

Implementation process

The implementation process for the BPCHs is summarised in Table

<table>
<thead>
<tr>
<th>Process</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>1*</td>
<td>Identify productivity improvement factors.</td>
</tr>
<tr>
<td>2*</td>
<td>Identify alternatives and preliminary evaluation - advantages/disadvantages.</td>
</tr>
<tr>
<td>3*</td>
<td>USA Benchmarking Tour</td>
</tr>
<tr>
<td>4*</td>
<td>Invite tenders for the supply of a Coal Clearance System based on BPCHs.</td>
</tr>
</tbody>
</table>

COAL98 Conference Wollongong 18 - 20 February 1998
<table>
<thead>
<tr>
<th>Step</th>
<th>Description</th>
<th>Evaluation Criteria</th>
</tr>
</thead>
<tbody>
<tr>
<td>6</td>
<td>Review of Production Process</td>
<td>Panel layout and design, wheeling routes, battery charge/change station, manning, infrastructure.</td>
</tr>
<tr>
<td>8</td>
<td>Construct/Install Infrastructure</td>
<td>Battery Charge/Change Station, Ratio Feeder</td>
</tr>
<tr>
<td>9</td>
<td>Six Month Trial</td>
<td>Modifications to Process, Audits and Review, Detailed assessment of performance.</td>
</tr>
<tr>
<td>10</td>
<td>Decision to Proceed with Performance Hire</td>
<td>Safety, Powercoal Investment Criteria.</td>
</tr>
<tr>
<td>11</td>
<td>Performance Hire</td>
<td>Audits and Review of Performance.</td>
</tr>
</tbody>
</table>

**PRELIMINARY EVALUATION**

**Shuttle cars - the existing system**

The existing coal clearance system utilised Joy 15SC Shuttle Cars and clearly an option to be carefully considered was to seek incremental improvements in performance from the existing system.

In considering this option the following factors were incorporated into a capital replacement financial model:

- The existing fleet was 16 years old;
- Overhaul costs;
- Maintenance costs and availability;
- The high cost of trailing cable repairs;
- Wheel unit costs;
- Average payload 9.5 tonnes; and
- Productivity plateau of 800 tonnes per shift average.

The financial model indicated that a quite modest productivity improvement would justify the capital cost of a new coal clearance system. Following detailed analysis, there was limited scope only for safety and productivity improvements with continued use of shuttle cars.

**Alternate coal clearance systems**

Alternate systems considered were:

- Continuous haulers - roof or floor mounted;
- Bridge conveyor systems;
- “Bendicar” system;
Battery powered coal haulers; and

Diesel powered coal haulers

Advantages and disadvantages

A preliminary evaluation of the alternatives was undertaken based on a simple ranking system. Battery haul cars were considered the most appropriate system for use at Cooranbong. The advantages and disadvantages of BPCHs compared to other alternatives is given in Table 2.

Table 2 - BPCHs Advantages and Disadvantages

<table>
<thead>
<tr>
<th>Safety Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>Elimination of trailing cables - arcing in hazardous zone.</td>
<td>Physical size of the BPCHs.</td>
</tr>
<tr>
<td>Manual handling - trailing cables.</td>
<td>Stored energy - potential to ignite CH4 during ventilation failure.</td>
</tr>
<tr>
<td>Articulation - improved roadway conditions.</td>
<td>Chemical energy - burns, fires.</td>
</tr>
<tr>
<td>Ergonomics - driver compartment/seat.</td>
<td>Driver visibility when loaded.</td>
</tr>
<tr>
<td>Driver compartment canopies.</td>
<td></td>
</tr>
<tr>
<td>No diesel fumes.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Productivity Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>16 tonne payload.</td>
<td>Less capacity than continuous haulage.</td>
</tr>
<tr>
<td>Rapid coal discharge.</td>
<td>Battery life - roadway conditions/grades.</td>
</tr>
<tr>
<td>Flexible wheeling routes.</td>
<td>Cycle time to achieve optimum battery performance.</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Cost Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>No trailing cable/wheel unit repairs.</td>
<td>Battery Charge/Change station.</td>
</tr>
<tr>
<td>Less mechanical components.</td>
<td>Requires special ratio feeder.</td>
</tr>
<tr>
<td>Advantage gained from one or more units.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Employee Morale Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>No diesel fumes.</td>
<td>Battery life/changing.</td>
</tr>
<tr>
<td>No trailing cables.</td>
<td></td>
</tr>
<tr>
<td>Flexible wheeling routes.</td>
<td></td>
</tr>
<tr>
<td>Not radically different to shuttle cars/training.</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Flexibility Advantages</th>
<th>Disadvantages</th>
</tr>
</thead>
<tbody>
<tr>
<td>Can handle variable seam conditions.</td>
<td>Requires battery charge station.</td>
</tr>
<tr>
<td>Flexible wheeling routes.</td>
<td>Requires special ratio feeder.</td>
</tr>
<tr>
<td>Materials transport.</td>
<td></td>
</tr>
<tr>
<td>Outbye maintenance.</td>
<td></td>
</tr>
</tbody>
</table>

| Other Advantages | |
|-----------------| |
| Proven technology - 20 years in USA. | |
In May 1997 the author undertook a study tour of high productivity United States mines using PCM systems and BPCHs. Particular emphasis was placed on visiting operations with “soft” or wet floor conditions and/or operations in dipping and undulating coal seams. Galatia Mine (Illinois) was also inspected. This mine uses diesel haul cars on a moderately soft floor in gateroad development. Productivity in gateroad development was 1000 tonnes per eight hour shift or 13 tonnes per face man hour.

Table 3 summarises productivity results and shows a comparison with Cooranbong using Joy 15SC shuttle cars

<table>
<thead>
<tr>
<th>Mine</th>
<th>Tonnes</th>
<th>Face Hrs</th>
<th>Manning</th>
<th>Tonnes/Man/Hr</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cooranbong - Best</td>
<td>2000</td>
<td>7</td>
<td>10</td>
<td>28.6</td>
</tr>
<tr>
<td>Grand Canyon - Avg.</td>
<td>2300</td>
<td>10</td>
<td>9</td>
<td>25.6</td>
</tr>
<tr>
<td>Unicorn - Avg.</td>
<td>1850</td>
<td>8</td>
<td>11</td>
<td>21.0</td>
</tr>
<tr>
<td>Darby Fork - Avg.</td>
<td>1300</td>
<td>8</td>
<td>10</td>
<td>16.2</td>
</tr>
<tr>
<td>Cooranbong - Avg.</td>
<td>800</td>
<td>7</td>
<td>9</td>
<td>12.7</td>
</tr>
</tbody>
</table>

Table 3, also shows that high productivity is achievable with BPCHs. All operators were enthusiastic about their particular brand of haulers. It should be noted that the author identified numerous factors that contributed to the productivity results in the United States mines and that the use of BPCHs was but one of these factors.

Two significant factors relating to the BPCHs observed in the mines visited were roadway conditions and the design and location of the battery charge/change stations.

Floor conditions were observed to be significantly better than typical Australian mines. This was in part due to geology, but, in this author’s opinion, equally due to road maintenance procedures. In all PCM operations a designated roadway cleanup employee and scoop-tram were part of the production crew.

With regard to underground battery charge/change stations, all were designed to be simple and mobile. Charge/change stations were regularly advanced with the panel and kept within 500 m of the face line. As well as maximising battery life, operators stated that this system allowed better inspections and maintenance and, if a problem did occur, a quicker emergency response.

All charge/change stations were unmanned and no operators were aware of any significant incidence (e.g. fire) involving a charging station in recent years. This was confirmed by independent sources through the United States' Mines Safety and Health Administration (MSHA).

TENDER ISSUES

Tender specification

In may 1997 a tender was issued seeking the supply of a coal clearance system utilising BPCHs for use in PCM systems at Cooranbong Colliery. Features of the tender were:

1. The number of BPCHs was not specified. A coal haulage rate (tonnes per hour) was specified to match the cutting rate of the miner.

2. The performance criteria specified was:
the system must be approved for use in NSW underground coal mines;

- System availability must exceed 97%;

- A coal haulage rate (tonnes per hour) from the continuous miner to the ratio feeder must be nominated and guaranteed under specified test conditions. This rate was to be verified once per month. A rate of 600 tonnes per hour would be necessary to match continuous miner performance; and

- The system must be compatible with conditions and the PCM process as practiced at Cooranbong Colliery

3. To submit a conforming tender, each tenderer had to spend at least 30 man hours underground to understand and be familiar with local conditions and mining processes.

4. Powercoal's preference was for a performance hire agreement based on tonnes hauled.

The tender was designed to ensure tenderers considered not only technology issues, but that they also consider how that technology would deliver performance as part of a total mining process. This caused tenderers to also consider issues such as wheeling routes, cycle times, charge station design and location, floor conditions and panel design.

It was Powercoal's intent to purchase performance rather than engineering potential.

**Tender outcomes**

After detailed evaluation, the tender was awarded to Long-Airdox Pty Ltd. Features of their offer included:

1. Detailed production cycle analysis including recommendations on pillar dimensions and wheeling routes to deliver superior productivity;

2. A battery management plan including charge station design and charging procedures;

3. The inclusion of a battery powered scoop-tram for roadway improvements and clean up;

4. Commitment to performance criteria - haulage rate and availability; and

5. Performance payments based on tonnes hauled.

Table 4 summarises technical aspects of the offer. Figs 1 and 2 show the Long-Airdox CHA818 Un-a-Hauler and Fig 3 shows the 488GLBC Un-a-Trac Scoop Tram.

The inclusion of the scoop tram was significant. Detailed modeling by Long-Airdox indicated a 1% improvement in roadway rolling resistance would deliver up to 20% improvement in unit productivity.

**APPROVAL AND INFRASTRUCTURE ISSUES**

**Battery charge/change station**

The Long-Airdox offer included a battery charge/changing station designed around a ground based turntable arrangement as shown in Fig 4. The turntable is powered by compressed air and allows discharged batteries to be uncoupled from the BPCH, the battery to be placed on charge and a fresh battery to be picked up. It is designed for single person operation. A single cut-through can accommodate two turntables that manages six batteries for two BPCHs.

The only disadvantage of the system is that the turntables are quite large and require some effort to erect. This renders the charge/change station less mobile and more difficult to construct. Battery charge/change stations observed in the United States often had the batteries simply sitting on the floor of a cleaned up roadway or hanging off chains from the roof. This made the charge/change stations simple and easy to move.
Table 4 - Technical description

<table>
<thead>
<tr>
<th>Component</th>
<th>Specification</th>
</tr>
</thead>
<tbody>
<tr>
<td>Battery Hauler</td>
<td>Long-Airdox CHA818 Un-a-Hauler</td>
</tr>
<tr>
<td>Number of Vehicles</td>
<td>3</td>
</tr>
<tr>
<td>Capacity</td>
<td>16.4 tonnes</td>
</tr>
<tr>
<td>Tram Speed</td>
<td>8 km/hr</td>
</tr>
<tr>
<td>Tram Height</td>
<td>1840 mm</td>
</tr>
<tr>
<td>Ground Clearance</td>
<td>310 mm</td>
</tr>
<tr>
<td>Mass - Empty (with battery)</td>
<td>28,530 kg</td>
</tr>
<tr>
<td>Max. Operating Grade</td>
<td>1:4</td>
</tr>
<tr>
<td>Batteries</td>
<td>3 per vehicle</td>
</tr>
<tr>
<td>Battery type</td>
<td>1375 Amp-hour lead acid</td>
</tr>
<tr>
<td>Scoop Tram</td>
<td>Long-Airdox 488 GLBC Un-a-Trac</td>
</tr>
<tr>
<td>Number of vehicles</td>
<td>1</td>
</tr>
<tr>
<td>Tram Speed</td>
<td>8 km/h</td>
</tr>
<tr>
<td>Tram Height</td>
<td>1575 mm</td>
</tr>
<tr>
<td>Ground Clearance</td>
<td>380 mm</td>
</tr>
<tr>
<td>Mass with battery</td>
<td>19,320 kg</td>
</tr>
<tr>
<td>Batteries</td>
<td>3 per vehicle</td>
</tr>
<tr>
<td>Battery Type</td>
<td>1000 Amp-hour lead acid</td>
</tr>
<tr>
<td>Load Capacity</td>
<td>9000 kg</td>
</tr>
</tbody>
</table>

To determine the best system, two charge stations were constructed - one turntable system and one ground based. For the ground based system, batteries are placed on the ground for charging and cooling and picked up using a jumper cable as shown in Fig 5. The jumper cable is managed by a lightweight monorail hung from the roof.

Both systems will be trialed for two months and a decision will be made on the preferred system.

Fig. 1 - Long-airdox CHA818 un-a-hauler - battery end
Fig. 2 - Long-airdox CHA818 un-a-hauler - coal discharge end

Fig. 3 - Long-airdox 488GLBC un-a-tram scoop tram
Fig. 4 - Battery charge/change station - turntable system
Fig. 5 - Battery charge/change station - ground based system
The Coal Mines Regulation (Electrical - Underground Mines) Regulation 1984, Clause 38 (b) (NSW Govt 1984) states:

"The manager of a mine shall ensure that every battery-charging station at the mine is –

1. lined with non-flammable material;
2. provided with suitable and sufficient means for combating outbreaks of fire;
3. so designed and operated that the air from across the battery racks passes directly into return air, unless its situation is otherwise approved; and
4. continuously manned during the time that battery charging is in progress, unless a monitoring system approved for the purpose has been installed to enable battery charging to be performed unattended."

Compliance with these provisions and other safety requirements is achieved by:

- The station is located outside the hazardous zone;
- Ribs stonedusted to a standard of not less than 90% incombustible content with stonedust wet applied;
- Floor thoroughly cleaned and stonedusted;
- Direct ventilation to return - 3 m³/s - controlled by adjustable regulator; and
- Lighting, first aid equipment, telephone, signs, quick flush shower located at the charge station.

A Risk Review was undertaken and the following features are in place to gain approval for unmanned operation:

- CO monitoring with power trip and surface alarm;
- Air velocity monitor with power trip;
- Maintenance and inspection procedures;
- Safe Operating Procedures; and
- Provision to disconnect electricity from the surface.

Based on the above monitoring system, an application has been made to the Department of Mineral Resources (DMR) to have the station approved for unmanned operation. At the time of writing, approval for unmanned charging of short duration (up to two hours between shifts) has been granted.

**Ratio feeder**

Due to the coal discharge action of the BPCHs, a purpose-built ratio feeder is required incorporating an open, drive-in rear section. It was intended to use a Stamler BF-14B-9-7C Feeder Breaker with an open back and high sides. This type is commonly used in the United States. This design, however, could not be used in its standard form due to recent guidelines issued by the Department of Mineral Resources. The guidelines, titled MDG31 “Design Guidelines for Construction of Feeder Breakers”, apply to all ratio feeders supplied after 1 January 1997. Clause 3.8.7 of MDG31 states:

"the inbye conveyor feed end shall be fitted with an end plate across the full width of the surge bin the height of which is equal to the lowest side plate for the remainder of the surge bin."

This clause does not permit the use of an open back feeder with high side walls. Officers from the DMR confirmed that the guidelines could only be satisfied by a "hard" barrier. "Soft" barriers such as warning lights and restricted access zones were not acceptable.
Stamler, after many “brainstorming” sessions with local and overseas engineers and involvement from colliery personnel, have designed a heavy duty plastic drive through barrier that is erected and hung from the roof. It must be dismantled, moved forward and re-hung from the roof with each belt move. The inclusion of the barrier also had some unintended consequences during the training programme. The barrier meant that shuttle cars and BPCHs could not use the same ratio feeder. A phased introduction of the BPCHs became difficult as detailed in a later section.

IMPLEMENTATION PROGRAMME

Risk reviews/HAZOP studies

Four formal risk reviews were carried out during the implementation process:

1. A risk review carried out by Long-Airdox to identify core risks as part of the Tender Conditions;
2. A risk review into the design, construction and management of an unmanned battery charge/change station;
3. A risk review into changing and charging of batteries at a charge/change station; and
4. A risk review into the operation of BPCHs in a PCM production unit including wheeling routes, discharge into the ratio feeder, battery management, ventilation requirements and roadway grades.

Each risk review team consisted of management, team leaders, operators and technicians from Long-Airdox. The risk review techniques used were based on DMR MDG 1010 “Risk Management Handbook for the Mining Industry”. From each of the above, Safe Operating Procedures (SOPs) were developed for each process.

Some key risks identified during the risk review process were:

- Poor off-drivers side visibility while reversing a loaded BPCH;
- In the event of a ventilation failure, BPCH’s must be immediately removed from the hazardous zone;
- The BPCH’s operate with little noise and no trailing cable. There is increased risk of collision in a highly productive PCM unit; and
- Risk of crush injury of pedestrians while changing batteries in the battery charge/change station.

Training

As part of their tender offer, Long-Airdox supplied a comprehensive training programme that incorporated:

- vehicle operation;
- battery management including charging/changing;
- engineering/103 Scheme inspections and maintenance; and
- troubleshooting procedures.

A trial area was set up on the surface with roadway/cut through dimensions painted on a paved area. A battery charge station was also constructed. Employees from all shifts were trained in the SOPs and appointed competent in writing by the Mine Manager.

Underground trials

Underground trials commenced on the 1 November 1997 in the Bay 1A panel. The battery charge/change station was located approximately 500 m from the face line. This location would allow a planned section move in December 1997.
It was originally intended to phase in the BPCHs by running one hauler with two Joy 15SC shuttle cars. As well as allowing improved operator training, there would be less risk to production results in the event of commissioning problems. The requirement for a hard barrier on the rear of the ratio feeder, however, meant that shuttle cars and BPCHs could not use the same ratio feeder.

As a compromise the open back ratio feeder was located as a side loading point one cut through behind the normal ratio feeder. Whilst allowing dual operation, the system was cumbersome (two feeder set ups per belt move including rear end barrier) and unproductive (the BPCHs had to travel an extra 200 m per cycle). Despite these inconveniences underground commissioning commenced in early November, 1997.

By allowing the BPCHs to run in parallel with the shuttle cars, focus has been on safe operation and operator familiarisation rather than productivity results. The parallel operation has also allowed battery charge/changing procedures to be reviewed and modified.

Battery management

From the outset, it has been emphasised to operators that battery changing is part of the production cycle. It must not be left to maintenance teams or the "next shift". It is anticipated that each vehicle will require a battery change once per shift. To ensure hauler performance, it is essential the battery management procedure is rigorously applied. Each hauler is designated a turntable and a set of three batteries. In each eight hour production shift each of the three BPCHs must change their battery once.

The protocol developed is that on each production shift, hauler #1 will leave the face area for a battery change at the start of the first miner flit. It is anticipated that the battery changeout will take 15 minutes (excluding traveling time to the charging station). Haulers #2 and #3 will repeat this procedure at the start of miner flits two and three. Discipline in this area is critical to system performance.

Floor management

A key feature of high productivity PCM units in the United States is roadway management. A designated employee is permanently employed using a scoop tram for roadway cleanup and repairs. The advantages of this are:

- improved safety - slips, trips, falls;
- improved productivity - faster flits and wheeling;
- reduced machine maintenance and repair costs; and
- improved battery life.

A battery powered scoop tram is included as part of the coal clearance system and, as part of the supply contract with Long-Airdox, it has been designated to work solely in the vicinity of the BPCHs.

To further improve roadway conditions it is proposed to trial "dragging rails" beneath the BPCHs to systematically grade roadways during production. This technique has proven successful in a number of mining operations.

Manning and equipment

Equipment to be utilised in the PCM unit with BPCHs is:

- 1 x Joy 12CM12 Continuous Miner
- 3 x Long-Airdox BPCHs
- 1 x Long-Airdox Scoop Tram - roadway cleanup
- 1 x CRAM Mobile Roof Bolter
**• 1 x 913 Eimco Roof Scaling machine**

**• 1 x Stamler Ratio Feeder**

Manning will consist of one Team Leader, one Engineering Technician and nine Production Employees. Target productivity is 1500 tonnes per 8 hour shift or 19.5 tonnes per face man hour. This is consistent with United States' PCM results.

### RESULTS TO DATE

A crucial aspect of the six month trial will be battery performance in the roadway conditions encountered at Cooranbong Colliery. At the time of writing, the battery powered scoop tram had not arrived. Systematic roadway cleanup as part of the production cycle, a key component of the system, will commence in January 1998.

Initial underground trials of the BPCHs has highlighted an issue that must be addressed by suppliers. Lead-acid batteries require "cycling" to reach maximum capacity. Advice is that to reach maximum capacity, a new battery will have to be cycled approximately 15 times. It has also been advised that optimum battery performance is achieved by loading the battery in a manner similar to its final use, that is, the cycling process is best carried out underground. For a single coal hauler with three batteries this requires approximately 15 days of operation to reach battery design capacity. From a production viewpoint this has a number of implications:

- if unit production is critical to the business, BPCHs must be used in conjunction with other coal clearance systems for a period of time;
- due to MDG31 guidelines, BPCHs and shuttle cars cannot share the same ratio feeder; and
- extra manning is required for little productive output during the cycling period.

Despite these commissioning issues the initial results are encouraging. Operator feedback is positive with drivers impressed with the ergonomic design of the operator’s compartment. Driver comfort on rough roads is excellent due to the improved seating design. Operators also state that vehicle articulation delivers excellent manoeuvrability and less roadway degradation around corners. Discharge into the ratio feeder has so far been trouble free. The elimination of trailing cables is seen as a significant safety and operational improvement.

The vehicles have handled the roadway conditions well but battery performance has been difficult to evaluate due to battery "cycling" requirements during battery commissioning. No vehicles, however, have yet been bogged and four wheel drive assist has been used infrequently.

Battery charging/changing has been performed in both charge stations with operators stating a preference for the turntable system. When picking up a battery the hauler must be perfectly “square on” and level with the battery. The turntable system does this more easily than the ground based system. Some refinements to the ground based system are being considered.

The six month trial will continue until the end of June 1998. Periodic reviews of total system performance will be conducted during this time. A decision to extend the trial into a long term performance hire arrangement will be made at the conclusion of the trial.

### CONCLUSION

Powercoal's Cooranbong Colliery has introduced battery powered coal haulers into the place changing process to improve safety and productivity. BPCHs have demonstrated high productivity and safe operation over many years in the United States' coal industry. The vehicles represent proven technology.

Powercoal's philosophy is to purchase performance rather than engineering potential. This is reflected in a partnering arrangement with the supplier, Long-Airdox, that incorporates:
• a six month trial period;
• performance based payments;
• guaranteed performance criteria - availability and haulage rate; and
• the option to extend the trial into a long term performance hire arrangement.

Limited operating experience to date clearly demonstrates safety improvements and improved flexibility compared to previous coal clearance systems.

Crucial to the success of the trial will be battery performance on the roadway conditions at Cooranbong Colliery. At present it is too early to draw any conclusions on this issue.
Operating a Continuous Haulage System at United Colliery

V Istin 1 and A Boyling 2

INTRODUCTION

United Collieries Pty Limited is an Australian mining company operating one underground coal mine, United Colliery at Warkworth in the Hunter Valley district of New South Wales. Centrally positioned, United Colliery is adjoined by other coal mining operations and is located 80km north-west of Newcastle and 20 kilometres south-west of Singleton. (Fig. 1) High standard road and rail links provide ready access to customers and suppliers.

Fig. 1 - Location plan

United Collieries Pty Limited is a joint venture between the United Mine Workers (CFMEU - Mining and Energy Division) and Abelshore Pty Ltd. The involvement of a union, in this instance the United Mine Workers, as a joint venture

1 General Manager United Collieries Pty Limited
2 Development Engineer United Collieries Pty Limited
partner and shareholder in a mining operation is understood to be unique in the world. An exploration authorisation was granted to the Australian Coal and Oil Shale Employees Federation in 1979. This area was a replacement for the Nymboida colliery lease which had been worked by the union until reserves were exhausted. The authorisation covered an area of approximately 10 square kilometres and contained both open cut and underground reserves in many seams. Open cut reserves however were insufficient to warrant the development of a major opencut mine and the majority of reserves were accessible only by underground mining.

A resource rationalisation with the adjacent Wambo mine took place in April 1991. Wambo gained four of United’s open cut coal seams and United gained access to an area of one seam within Wambo’s lease, the Woodlands Hill. The reserve reallocation allowed Wambo to expand their existing open cut mine and United to develop a viable underground mine, initially in the Woodlands Hill seam, with potential to later mine other seams.

**GEOLOGY AND RESERVES**

Coal seams within United’s lease belong to the Permian Age Wittingham Coal Measures. They are overlain and interbedded with hard competent sediments consisting of mudstones, siltstones and sandstones. In-situ resources of 262Mt in 9 seams have been delineated within the lease including 106Mt in the Woodlands Hill Seam. An adjoining authorisation area is being explored for both underground and open cut development potential.

The Woodlands Hill is the only seam currently being mined. Working height ranges from 2.5m in the far north of the lease to 4.3m in the south west, averaging 3.6m. The seam is overlain with mudstone and to assist with roof control approximately 300mm of top coal is left against the mudstone. The working floor is composed of approximately 200mm of carbonaceous mudstone underlain by siltstone or sandstone. Floor conditions have generally been good due to the comparatively strong floor material. Occasionally broken floor conditions have resulted from loss of floor horizon, generally in intersections, and in faulted areas where the floor strength is usually lower.

**OPERATIONS**

Development of the underground mine commenced in 1991 with the drivage of entries to the Woodlands Hill seam, 80 metres below the surface, and construction of surface infrastructure including mine fan, administration and bathhouse complex.

Production started in January 1992 with one continuous miner unit. The mine was expanded in October 1995 with the introduction of a second unit extracting pillars using mobile roof supports. In November 1996 a third unit commenced operation and surface infrastructure was expanded including a new mine fan and extensions to administration, stores and bathhouse areas.

**MINING METHODOLOGY**

Mine development commenced in January 1992 utilising Joy 12CM12D continuous miners fitted with “on board” roofbolters loading into two shuttle cars. The continuous miner would cut out for 6 metres and then cutting would cease while the roof was supported. Average productivity was 540 tonnes per unit shift.

In pursuit of improved productivity the system was modified in October 1993 with the introduction of a “dual miner panel” system. This system utilised two continuous miners used alternately with roof support being erected in one face while production continued at an alternative face. Three shuttle cars were used in the section but only two on production at any time. Average production increased to 1,060 tonnes per unit shift.

In April 1994 the “cut and flit” system of mining was introduced with three shuttle cars used continually. Average productivity to date is approximately 1,200 tonnes per unit shift with a peak of 2,829 tonnes in an eight hour shift.
Current mining operations are based around three continuous miner sections. Two units, one on development using the cut and flit system and the other on pillar extraction, both utilise shuttle cars to load coal from the continuous miner onto the panel conveyor belt. The third unit operates in development using the cut and flit system but utilises a chain type continuous haulage system instead of shuttle cars. In 1998, run of mine output of two and a half million tonnes is planned off the three units.

CONTINUOUS HAULAGE

In 1997 United purchased a Chain Haulage System to be used in both cut and flit development and subsequent pillar extraction. The system is 103 metres long and will increase the productivity of both cut and flit development and pillar extraction to approximately 2,500 tonnes per unit shift.

Joy Mining Machinery was the successful tenderer from a group of three suppliers, all from the USA. The Joy system was selected after an exhaustive tender evaluation.

Tender specifications included the following requirements:

- Carrying capacity of 30 tonnes per minute (to match the rated capacity of the continuous miners);
- A minimum length of 100 metres;
- Ability to gain NSW approval which would include many safety requirements in excess of the American systems;
- Radio remote control of the inbye unit;
- PLC control;
- Visual display of the machine status in each operators cab;
- Ability to work on in 5 grades;
- Ability to operate in 5.0 m roadways with square (90 degree) intersections;
- Sound mechanical and electrical design;
- Tight delivery schedule;
- Strong support from the manufacturer; and
- Competitive price.

A representative working group consisting of a cross section of United employees visited six (6) high productivity mines using continuous haulage systems in the USA. The manufacturing plants of the three tenderers were also inspected. The experience gained by the visit not only assisted the working group in carrying out an extensive evaluation process, but it was also invaluable in customising a haulage system to United's requirements. Observations made in the American mines also assisted in the development of plans and procedures for safe and efficient operation of the system at United.

The continuous haulage system shown in Fig. 2 is made up of the following components.

1. 1 x breaker mobile bridge carrier
2. 4 x mobile bridge carriers.
3. 5 x bridges
4. 1 x belt dolly

5. 120 of metres of rigid belt frame.

Fig. 2 - Chain haulage system general arrangement

United's continuous haulage system is the first equipped with a sophisticated Joy JNA computer control system, providing onboard computer control of overload levels, start-up sequences and various safety features. The computer system also
provides each operator with information on the operational status of the whole machine and assists in diagnosing system faults. United's continuous haulage is fitted with a voice communications system linking each haulage operator to the continuous miner operator, other crew members and section visitors.

The inbye unit is equipped with radio remote control which enables that unit to operate beyond the last roof support. This feature allows the current maximum seventeen (17) metres cut to be achieved with the inbye unit operator remaining in his cab under supported roof, and provides the potential to increase the depth of cut to as much as thirty six (36) metres as experience is gained with the system.

Successfully extending the depth of cut significantly beyond the current seventeen (17) metres will require issues such as directional guidance and horizon control of the miner to be addressed.

Technology addressing some of the issues of deep cut mining is currently being developed by Joy Manufacturing for highwall mining, and United is working with Joy in assessing the applicability of that technology in the continuous haulage operation. Installation of television cameras on board the continuous miner is being considered as a means of enhancing driver vision when deeper cuts are used.

**Panel design**

The first panel (five heading layout) as shown in Fig. 3 has been designed to maximise production from the continuous haulage system. Design factors considered included the maximum reach of the haulage unit, optimum pillar size to ensure pillar stability and to satisfy statutory requirements, and the need for a safe and efficient method of pillar extraction utilising the continuous haulage system.

**Development mining**

The continuous haulage unit is used in a cut and flit development mining system. The depth of cut is initially a maximum seventeen (17) metres, however it is proposed to incrementally increase the depth of cut to a maximum of thirty six (36) metres as operational experience is gained and suitable controls are developed. Fig. 4 shows the mining sequence for development of the first panel using the continuous haulage system.

The continuous miner power supply cable is run along the full length of the chain haulage system minimising the need to manhandle the cable when flitting.

**Roof and rib support**

The roof is supported with 2.1 metre fully encapsulated AVH roof bolts with 150 mm steel plates. Rib bolts are not normally installed but have been occasionally used in faulted areas. Two (2) Fletcher CDDR-17 roof bolting machines, each equipped with two bolting masts are used to install the supports.

Roof support for 5.5 metre wide roadways consists of rows of roofbolts installed 1.4 metres apart and roof support for the conveyor belt roadway, which has a maximum width of 6.5 metres, consists of rows of roofbolts installed 1.0 metre apart in accordance with the mine support rules.
Fig. 3 - Typical panel layout CM in final sequence
The continuous haulage system is used in conjunction with a radio remote controlled Joy 12CM12D fitted with a 5.4 m Ventilation

The ventilation system in the chain haulage panel has been designed with the following features in mind.

- The requirement for relatively narrow roadways;
- The requirement for large pillars designed for subsequent extraction at depth;
- The need to cope with higher levels of dust and gas production;
- The higher advance rates reducing the ability to efficiently install ventilation;

Fig. 4 - Current sequence plan with compressed air fan locations

COAL98 Conference Wollongong 18 - 20 February 1998
The use of single pass continuous miners at United Colliery; and

The future requirements to cut out in excess of thirty metres.

The first continuous haulage panel has been designed as a five (5) heading layout with the conveyor belt heading being the centre heading. Intake air is directed along the two (2) right hand roadways of the panel and returned via the remaining three (3) roadways, one of which is the belt roadway (ie. homotropal ventilation). Fig. 5 shows a general layout of the panel ventilation.

Ventilation of the continuous miner working place is by exhaust ducting mounted onto the continuous haulage system. It runs from the coal receiving hopper at the rear of the continuous miner to the auxiliary fan in the conveyor belt heading. The ducting is a mixture of 700 mm diameter fibreglass and 720 mm diameter semi rigid spiral wound tube. This tube provides the flexibility required at pivot points and accommodates the 1.2 metre compression on the MBC dollies.

The fan is mounted on wheels and is located on the section conveyor belt frame. It is towed in and out along the conveyor by the chain haulage system. The fan, which is fitted with an attenuator, exhausts into the belt roadway. Ventilation to all other places is currently via a forcing fan (17.5m³/s open circuit capacity) located in the intake airway with 725mm diameter lay flat ducting directing air to each face. The air quantity at each face is regulated by a fibreglass butterfly valve located in a Tee piece on each intersection. The forcing ventilation system provides excellent ventilation for the roof bolters and diesels.

Due to the working height being lower than expected, we are currently experimenting with compressed air auxiliary fans positioned at the entry of each heading. Air is directed to each face via 500mm diameter layflat ducting. With this arrangement the chain haulage system will no longer have to pass under the layflat tubing and the excellent ventilation of the faces being bolted and cleaned will be maintained.

Pillar extraction

It is proposed to utilise the continuous haulage system to extract the pillars formed during development. The panel layout has been designed to take into account a proposed pillar extraction sequence as shown on Fig. 6 and mobile roof supports will be used in conjunction with the haulage. Details of the proposed pillar extraction method will be finalised after continuous haulage operational experience is gained and geological features in the mining area are taken into account.

Panel services

At full production it is anticipated that the panel services will be advanced every second day. To ensure these service moves are completed efficiently and on schedule, intensive planning and design work involving many United personnel was required.

Some of the results included:

- A redesigned panel conveyor structure for speedier extensions;
- An integrated transformer / DCB, track mounted for speedier power supply moves;
- Flexible high voltage cable loaded in to a purpose built trailer which allows the cable to be quickly installed and retrieved;
- Power supply cables in custom made lengths;
- The on board exhausting ventilation system and forcing ventilation layflat ducting which requires less labour to move; and
- A flexible fire line is used in the area of the belt frame and can be quickly advanced. It is later replaced by a steel fire line once the belt frame is inbye that location. This fire line was originally mounted to the belt frame.
Fig. 5 - Original ventilation layout
Fig. 6 - Typical extraction sequence
Manning

The continuous haulage system development crew is twelve (12) production mineworkers and one (1) supervising production mineworker. The production functions are generally as follows:

- 1 production supervising mineworker (deputy);
- 1 continuous miner operator;
- 5 continuous haulage operators;
- 4 roof bolter operators (when required); and
- 2 utility men

The inbye MBC operator also performs the function of miner cable hand if required. All crew members are fully trained and authorised to operate the continuous haulage system.

Electrical and mechanical mine workers are called into the panel on an as needed basis.

Procedures.

A number of risk assessments were held before the introduction of the chain haulage system. These were:

- Design Risk Assessment;
- Operational Risk Assessment; and
- Ventilation Risk Assessment

The Design and Operational Risk Assessments for the continuous haulage system were conducted by a team consisting of members of the haulage working group, mine management and Joy personnel, and were important in the development of many of the machine safety features.

All of the risk assessments identified the need for management procedures to ensure the safe and efficient operation of the system. As part of their training all operators were fully instructed in these procedures prior to their being authorised to operate the continuous haulage system.

Operational experience

It is early days in the use of the Chain Haulage System.

By the end of December the system performance was starting to come together with one shift exceeding 1,700 tonnes and many shifts exceeding 1,000 tonnes. Current average productivity though is still only around 900 tonnes per shift. As problems are identified they are worked on until remedied.

A summary of problems to date includes:

1. Onboard ventilation system.
   - Collapse of flexible ducting in the area near the exhausting fan;
   - Damage of components;
   - Seized wheel bearings on exhausting fan;
• Vibration trips on exhausting fan;
• Broken coupling between exhausting fan and CHS dolly; and
• Blockage of coal receival hopper ductwork with coal, causing collapse of tubing along the system.

2. Layflat ventilation.
• Layflat tubina torn down due to insufficient seam working height
• Coupling joints coming apart
• Vibration trips and low current trips on forcing fan

3. Water in the panel
• Excessive slurry on roads
• Excessive slurry on belt
• Excessive material carry back
• Soft floor

4. Soft floor and chain haulage digging up floor.
• CHS bogged and “stuck” in holes;
• Undulations in conveyor belt structure;
• Difficulty advancing conveyor belt frame;
• Broken connecting pins.

5. Damaged and tangled power supply cables to CHS

6. Trapping shoes of CHS and exhausting fan damaged belt signalling system

7. Inexperienced operators
• Slower flitting;
• MBC’s getting bogged and stuck;
• MBC’s tearing up floor; and
• Not working as a cohesive 5 person machine.

8. Chain Haulage System
• Broken connecting pins;
• Broken support for connecting pin;
• Area lighting incorrectly located resulting in damage;
• Unreliable voice communications hardware;
• Radio remote control problems;
• Excessive water usage adding to panel water problems; and
• Damaged cab support rams.

9. Hydraulic and electrical problems on the CHS

10. Continuous miner drivers adapting to cut coal continuously instead of the stop / start technique used with shuttle cars

11. Initial high downtime on the continuous miner

A limiting feature of the chain haulage system is that in general one problem stops the total system and the movement of coal.

Noticeable improvements came about after the groundwater make in the panel and water discharge from the CHS and miner was controlled and reduced.

Unfortunately, as many of the problems with the continuous chain haulage and ventilation systems were being solved, the panel advanced into a zone of seam faulting. The associated strata dislocation and weak roof necessitates a reduction in cut out distances, thus requiring the CHS to be flitted more often, consequently reducing productivity. The faults are accompanied by an increased methane make causing the continuous miner cutter heads to automatically stop when the sensor detects 1.8% methane. To overcome the increased methane make, further improvements of the on board ventilation system are being directed at achieving higher air quantities and advancing the exhausting air ducting closer to the face. The final design may have the ventilation ducting directly connected onto the continuous miner.

United is also looking at the processes of cutting and flitting to identify areas for performance increases so that the available time is used more productively. The cutting cycle will be analysed so that the continuous miners are operated closer to their peak cutting rate of 30 tonnes per minute rather than the 6 to 10 tonnes per minute rates (over a 5 minute cycle) being now achieved.

CONCLUSION

The introduction of the Chain Haulage System at United Colliery has shown the productive potential of this type of system in Australian mines. The productivity of the Chain Haulage System will continue to improve as overall panel downtime is reduced and operator experience increases. Further improvements in productivity will occur when areas relatively free of geological disturbances are mined and the implementation of deeper cuts (up to 36 metres) will enable a higher proportion of the available time to be spent cutting.
Dragline Digging Methods in Australian Strip Mines - A Survey

H Mirabediny¹ and E Y Baafi¹

ABSTRACT

Open cut mining in Australia is facing the greatest challenge in its history in attempting to compete not only with other operations internationally, but also with underground operations domestically. Most flat dip and shallow depth surface-mineable coal reserves have been depleted during the last two decades and new open cut operations must extract deeper coal deposits. As open cut coal mines move into deeper areas and the stripping ratios increase, the relative cost of overburden removal also increases. It therefore becomes even more important to design the mine around the optimum overburden removal scheme. The deeper mines are usually multi-seam operations with a more complex geology and with more geotechnical and hydrological problems. Deeper mines are subjected to greater problems requiring more detailed mine planning and design, such as selection of the optimum mining method and pit layout. In planning and design of such operations, the number of alternative methods which need to be considered is consequently greater.

Dragline productivity and its stripping capabilities are directly affected by the selection of digging method, strip layout and pit geometry. Every mine has a unique combination of geological conditions. The operating methods that work well at one mine may not necessarily work at another site. Selection of an optimal stripping method, strip layout and pit geometry for a given dragline must be considered with respect to the geological conditions of the mine. With increasing geological complexity of Australian strip mines, it is becoming more important to use sophisticated techniques such as computerised mine planning methods to assist in optimising dragline operations.

THE PROBLEM

In the past twenty years the walking dragline has emerged as the dominant overburden removal machine in surface coal mining operations in Australia. There are now over 60 large walking draglines operating in Australian open cut coal mines (Aspinal, 1992). Four new units were expected to be ordered in 1996 and possibly another four units in the next five years. The book value of these new draglines is about A$800 million (Hamilton, 1996). In NSW, there has been a significant growth in using dragline operations compared with other mining methods since 1980 (Fig 1). Unlike underground mining, the productivity of Australian open cut coal mining has been disappointingly static during the last two decades with the annual raw coal output per man employed remaining the same as it was in 1970/71 (Wentworth, 1988). Although there are several reasons for this steady status, the major factor is due to insufficient technical improvement in mining methods as the geological conditions become more complex.

Overburden depths at many mines have already reached depths which draglines alone cannot handle without additional pre-stripping equipment. Many Australian mining companies are currently faced with the decision either to continue stripping to increasing depths or to commence underground mining operations. These specific conditions require an extensive analysis of each dragline's working method to establish:

- the operating limits for the machine;
- the productivity during chop cut and rehandling operations; and
- the efficient sequences of different mining activities.

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A review of several case studies of stripping operations by Atkinson et al (1985) clearly indicated that the stripping capabilities of the draglines used in Australian open cut coal mines were not fully utilised, resulting in low operating efficiency. There are several ways to increase the efficiency of overburden removal operations, such as improved design of dragline components. However, dragline productivity improvement through the modification of the digging method is the most cost effective and usually the most efficient means (Pippenger, 1995). The feasibility of significant improvement in dragline performance (up to 20%) through modifications to the digging method has been reported by several mines. The idea of modifying the digging method becomes increasingly more attractive as stripping ratio increases during mine life, particularly in multi-seam operations.

In order to improve the efficiency of a dragline operation it is necessary to have a thorough understanding of the characteristics of the digging method and the sequencing of the excavation operations. There is no comprehensive study evaluating the various digging methods currently in use by Australian open cut coal mines. Very limited information can be found describing innovative digging methods and most of them are internal and confidential mine reports. Most of the available literature describe basic dragline digging methods applied to the US coal fields. Australian dragline mines generally have greater overburden and to some degree have more complex geological conditions than US and European strip mines. Small draglines are rarely used and no tandem dragline operation currently exists in Australia. Many Australian dragline operations are using innovative digging methods to cope with these more difficult geological conditions and to increase dragline capabilities such as maximum reach and dump height. Because of the deeper overburden, most Australian strip mines have wider pits, typically 60-80m versus 40-50m pit width overseas, to reduce overall rehandle, dragline walking time and avoid both spoil and highwall failures.

A study was conducted to highlight the current status of the use of dragline in Australian coal mining. As the first step a questionnaire was prepared and sent to twenty eight open cut coal mines with a total of about sixty large walking draglines as the major overburden removal units. The questionnaire sought information about general geology of the coal deposit, the mine’s dragline(s) and other major equipment specifications and details of the pit geometry with a particular reference to the dragline digging methods. A number of site visits was also undertaken to directly observe and evaluate current dragline operations.

Of the twenty eight mines, twenty one mines (75%), covering fifty one dragline operations responded to the questionnaire. One mine has stopped using its dragline. The remaining 25% did not respond because of either lack of

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2 NSW Coal Industry Profile, 1996

**COAL98 Conference Wollongong 18 - 20 February 1998**
Operational data or the company did not have personnel available to gather the requested data. The information provided by the mines was classified according to the mine geology. The details included number of dragline passes, number of lifts per pass, dragline positions, whether or not a throw blasting technique is used, and cut and spoil procedures. The results of the questionnaire have been summarised in Table 1.

**RESULTS**

Various sizes of draglines are in use in Australian mines. The bucket size of the current draglines varies over a wide range of 12 to 103 m$^3$. Normally smaller draglines are used to remove the shallow depth interburdens (less than 30 m). Most of the recently ordered draglines or those under contract have larger stripping capacities when compared with the old generation of draglines (Seib and Carr, 1990). The dominant form of dragline ten years ago was a medium size dragline such as BE 1370W or Marion 8050 with bucket capacity around 47 m$^3$ (Atkinson et al, 1985). The new generation of draglines in Australian mines have an average bucket capacity around 75 m$^3$. Contributing factors toward the very large draglines are the increasing overburden depths, the need to increase stripping capacity of the mine to reduce unit stripping cost, and advances in dragline manufacturing technology. Fig 2 shows the changes in dragline size and its stripping capability during the last two decades.

Ideally the digging method which results in the highest coal exposure rate should be adopted for a particular operation. The choice of strip geometry is mainly governed by the selected stripping method and the size of dragline. Seven digging methods were identified to be representative of most of the Australian dragline operations. The common stripping methods were:

- simple side cast;
- standard extended bench with an advance bench;
- split bench (deep stripping);
- chop cut in-pit bench;
- extended key cut;
- single highwall and double lowwall multi-pass; and
- double highwall and single lowwall multi-pass.

In the last ten years as shown in Fig 3 there has been a significant tendency towards digging techniques with higher productivity such as extended key cut and in-pit bench methods. There are a variety of reasons for modifications to the conventional techniques, including:

- changes in geology such as significant increases in overburden depths;
- introduction of more efficient cast blasting techniques;
- development of multi-seam operations; and
- requirement for closer control on production costs.
Table 1- Summarised results of the digging method survey

<table>
<thead>
<tr>
<th>No</th>
<th>Number of Seams</th>
<th>Coal Thickness (m)</th>
<th>Waste Thickness (m)</th>
<th>Strip Width (m)</th>
<th>Model</th>
<th>Bucket Size (m$^3$)</th>
<th>Operating Radius (m)</th>
<th>Dump Height (m)</th>
<th>Dig Depth (m)</th>
<th>Digging Method</th>
<th>Productivity (Mbcm/y)</th>
<th>Digging Method Description</th>
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</thead>
<tbody>
<tr>
<td>1</td>
<td>1</td>
<td>8</td>
<td>55</td>
<td>80</td>
<td>Marion 8200</td>
<td>55</td>
<td>87.2</td>
<td>45</td>
<td>47</td>
<td>Modified Low wall Technique</td>
<td>12 (Total)</td>
<td>After a cast blast, a small fleet of shovel and truck reduces the overburden depth from 35m to 55m. Then the dragline with a modified lowwall pass removes the rest of the overburden. Using this method, rehandle is reduced from 30% to 7%.</td>
</tr>
<tr>
<td>2</td>
<td>1</td>
<td>0.5 - 6</td>
<td>25</td>
<td>60</td>
<td>BE 1370-W</td>
<td>47</td>
<td>87.2</td>
<td>57</td>
<td>47</td>
<td>Cast Blasting &amp; Extended Bench</td>
<td>9.8 (Total)</td>
<td>In some areas the dragline removes two seams in one pass, chopping the first interburden. The main overburden is initially cast blasted with 30% throw, then the dragline removes remaining material using an Extended Bench method.</td>
</tr>
<tr>
<td>3</td>
<td>[3] Seam 1</td>
<td>0.5 - 1</td>
<td>26 - 30</td>
<td>60 - 70</td>
<td>BE 1570-W</td>
<td>52</td>
<td>92</td>
<td>47.8</td>
<td>45.7</td>
<td>Three Pass Method</td>
<td>6.5 (Prime)</td>
<td>The first pass method is a standard underhand technique with a highwall keycut. The spoil is directly cast into the previous void with no bridging involved. In the second and third pass dragline technique is a lowwall pass involving the digging of the mid-burden from lowwall pad.</td>
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<tr>
<td></td>
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<td>4</td>
<td>1 (up to 4 splits)</td>
<td>18 - 22</td>
<td>65</td>
<td>70</td>
<td>BE 1350-W</td>
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<td>85</td>
<td>30</td>
<td>45</td>
<td>Two Highwall Passes</td>
<td>10 (Total)</td>
<td>Two highwall pass standard method. The first pass is a Simple Side Casting with some rehandle for the key cut. The second pass is a standard key with the Extended Bench.</td>
</tr>
<tr>
<td>5</td>
<td>1 (up to 6 splits)</td>
<td>20</td>
<td>70</td>
<td>80</td>
<td>Marion 8750</td>
<td>103</td>
<td>87</td>
<td>57</td>
<td>63</td>
<td>Standard Extended Bench &amp; Advance Bench</td>
<td>22 (Total)</td>
<td>A single pass Extended Bench method where overburden thickness is less than 45m. A single pass Extended Bench with overhand chopping and cast blasting for more than 45m overburden. A two pass Extended Bench where overburden thickness exceeds 60m. The first pass is more productive.</td>
</tr>
<tr>
<td>No</td>
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<td>Geology Condition (m)</td>
<td>Dragline Specification</td>
<td>Digging Method</td>
<td>Productivity (Mbcm/y)</td>
<td>Digging Method Description</td>
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<td>6</td>
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<td>90</td>
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<td>Extended Key Cut</td>
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<td>33-40</td>
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<td></td>
<td></td>
<td>10.5 (Prime)</td>
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<td>Lowwall In-Pit Bench</td>
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<td>[3] Seams 1</td>
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</table>

Two of the three pits are mined using standard Extended Bench method with a 10 metres advanced bench, and the third pit uses Extended Key Cut associated with cast blasting.

The overburden is removed in a two-stage operation. The first stage involves the use of shovel/loader and truck pre stripping operation to provide a dragline working level of 33 m. An 11% improvement in performance was obtained in 1993 changing digging method from Extended Key Cut to Extended Bench.

The overburden is removed in a two-stage operation. In the first stage dragline sits on a pad prepared on spoil pile and chops a narrow main cut. The material is spoiled in previous pit to make a new pad (bench) for next dragline position. In the second stage dragline removes the old pad and spoils material into final position.

The first pass method is a standard underhand technique with a highwall keycut. The spoil is directly cast into the previous strip void with no bridging involved. In the second and third pass the digging method is a lowwall pass involving the digging of the mid-burden from lowwall pad.
<table>
<thead>
<tr>
<th>No</th>
<th>Number of Seams</th>
<th>Coal Thickness (m)</th>
<th>Waste Thickness (m)</th>
<th>Strip Width (m)</th>
<th>Geology Condition</th>
<th>Dragline Specification</th>
<th>Digging Method</th>
<th>Productivity (Mbcm/y)</th>
</tr>
</thead>
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<tr>
<td></td>
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<td>Model</td>
<td>Bucket Size (m³)</td>
<td>Operating Radius (m)</td>
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<td>10</td>
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<td>70</td>
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<td>Marion 8050</td>
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<tr>
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<td>BE 1370-W</td>
<td>47</td>
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<td>3 BE 1370-W</td>
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<td>2 Marion 8050</td>
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<tr>
<td></td>
<td>Seam 2</td>
<td>4 - 6</td>
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<td>[2]</td>
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<td>15 - 45</td>
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<td>2 BE 1350-W</td>
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<td></td>
<td>Seam 2</td>
<td>8 - 10.5</td>
<td>10 - 30</td>
<td></td>
<td></td>
<td>4 Marion 8050</td>
<td>48</td>
<td>87.2</td>
</tr>
</tbody>
</table>

Digging Method Description

The Extended Bench method is used in the single seam. The maximum overburden depth is generally 45m. The second method (Extended Key Cut) is employed in multi-seam. The critical factor in this method is spoil room. When designing a multi-seam area, a maximum spoil height of 80m is used.

A trial was implemented at the mine to establish an Extended Key Cut method. Although higher swing angles are required in this technique, the overall coal exposure rate is increased due to lower rehandling of the material. Rehandle decreased from 63.4% to 47.1% and a 17% time saving was provided through the implementation of the new method.

The first interburden is cast blasted and the material is pushed with dozer or a small fleet of truck and shovel to make an in-pit bench for dragline. The dragline pulls back material of the second interburden.

Coal is removed by open-cut mining methods, using five draglines for overburden removal. A BWE and associated conveyors used to assist in pre-stripping operations. Two truck and shovel stripping fleets also operate at the mine.
<table>
<thead>
<tr>
<th>No</th>
<th>Number of Seams</th>
<th>Coal Thickness (m)</th>
<th>Waste Thickness (m)</th>
<th>Strip Width (m)</th>
<th>Geology Condition</th>
<th>Dragline Specification</th>
<th>Digging Method</th>
<th>Productivity (Mbcm/y)</th>
<th>Digging Method Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>14</td>
<td>2</td>
<td>4 - 5</td>
<td>30 - 55</td>
<td>60</td>
<td></td>
<td>3 BE 1370-w</td>
<td>Chop Cut In_Pit Bench &amp; Stacked Multi-Seam Method</td>
<td>N.A.*</td>
<td>The first method involves a three-pass operation in conjunction with cast blasting. Using the chopped material an in-pit bench is constructed progressively along the pit to complete the first pass. The final pass involves removing the remaining materials in spoil pile. This method showed a 20% increase in the uncovering rate compared with standard Extended Bench method as a result of a decrease in rehandle about 25%.</td>
</tr>
<tr>
<td>15</td>
<td>1 (up to 3 Splits)</td>
<td>4 - 6</td>
<td>20 - 60</td>
<td>45 - 70</td>
<td></td>
<td>4 Marion 8050</td>
<td>Extended Bench, Extended Key Cut &amp; In-Pit Bench</td>
<td>25.2 (Total)</td>
<td>A two-pass sequence with the alluvial unit being stripped down to expose the top of the sandstone forming an in-pit bench on which the dragline would operate to strip out the sandstone unit. Drilling and blasting operations would follow the stripping of the first pass.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>BE 1370-W</td>
<td></td>
<td>15.0 (Prime)</td>
<td></td>
</tr>
<tr>
<td>16</td>
<td>5</td>
<td>3.5 - 4</td>
<td>15 - 50</td>
<td>60 - 70</td>
<td></td>
<td>Marion 8200</td>
<td>Extended Bench, Extended Key Cut &amp; In-Pit Bench</td>
<td>39.4 (Total)</td>
<td>A combination of three common methods is used. The current methods are standard Extended Bench with chop, in pit bench method and Extended Key Cut. A 72 m³ replacement dragline was erected on site in December 1993.</td>
</tr>
<tr>
<td></td>
<td>Seam 1</td>
<td>0.5 - 1</td>
<td>26 - 30</td>
<td></td>
<td></td>
<td>Marion 8200</td>
<td></td>
<td>29.0 (Prime)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>Seam 2</td>
<td>0.5 - 1</td>
<td>1.5 - 5</td>
<td>40 - 50</td>
<td></td>
<td>Marion 7900</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Seam 3</td>
<td>0.4 - 4</td>
<td>2 - 6</td>
<td></td>
<td></td>
<td>Marion 7901</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>BE 1370-W</td>
<td>Single Highwall &amp; Double Lowwall passes</td>
<td>9.0 (Prime)</td>
<td>The first pass method is a standard underhand technique, with a highwall keycut. The spoil is directly cast into the previous void with no bridging. In the second and third passes digging technique is a lowwall pass involving the digging of the midburden from lowwall pad.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>No</td>
<td>Number of Seams</td>
<td>Coal Thickness (m)</td>
<td>Waste Thickness (m)</td>
<td>Strip Width (m)</td>
<td>Model</td>
<td>Bucket Size (m$^3$)</td>
<td>Operating Radius (m)</td>
<td>Dump Height (m)</td>
<td>Dig Depth (m)</td>
</tr>
<tr>
<td>----</td>
<td>----------------</td>
<td>-------------------</td>
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<td>----------------</td>
<td>-------</td>
<td>-------------------</td>
<td>----------------------</td>
<td>-----------------</td>
<td>--------------</td>
</tr>
<tr>
<td>18</td>
<td>[3] Seam 1</td>
<td>2 - 4</td>
<td>12 - 50</td>
<td>50 - 70</td>
<td>4 Marion 8200</td>
<td>57</td>
<td>87</td>
<td>48</td>
<td>60</td>
</tr>
<tr>
<td></td>
<td>Seam 2</td>
<td>0.5 - 3</td>
<td>6 - 30</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>19</td>
<td>Seam 1</td>
<td>2 - 19</td>
<td>13 - 32</td>
<td>55 - 60</td>
<td>Marion 7820</td>
<td>28</td>
<td>80</td>
<td>45</td>
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<tr>
<td></td>
<td>Seam 2</td>
<td>2 - 10</td>
<td>6 - 22</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
The selection of the best digging method depends on a combination of geological conditions, dragline size and characteristics, and management planning targets. The nature of the coal deposit and geological conditions such as the number of seams, overburden/interburden thickness and coal thickness are among the most important factors governing the choice of a digging method. Other factors such as geotechnical conditions, spoil stability, blasting techniques, material strengths and engineering and operator's experience are also important in the selection of a digging method. The combination of various factors results in a wide variety of methods at strip mines. Shared experience among different sites of a company owning various draglines is an important factor in the selection of a digging method. For example, BHP-Utah Coal Limited (BUCL) operates 35 draglines of varying sizes across the Bowen Basin of Central Queensland (Hill, 1989). The four common methods used by the BUCL group are:
1. standard extended bridge;
2. deep prestrip (split bench);
3. extended key cut; and
4. in-pit bench.

The stripping operations commenced with box-cuts on the shallow area at depths of 15 to 25m. The depths have increased over the years and average overburden depths now are around 50 to 55m in single seam operations. In many cases additional waste stripping is occurring ahead of dragline operation. In some instances, draglines are being used to dig depths as much as 70 metres.

Unlike overburden depth which is mainly governed by the geology, strip width is an operating factor which can be varied within a practical range. Variations in strip width affects dragline productivity. Pit geometry, especially the strip width, must be evaluated in conjunction with the digging method adopted by the mine. Wide strips (greater than 60m) are more preferable for methods such as the standard extended bench method due to the reductions in rehandle, while narrower pits are more productive for methods using a cast blasting technique, such as extended key cut or in-pit bench method. The strip widths currently employed ranged from 40 to 90 metres with an average of 60 to 70 metres.

Computer simulation and digging method selection

Draglines move waste at the lowest cost per unit volume only when they work within their normal range. Both efficiency and productivity of a given dragline drop off dramatically with changes in its effective operational factors. To improve the performance of a dragline, its mode of operation and influencing parameters must be fully understood and analysed. Finding the normal working ranges for a given dragline and optimising its operation requires repetitive arithmetic and analytic solutions. This problem is ideally suited to the application of computer aided simulation methods. Better mine planning and mining method selection through computer simulation has been successful in many cases and this has been strongly recommended for Australian operations (Atkinson et al, 1985; Hill, 1989; Wentworth, 1988; Aspinal, 1992; Sengstock, 1992). A computer simulation model which can simulate different mining methods (particularly the innovative ones) is a useful means for selection of the optimum dragline digging method for a given geology.

Computer simulation of dragline operation has the potential for rapid, low cost analysis of different mining scenarios. Simulation of the dragline operation enables an operator to test the logic of how the machine should be used, and the design of optimum operating methods for the varying mining conditions. Such an application may also be used as a training simulator or to evaluate dragline performance with a given set of geological and operational conditions. Computer simulation can also be used for evaluating proposals for modifications to existing operations and is also useful in comparing the performance of different types of new draglines which are being considered for purchase (Hill, 1989).

Due to the variety of digging methods currently used by open cut mines, a more general approach was necessary for simulation rather than using standard digging methods such as extended bridge. As a result of this study, a dragline simulation model has been developed which can be used in evaluation and optimising different dragline operations. A highly flexible simulation language “DSLX” was used to program different dragline digging scenarios in the model. Such an approach provided a library of various dragline digging techniques. The results from the simulation stage are then aggregated with time study data to estimate productivity and costs of the operation. The final decision then can be made based on either the highest production rate or the lowest unit cost from various digging techniques. An example of such a comparison for a single seam dragline operation is shown in Fig 4. The process of the modelling and results of different case studies have been discussed in detail in previous papers (Baafi, Mirabediny and Whitchurch, 1995; 1997).
CONCLUDING REMARKS

Productivity and efficiency of walking draglines can be improved by modifications in dragline digging methods. To select the most suitable digging method and working parameters for dragline operations, the first step is to analyse characteristics of various digging methods. A survey was conducted with the objective of evaluating the effects of various digging methods currently used by Australian dragline operations. The survey was conducted through sending a questionnaire to twenty eight mines covering more than sixty dragline operations. The questionnaire sought information about the general geology, major equipment specifications, digging methods and pit geometry. The surveyed showed that with the natural increase in overburden depths and complex geology, most strip mines have introduced various innovative dragline digging methods and larger draglines.

All the possible options can then be tested on a specific set of geological and mining conditions via a computer simulation model. Such an approach has been developed and applied to several Australian open cut coal mines both in NSW and QLD.

ACKNOWLEDGMENT

The authors are grateful to ECS International Pty Ltd for assistance and support provided to this study. The strip mines in the Hunter Valley (NSW) and the Bowen Basin (QLD) are acknowledged with thanks for permission to visit their operation and for providing data during the digging method survey.

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Ulan Highwall Project

P Ferguson

HIGHWALL MINING – DECISION TO MINE

Ulan had completed operations in the first Open Cut area due to high stripping ratios and the presence of geological features which destroyed a significant portion of the coal in the upper part of the seam. The pit had been abandoned in 1994 and operations had moved to a new location to the north west.

It was timely to investigate the possibility of recovering more of the coal seam via the use of secondary extraction methods as it could be demonstrated that it would not be viable to continue with either conventional Open Cut or Underground methods. Previous mine plans had centred around the backfilling of final voids with plant rejects and tailings, effectively precluding any further coaling in the region.

In early 1995 a review of Open Cut mining plans was undertaken. As a part of this review, two regions of coal, the old pit and a section of the new resource were identified as sites for remnant coal extraction. These areas would be available with only minimal additional preparation work.

The Ulan seam is a 10m thick seam (see Fig. 1), however the first 6m has a weight average insitu Ash of 36%. There is a limited market for this type of coal. Even after washing at 66% yield the product Ash is still approximately 22%. The Open Cut mines the Ulan seam in two passes. The first 6m bench is used for Domestic Supply Coal, while the lower 4m produces a good quality export coal. The underground mines only the lower ply of the Ulan seam - which has an insitu Ash of around 12%. As a significant proportion of the upper bench coal in the old Open Cut pit had been destroyed, Highwall Mining would only be used to access the D Section. (U/G Working Section). In other high ratio remnant areas Highwall Mining could not be justified to produce the high Ash product.

Ulan Coal Mines Ltd, Mudgee
The geotechnical design at Ulan was such that pillars should be long term stable. To this end, if it could later be determined that more of the coal seam could be extracted via Highwall Mining - the areas would allow this to occur.

With potential locations in mind it was then time to investigate the usefulness of this coal to the mine plan. Highwall mining was attractive to the Ulan Joint Venture because of a variety of key reasons:-

1. Increase Resource Recovery
2. Additional Source of Coal
3. Blending Synergies
4. Strategic benefits

Resource recovery is a key issue at Ulan and at the time of the investigation into Highwall Mining, Ulan was also preparing its application for a significant lease extension.

It was therefore important to be demonstrating an ability to utilise reasonably new technology to recover coal from remnant regions of the existing lease, while requesting extensions.

The Ulan mine is a Joint Venture project between Exxon Coal & Minerals Australia (36%), Mitsubishi Development (49%) and the State Superannuation Board of NSW (SASTC) (15%). Although Exxon also manages Lemington Coal Mine, near Singleton in the Hunter Valley (100%) - the two mines produce quite different coal types. To that end, production at Ulan is limited to an Open Cut and an Underground mine. Ulan is a 5Mtpa coal producer, with approximately 4.2Mtpa of export sales (Forecast 98). As a single mine operator, production exposure is centred on the performance of the companies underground mine. Highwall mining was then seen as an advantage to the company as it became the "Claytons", sister mine. With a project keen to expand, the benefit of another source of supply became an important concept in maintaining a program to support supply at existing levels.

With another potential source of supply coal came another source of quality. In recent years the underground has progressed through several panels of poorer quality (higher sulphur) coal. While the underground does not have the flexibility to move from area to area (from a pure quality perspective) - the Highwall operation does allow this flexibility. It was possible to develop a mine plan which would have the Highwall System working in a complimentary area to that of the Underground. The overall effect being to minimise the need for washing of coal types for blending. This, then has a marked effect on production costs.

Finally, from a strategic perspective, the Highwall System also offered some key advantages. With the mine looking at opportunities to expand production, the Highwall System offered a low cost means of ramping up production levels and helping to build markets prior to any project capitalisation - such as another longwall operation.

The initial success of the Highwall Mining operation at Ulan (forecast 1Mtpa) does mean that it can be evaluated, in its own right, as a medium term source of supply.

**SELECTION OF MINING SYSTEM**

During the planning evaluation process, from early 1995, Ulan had also started to investigate and evaluate Highwall Mining Technologies. At that time operations at Oakey Creek and German Creek were in progress utilising two different contractors. No other Continuous Highwall Mining Systems were operating in Australia. The main systems available at that point were Continuous Miner/Add Car Systems (Addington based technology), Push beam technology and the Archveyor system (Arch Minerals).

It was not until the middle of 1996 that Ulan finally had board approval to include the Highwall Mining into its business plan for the following year. Ulan set a target date to start Highwall Mining from July 1, 1997.

The second half of 1996 was spent setting up a contract, short listing potential contractors for the project as well as receiving and evaluating tenders. While this was happening Ulan was also involving the on site and district levels of the union and its workforce on an understanding and agreement for the project.
Additionally, the company had started discussing the project with the Senior and District Inspectors of Mines. Ulan was advised to submit a report on resource recovery of the areas to the Department of Mineral Resources (DMR) (Geology Branch) to evaluate whether or not the project could gain support by the DMR. A date was also set for a planning focus meeting to discuss the approvals and planning process required by Ulan. This meeting was duly held, November 13 & 14, 1996.

The contract to Highwall mine at Ulan was awarded to Mining Technologies Australia, (MTA) initially a letter of intent was provided on December 13, 1996 with formal approval signed February 27, 1997.

MTA were chosen because of the following key reasons:-

1. Could supply a system to start July 1, 1997.
2. Had demonstrated ability to Highwall mine in Australia.
3. Had a trained workforce - mostly Australians.
4. Would provide a system capable of mining 500m.
5. Had an inertial navigation system for guidance control.
6. Had an established safety record.
7. Had established procedures.
8. Were advanced in negotiating an Industrial agreement in the Western District of NSW.

The contract period was set by the mining of the previously discussed regions, or a time limit of 18 months, whichever came first.

**APPROVAL PROCESS**

As Ulan was to be the first Highwall mining project (utilising a continuous miner system) in NSW, the approval process would be the first of its type. The Department of Mineral Resources had, in 1996, established a guideline for the approval of Highwall Mining. However, the NSW approvals process and The Act were never written with a view to Highwall Mining. While many of the principles of Underground or Open Cut mining can be transferred to a Highwall mining system - the question as to whether or not the Highwall mining is covered by the Underground or Open Cut regulations was not clear cut.

It was established that the Highwall mining operation would be treated as an Open Cut operation, even though there would be no intended surface disturbance. Ulan Open Cut was applying for the approval to operate the system from a final void in the Open Cut. Although the mining system would be operating underground, all personnel would remain outside of the entry, similar to an Open Cut situation.

Broadly, the approvals process for Highwall mining can be summarised as follows:-

1. **DMR Approval Process:**
   a. Resource recovery approval.
   b. 7 year Open Cut & Highwall mining approval including:
      * Highwall Mining Management Plan & Risk Review
      * Geotechnical Design Report.
   c. Equipment approvals.
2. Department of Urban Affairs & Planning (DUAP) approval.
   a. Applications under S102 of the Environmental Planning & Assessment Act 1979 to alter Development Consent to include Highwall Mining.

3. Industrial Approval

   Contractors certified agreement with Construction, Forestry, Mining and Energy Union (Western District) to permit Highwall mining at Ulan Coal Mine Open Cut.

The approvals process was tackled on a number of fronts. The geotechnical report, supplied by the Minserv Group was available in March 1996. A second report investigating the effects of groundwater on Highwall mining at Ulan was completed by July of 1996. With exploration commitments, this part of the process took 8 months to complete.

Key findings in the geotechnical report into Highwall Mining at Ulan Coal revealed the following major points:

- The mining section would range in thickness from 2.9 to 3.6m.
- The DTP (D-Tops), forming the immediate roof of the Mining horizon would be a stable roof.
- The dips on the coal seam would range between \( \theta \) and \( \phi \)
- That coal strength in the mining areas ranged between 35 and 50 Mpa (UCS)
- 2.25m pillars and a 3.5m wide mining entry would provide factors of safety between 2.05 and 3.45 - depending on cover depth.
- Recommended that any Highwall Mining system used to cut slender pillars be fitted with guidance equipment to ensure pillar thickness was monitored and maintained.
- A face length recovery of 61% and maximum penetration of 350m would likely yield .3Mt from the regions proposed in the contract.
- Recommended additional work include an investigation into the effects of static pressure heads on ground water as it was suggested that heads exceeding 5m would be detrimental to mining.

Once a contractor had been established a formal risk review process was undertaken, which included key members from Mining Technologies Australia and The Principal. This occurred February 3 through 6 of 1997. At the end of this review a comprehensive management plan was constructed to deal with the risks identified.

The Seven Year, Standard Open Cut, approval was carried out concurrently with the management plan. Final approvals were received prior to mining commencing on July 1, 1997.

**COMMISSIONING AND PRODUCTION RESULTS**

The first MTA personnel arrived on site on May 16, 1997. Ulan was fortunate in that it was able to provide MTA with the old underground mine offices and surface facilities for the project. MTA refurbished the offices, an old warehouse and bathhouse for their use.

Equipment started arriving in June. Initially ancillary equipment, then loaders (988 & 990) and eventually 28, 12.5m long Addcars. The Hog Miner (ex Moura) and the launch platform came last of all. The launch platform was transported in two sections, due to height restrictions on public roads. The top deck weighted in excess of 150T and required a 4 x 50T crane lift to erect it on top of its base.

With the equipment ready to go and final approvals received, mining commenced in the west pit. The first mining hole achieved 300m and during the mining of this hole some, albeit minor, ground water depressurisation occurred. The second
hole was dry. The first month's mining resulted in the completion of 41 entries, each averaging only 141m. A total of 96,020 tonnes of coal was mined for the period. Significantly, the system was mining in a region in the west pit known to contain geological impediments. These consisted of a diatreme and dykes emanating from that source.

A table of production results by month and graphs of entry depths, Fig. 3 and monthly production tonnages, Fig. 4 are shown below.

Table 1- Production Statistics

<table>
<thead>
<tr>
<th>MONTH</th>
<th>TONNES MINES</th>
<th>ENTRIES</th>
<th>TOTAL DEPTH</th>
<th>DEPTH AUG ENTRY</th>
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</thead>
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<tr>
<td>JUL97</td>
<td>96,020</td>
<td>41</td>
<td>5,772</td>
<td>141</td>
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<td>AUG97</td>
<td>96,200</td>
<td>18</td>
<td>6,267</td>
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<td>* SEP97</td>
<td>105,600</td>
<td>23</td>
<td>6,694</td>
<td>291</td>
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<td>OCT97</td>
<td>122,800</td>
<td>20</td>
<td>7,392</td>
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</tr>
<tr>
<td>NOV97</td>
<td>124,000</td>
<td>16</td>
<td>7,741</td>
<td>484</td>
</tr>
<tr>
<td>*DEC97</td>
<td>113,400</td>
<td>18</td>
<td>8,051</td>
<td>447</td>
</tr>
<tr>
<td>TOTAL - 6 MTH</td>
<td>658,020</td>
<td>136</td>
<td>41,917</td>
<td></td>
</tr>
</tbody>
</table>

* Includes pit Move/Location Change

Fig. 3 - West pie Drives
The first two months of production were effected by the presence of geological intrusions - graph 1 clearly shows the presence of the diatreme. During the course of the first two months more addcars arrived at site, bringing the system up to its contract required 500m potential. Fig. 4 demonstrates monthly production from the Highwall System and Fig. 5 outlines the typical scatter of daily production from the system.

Moving to the next location in the west pit and mining back to the East certainly reveals a significant increase in average depth of penetration achieved away from the altered zone.

Besides issues associated with geology, teething problems existed with the navigational equipment on board the system. A significant number of hole throughs (one hole intersecting a previous entry) were occurring. It appeared that the slight cross grade on the pit dip was forcing the miner across into the previous entry while the miner was attempting to tram to stay straight - the effect was that miner was crabbing across into the previous entry, even though the heading was showing the hole alignment to be on course. The Horta is basically a super compass adapted by CSIRO for Highwall Mining applications and shows the heading at one point. It was not showing relative, lateral deviation. This problem has since been improved - to the point where hole throughs still occur but, perhaps only one in six or seven near full depth entries. The system continues to be developed by the contractor - and as Ulan now expects consistent performance at 500m, the
emphasis is on continuous improvement.

Previous to the Ulan operation, no remote Highwall Mining system had been set up to mine 500m. It is the author's understanding that a system in the USA was set up for 1500 feet (457m) - but that this depth had not been achieved. To that end, a hole mined at Ulan on 14 October 1997 at 506m became the first entry to pass the 500m barrier. A hole mined on 20 October 1997 at 510m (measured by addcars into pit) is therefore claimed to be the deepest ever entry from a remote Highwall Mining System.

While records of this order are certainly welcome, Ulan is more concerned with consistently deeper penetration results. Certainly future mining opportunities at Ulan will focus at depths beyond 500m. A continuous improvement process to increase depth of penetration does need investigation. After all, in 1996 500m was seen as a dream. Systems were only achieving 375m of penetration at other Australian sites. Ulan has good conditions for Highwall Mining and it is an expectation of the site's management that increases in depth are obtained.

While the contract production has continued to move in a pleasing direction, not all of the Highwall operation has run smoothly. Managing a system capable of producing over 100KT per month does require a commitment from the Principal. Ulan has suffered from larger than intended stockpiles in part because of industrial activity in the Open Cut Mine. The negotiation of an enterprise agreement has meant some flexibility has been lost because of protected actions being invoked. In the longer term Ulan will certainly be a very prospective site for ongoing and successful Highwall Mining conditions.

The safety features of the Highwall Mining system have not been lost on the principal. The contractor has mined for over 6 months without a lost time injury. Several Recordable incidents have occurred, and 4 soft tissue injuries have been recorded. Most notably these have involved finger/hand injuries, twisted ankles and a pinched foot as an operator attempted to kick out a piece of coal from the clevis plates of an addcar. The principal manages the contract with a project manager, a safety coordinator and normal Open Cut statutory cover. Daily meetings between the contractor focus on resolving safety and production related issues.

**POSSIBILITIES FOR THE FUTURE**

With initial results from the Highwall operation showing steady improvement in resource recovery it has been possible to further investigate the future of this mining method at Ulan. Several ideas for the future are being evaluated. The first it the use of trench excavation and subsequent Highwall mining to recover coal remnants under steeply tipping topography. Trench strategy involves accelerated opening of final pit endwalls which will ultimately allow Highwall mining on only one side of the trench (Fig. 4). Beyond this, a series of full blown trenches which would allow double sided excavation are also being evaluated.

The second, significant, Highwall project under evaluation involves a multi layer Highwall mining technique. In this situation the Highwall system would be used to extract the value, Coal section first (Fig. 5). After the extraction of this section, an initial bench of coal, or spoil, would be used to build the floor up to the base of the floor of the second, upper pass. Mining would then recommence at this level, with coal being used to continue to build the floor pad for the miner. As the miner progressed, coal could be recovered from the floor pad. Fig. 6 demonstrates the stacked upper pass.
The first stage of trenching has already commenced at Ulan. A trench is being excavated along the northern boundary of the existing Open Cut. The coal will be mined from the trench, thus creating a final endwall for Highwall operations to advance from. In a typical cover of 40m of overburden, the value of Highwall coal has the effect of reducing the in situ ratio (Prime/ROM T) from 2.6 BCM/tonne to 1.10 BCM/tonne - See table 2 - profile by depth of trench per 100m reserve block advance. It is assumed that double sided spoil stacking is approximately as productive as Highwall chopping. Simplistically it is 2.6/1.10 = 236% more efficient to orient stripping equipment to single sided trenching and benefit by gaining more "exposure" to coal. If it were possible to double side Highwall mine from the trench the effective strip ratio would further reduce to 0.70BCM/tonne. (371% more efficient than typical stripping). Ulan has a current Open Cut operation that is bounded on three sides by steep topography. It is therefore very prospective in terms of evaluating the usefulness of trench mining.

In terms of upper pass Highwall Mining, Ulan has had preliminary geotechnical design conducted on the upper pass section. Initial work suggests that with an increase in pillar width at the base to 2.75m it would be possible to mine sections of the plies above. As previously noted, the quality of the upper seam is poor (See Fig. 1) - however, the plies UBI & 2 and UC1 would provide a weight average product at 28% Ash. This may well be valuable as a future Domestic supply coal, to supply existing contracts or for future contracts if the coal price were to appreciate.

**SUMMARY**

Initial Highwall Mining results at Ulan have been pleasing. Ulan is a very prospective site which has both the areas and conditions to enable ongoing Highwall Mining operations. Ulan views the use of the Highwall Mining System as a means to maximise resource recovery by extracting coal which would not otherwise be taken. Indeed, the value of this coal continues to underpin future Open Cut operations and therefore assist in supporting 170 jobs in the Open Cut. The Highwall System has further created local opportunities for Mudgee people to gain skills in the Mining Industry. The Contractors employs 36 persons, most of whom are housed in the towns of Mudgee, Gulgong and the surrounding districts. The significance of this cannot be understated at a time when the coal industry is suffering substantially.

Ulan will continue to evaluate the potential for Highwall Mining operations as its business plan and economic conditions allow.
Hydraulic transport of coal has been used for many years on the West Coast of New Zealand to extract small blocks of steeply dipping coal. Recent development at Strongman 2 Mine has seen the use of high pressure monitors to cut coal and a corresponding increase in the scale of production.

This paper provides a brief introduction to hydraulic mining, describes the present system used at Strongman 2 and discusses future potential development.

HYDRAULIC EXTRACTION OF COAL IN NEW ZEALAND

The use of water for the transport of coal underground is recorded as far back as 1927. It became the method of choice for small private operations on the West Coast of the South Island, as it required very little capital investment and allowed small, steeply dipping blocks of coal to be mined competitively with the larger State owned conventional operations. Eventually as pressure on production cost increased and some union resistance was overcome, larger State operations such as Strongman (opened 1939) came to utilise hydraulic transport. This was generally from the face to an underground dewatering station followed by conventional transport to the surface. The use of large centrifugal pumps to complete the slurry journey to the surface also became common practice.

These operations were handicapped by the need for shotfiring to break the coal, the need for numerous production places and the extraction requirements of old workings laid out for conventional haulage, often with many collapsed roadways. Hydraulic transport allowed narrow splits to be driven through pillars in these fallen panels, enabling reasonably safe extraction to be carried out with good recovery ratios.

To trial monitor extraction a system was installed at the established Strongman Mine in 1992, utilising two high pressure pumps, a 200mm high pressure pipeline and monitor face units which had become surplus with the closure of Sunagawa Mine in Japan.

Extraction by monitor of old workings in Panel 3 and the Main East Headings was completed in 1994 and the mine closed.

The trial showed that the hard Strongman type coal could be cut successfully, particularly where significant roof weight was present, and the generally hard roof and floor resulted in high seam recovery and little floor contamination.

Previous shotfiring extraction had resulted in a number of goaf fires as small coal was often left behind after pillar falls. No sign of spontaneous combustion occurred during monitor extraction, due to the washing of fine coal from the goaf, the better recovery achieved, and the quick retreat rate.

While further drilling was carried out on the main Greymouth Coalfield, a smaller block of coal was identified to the east of Strongman Mine and the Strongman 2 Mine was established. The position of the Strongman Mine and the Greymouth Coalfield is shown in Fig.1
The Strongman 2 Mine is situated north of Greymouth on the West Coast of the South Island. It produces approximately 360,000 tonnes per annum of low ash (5%), Bituminous high volatile B rank coal (Ro max 0.72, Sulphur 0.25%).

To date coal has been sold to Japan and South America as thermal coal. There is now a strong demand for the coal as semi-soft coking coal in Japan. Approximately 5% is sold as graded domestic product and a further 10% to the cement industry.

The exported coal is railed to the port of Lyttelton on the east coast of the South Island (220km) and shipped in Panamax size vessels. Very high internal freight costs, a high exchange rate and falling thermal prices have put pressure on the operation. The operation has become the lowest cost underground producer in New Zealand, while also maintaining the highest extraction ratio of any underground mine.

The operation utilises hydraulic coal cutting at the face for pillar extraction and hydraulic transport on the seam floor and in flumes to remove the coal to the surface. All the present workings are to the rise of the surface access allowing coal and water to flow from the mine to a surface dewatering plant. The water is then recirculated via a high pressure pump to the coal face. Development of the mine is by mechanical means with transport to the surface by flume using low pressure water.

Resources in the block presently being mined amount to 4 million tonnes with indicated reserves of 20 million tonnes on the mining licence. The geology of the mining area is typified by a very wide range of seam gradients up to 70°, a number of north-south trending normal faults, and areas of very low surface cover, with outcrops to the north. Seam thickness also varies from less than 2m to 15m.
The mine is accessed via a 4km, 1 in 8 grade road from the coastal highway. The topography is extremely rugged and the elevation of the portals is 412m above sea level. The Strongman 2 Mine plan is shown in Fig.2. A typical seam cross section is shown in Fig.3, showing shallow cover, steep grades and sublevels driven for extraction.

Fig. 2 – Strongman 2 Mine – underground mine plan

Fig. 3 – Strongman 2 Mine – Cross-section: E Seam
DEVELOPMENT OPERATIONS

Underground development at Strongman 2 began in 1994 and has included these components.

1. Initial stone drivage to access the coal of three 50m long drifts using an Anderson RH25 Roadheader loading into an Eimco 913 LHD. The 24 tonne RH25 was stretched to its limit cutting the 50 MPa rock, and a 35 tonne Mitsui S125 has proved more successful in later stone drivage.

2. Coal drivage to the rise using a Mitsui S125 Roadheader and 4.0m³/min of low pressure water to transport the coal. Grades of up to 1 in 3 were traversed and, with good roof conditions, advance rates of 23m were achieved in an eight hour shift. The steep grades and washed-out roadway behind the roadheader made material transport to the face by diesel equipment difficult.

3. Considerable roadway development has also been carried out utilising the high pressure monitor water to cut the face. This has been in very steep seam conditions unsuitable for mechanical equipment. The narrow roadways require minimal support but materials and supplies must be manhandled. This development was limited to the face area or sublevel roadways.

The present method of coal drivage is a 12CM6 Joy Continuous Miner loading into a 15SC Joy Shuttle car. The coal is trammed up to 200m and discharged into the coal and water flow from the monitor extraction area. This method has two major advantages, short dip development is possible and diesel material vehicles can transport equipment direct to the face, as the roadways are not washed out.

All roadway development is supported on 1.8m fully resin encapsulated roof bolts, four bolts to a row and rows 1.5m apart. Roof conditions are generally good with a hard sandstone immediate roof. Steel mesh is used with bolts in friable coal, or in stone drivage. Cable bolts, 4m and 6m in length are being trialled to improve intersection stability as some slabbing of the stone roof occurs. In the thicker seam areas (10m to 15m) development is carried out near the floor to facilitate extraction of top coal with the monitor. Coal ribs stand extremely well and do not require support.

Typical panel lay outs are shown for sublevel and subrise extraction in Fig.4 and Fig.5. Subrise is used for the flat grades and sublevel extraction for steep grades.

![Diagram of Coal Extraction by Monitor](image-url)

**Fig. 4 – Strongman 2 Mine – underground mine plan (sub-level monitor extraction)**
The majority of the coal at Strongman 2 is worked using a hydraulic monitor. The coal is cut at the face using a remote control hydraulically activated face monitor. The water and coal flows from the workings, (which are to the rise of the access portals) to the surface dewatering plant. The water is cleaned to approximately 50ppm suspended solids and pumped back through the monitor pump to the face. Cutting pressures of 1.33 MPa and water flows of 5m³/min are utilised.

The process is relatively continuous with the monitor usually cutting for approximately 75% of the shift.

The monitor system was chosen for the following reasons:

1. The very steep dipping seams, up to 70°;
2. The thick nature of the seams, up to 15m;
3. The hard nature of the roof and floor making hydraulic cutting and transport viable;
4. The inherent safety of the system, in that no piece of equipment or person is exposed out in the goaf area;
5. Previous experience with hydraulic transport of coal; and
6. Lower capital and maintenance costs of the coal winning and transport system.

The following are problems and issues that emerged once the system was operational.
1. Major surges in flow due to underground blockages were resolved by the use of steel flumes in flatter areas and increasing the capacity of the dewatering plant to handle surges in flow.

2. There was very little weight in the extraction area to assist water cutting, resulting in less than planned cutting rates. Depth of cover is now 90m and increasing.

3. Intersection of unexpected faults requiring stone drivage and modification of mine plans.

**MONITOR PRODUCTIVITY**

**Data**

The Monitor system productivity is dependent on three factors: system availability, utilisation and cutting rates. All operating data is analysed to enable improvement in the method of operation. An example of this data is included in the Appendix.

**System availability**

The key factors that affect the monitor system availability are

1. Six monthly overhauls of the monitor pump rotating assembly, resulting in two weeks downtime;
2. Contaminated water in the recirculation circuit resulting in monitor pump shut down;
3. Leaks in the high pressure face lines, requiring shut down and repair; and
4. Face units damaged or buried after pillar falls.

**System utilisation**

Key items that influence utilisation of the system are

1. Moving the face units back after pillar falls. (Alternative units are set up to enable alternative places to be worked.), and
2. Inspecting the face during the cutting process, which requires monitor shutdown.

**Cutting rates**

The production variable that generally has the greatest impact on productivity is the monitor cutting rate. This is measured as tonnes per minute that the jet of water will cut from the face. This ranges from 0.5 tonnes/min to 1.5 tonnes/min depending on

1. coal hardness;
2. roof weight;
3. cleat direction;
4. cutting distances;
5. pump output pressure; and
6. pump flow.
Coal hardness testing has been carried out utilising the Protodyadanov standard and a good picture of varying hardness throughout the present mine workings has been developed. A plan showing coal hardness contours is included in Fig. 6.

Maximum cutting distances have been reduced from 25m to 15m to improve cutting rates.

Pump output pressures and flow are continuously monitored to determine any dropoff in performance that will affect cutting rates.

---

**Fig. 6 – Strongman 2 Mine – Hardness contours (Protoyakanov)**

**HUMAN RESOURCES**

The Strongman 2 Mine operates in production mode 24 hours per day, seven days per week, with a short break at Christmas.

All production and maintenance staff work twelve hour shifts, face to face on a three day on, three day off roster system. A total workforce of 85 is involved in mining and loading out operations.

Production and maintenance staff are employed on identical employment contracts; they share equally in a production bonus, and they will carry out either work, depending on their skill levels.

Key points in the employment contract are
1. Hours of work, 12 hours per day face to face;
2. A three on three off, Nightshift, Dayshift roster;
3. Overtime paid for work on rostered days off;
4. Average total gross earnings approximately $46,000 per year;
5. A production bonus, the rate varying with output per man shift;
6. A redundancy agreement;
7. Six paid union meetings per year; and
8. A two year contract period with a cost of living (CPI) increase after year one.

Since the mine started in 1994, two days have been lost due to industrial stoppage.

The key to this has been

1. A realisation by the workforce of the economic necessity of competing in the international marketplace;
2. The fact that the mine is divided into small shift crews, 10 to 12 workers, has generated a small mine mentality where people feel part of the operation;
3. The Employment Contracts Act, which prohibits strike action outside normal contract negotiations; and
4. A small and flat management structure, committed to open communication with the workforce.

The staff structure is shown in Fig.7
HEALTH AND SAFETY ISSUES

The key factors affecting health and safety on the Strongman 2 site are as follows:

1. The site works under the Health and Safety in Employment Act 1992, following the repeal of the Coal Mining Act and Regulations. This is based on hazard identification and management, compared to the more prescriptive regulations. The New Zealand Government plans to introduce revised mining regulations some time in the future.

2. Although no methane has been detected in the present working, it is expected as depth of cover increases. The mine is therefore worked as a "Gassy Mine" with all flameproof equipment and operating procedures.

3. The coal is prone to spontaneous combustion and this is aggravated by major breaks to the surface. The action of the monitor in washing all small coal from the goaf and achieving a very high recovery helps off set this propensity to spontaneous combustion. One small fire is active in the goaf, remote from the working faces and is evident from smoke emitting from some surface cracks. The gases emitted are monitored using a surface Maihak tube.

4. Due to the steep seam gradients, access is often limited for men and materials vehicles, requiring manual handling of heavy pipes and equipment. The majority of miners accidents are the result of heavy lifting or slipping on steep gradients.

5. The monitor system is inherently safe due to the ability to cut pillar coal from a distance of up to 25m, allowing the monitor to be kept in a secure position and the operators are situated at least 10m further back.

6. The introduction of high pressure monitor water to the coal face poses a significant hazard, and the operation has to be controlled with strict procedures.

    The lost time accident frequency rate per 100,000 man hours is 6.9, the most serious accident to date being a badly crushed finger.

8. The key elements of safety management on the site are:

    a) a hazard identification and management system built on inspections and total worker involvement in site safety meetings;

    b) rigorous investigation of all incidents and accidents and follow up to ensure remedial action has been taken;

    c) thorough induction, ongoing training and licensing of operators; and

    d) the development of a culture that says accidents are not an expected outcome of mining operations.

THE FUTURE OF HYDRAULIC EXTRACTION

Due to the small resource available at the Strongman 2 Mine, it is expected that production will remain at approximately 360,000 tonnes per annum, to extract the remaining reserves in ten years.

Solid Energy New Zealand Ltd is part of a joint venture known as Greymouth Coal Ltd, which is in the process of establishing a large operation, potentially over one million tonnes per annum, to extract a thermal type coal from the Greymouth Coalfield based on monitor extraction.
Solid Energy is also developing the Mt Davy Mine, approximately 5km from Strongman 2. This will extract a high grade coking coal, low in sulphur, for blending with Buller coals (Fig 1). At a depth of 700m, this mine will certainly be the deepest ever developed in New Zealand. The main seam has been intersected by the first 1.1km drive. Work on strata control and managing gas and potential outbursts is being carried out; one outburst has already occurred. The monitor system will be ideal for the extraction of this coal due to the soft nature of the coal and expected high roof pressures. The mine is planned to produce 500,000 tonnes per annum.

**CONCLUSION**

Although this system of extraction is not used elsewhere in Australasia, it has been proven to be ideal for some of New Zealand's most geologically disturbed coal seams. The small deposits of thick low ash coal have benefited from a system that results in very high seam recovery, exceeding 75%. The low capital cost inherent in the monitor system has allowed the set up of operations in these small blocks of coal and the resultant low operating costs have allowed the operation to compete internationally.

The challenges for the future revolve around improving the availability of the high pressure pumping systems, the improvement in cutting effectiveness and the accessing of these often deep and heavily faulted blocks of coal.

**APPENDIX**

**STRONGMAN 2 MINE MAJOR PLANT – OPERATING DATA EXAMPLE**

| 1 | Ebara Monitor Feed Pump | 5 stage centrifugal pump |
|   |                         | 5200 R.P.M 1:3.5 Ebara gearbox |
|   |                         | 1550kw Toshiba induction motor |
|   |                         | 5m³/min discharge quantity |
|   |                         | 187 kgf/cm² max pressure |
| 5 | Mitsui Monitor Face Units |
| 1 | Mitsui S125 Roadheader   | Weight - 30 tonnes |
|   |                         | Cutter motor - 125kw |
| 1 | 12CM6 Joy Continuous Miner |
| 1 | 15SC Joy Shuttle Car     |
| 2 | Eimco 913 LHD            |
| 2 | ERIEZ H.D.S Primary Dewatering Screens | 1.4mm Aperture |
| 4 | CMI 10 inch Cyclones     |
| 1 | Eriez Intensive Dewatering Screen | 0.4mm Aperture |
| 1 | High Rate Superflow Clarifier | 8m diameter |
| 1 | Maihak Tube Bundle System |
INTRODUCTION

Longwall mining has been the dominant global coal mining method for decades. However, not until 1994 did longwall mining surpass continuous miner room and pillar extraction tonnage in the United States. This significant event, occurring in the most cost driven market in the world, confirms the efficiency and effectiveness of longwall mining.

Free market economics, with respect to energy costs, is replacing government controlled energy policies, siege mentality and early 20th century isolationist policies. Coal, as an energy source, is challenged by competing fuels and technologies. The twin threat of price and environmental bias will continue to pressure the industry. Hopefully coal’s abundance, reliability and low cost can off-set the eventual increase in compliance costs.

A recent Coal Age global longwall census indicated there are 1,712 fully mechanized longwalls operating in the 12 major underground coal producing countries and that the number of mechanized longwalls is increasing in China, Australia, South Africa and India. It is clear that longwall mining methods will continue to produce coal safely, at low cost and in large volume.

With the acquisition of Dobson Park by Harnischfeger Industries Inc. and the merging of Longwall International into Joy Mining Machinery, Joy now manufactures all the major elements of a modern longwall system. The challenging task of integrating roof supports, armored face conveyor (AFC) and shearer into a single operational system fell largely to the operating coal companies. Their ability to optimize system performance against a background of competing commercial imperatives as well as technological incompatibilities, was and continues to be difficult. A step change in reliability and productivity is seen at hand by Joy through the supply of complete longwall systems from Joy Mining Machinery.

However, the changing relationship between coal mine operator and equipment supplier will be as important as improvements in the equipment features and system integration.

There has been a continuous process of rationalization in the coal mining industry affecting both mining companies and machinery manufacturers. In the USA, the number of underground mines has fallen from 2,008 in 1981 to 885 in 1996. There were 946 underground coal mines in the UK in 1945 and this number now stands at 25. The competitive pressure has resulted in the emergence of a few big players and the loss of many small independents.

On the longwall equipment supply side there are only two major 'high-tech' suppliers of roof supports and face conveyors four suppliers of shearers. This rationalization is the result of the contraction in the coal mining industries in the United Kingdom and Germany, and the diminished ability of these markets to support multiple suppliers. In addition, today's equipment produces more coal per unit of production which reduces the number of opportunities and therefore the available market for suppliers. This industry rationalization in both coal production and equipment supply suggests that alliances between machinery manufacturers and coal producers are the way of the future.

BACKGROUND

Joy Mining Machinery was founded by Joseph Joy in 1919 and over a period of years became the major coal mining machinery supplier in the United States. Until the introduction of longwall in the 1960s the principal mining technique was

1 Vice President and General Manager, of NSW, Joy Mining Machinery
2 Manager Marketing and Sales, Joy Mining Machinery
room and pillar extraction. Equipment suppliers sought to mechanize the individual operations; drilling, blasting, loading and conveying. Joy successfully met this challenge with equipment colloquially referred to as a “conventional face”. This consisted of a drilling unit, undercutter, loader and shuttle cars. The mechanization process continued with the introduction of the continuous miner, so called because the drilling, undercutting, blasting and loading operations could be done by a single machine. Although longwall was introduced in the 1960s room and pillar was still the predominant method until the 1990s. The productivity and competition from room and pillar accelerated many of the advances in longwall mining. Joy entered the longwall market, as a manufacturer, in 1975 with the introduction of the ILS shearer. This was the first multi-motor, all-electric, shearing machine and broke with the traditional single motor, hydraulic designs of the day. By the late 1980s Joy had become the major supplier of shearsers in the US market.

In 1995, Joy merged Longwall International into its organization with the acquisition of Dobson Park, a United Kingdom conglomerate. Longwall International was itself the product of the long rationalization process of the UK coal machinery supplier industry. The latest merger before the Joy acquisition was between the globally-recognized manufacturers Meco International and Gullick Dobson. Joy is the only manufacturer of all the major elements of the longwall system (see Photo 1) which includes shearsers, roof supports, face conveyors, stage-loaders, crushers and pump stations, and is now the largest supplier of underground coal mining equipment in the world.

**INSTALLED BASE**

A Coal Age census (September, 1997) of the top 12 longwall producing countries is presented in Table 1 - Major Longwall Producers (1996)

<table>
<thead>
<tr>
<th>Country</th>
<th>Number of Longwalls*</th>
<th>Average Production per Longwall per Shift (tonnes)</th>
<th>Average Production per Longwall per Year (000) tonnes</th>
</tr>
</thead>
<tbody>
<tr>
<td>USA</td>
<td>69</td>
<td>3,475</td>
<td>2,502</td>
</tr>
<tr>
<td>Australia</td>
<td>30</td>
<td>2,360</td>
<td>1,558</td>
</tr>
<tr>
<td>United Kingdom</td>
<td>36</td>
<td>1,667</td>
<td>1,157</td>
</tr>
<tr>
<td>China</td>
<td>244</td>
<td>1,511</td>
<td>1,070</td>
</tr>
<tr>
<td>Canada</td>
<td>2</td>
<td>1,499</td>
<td>1,138</td>
</tr>
<tr>
<td>Germany</td>
<td>66</td>
<td>1,423</td>
<td>966</td>
</tr>
<tr>
<td>South Africa</td>
<td>8</td>
<td>1,236</td>
<td>1,020</td>
</tr>
<tr>
<td>Poland</td>
<td>350</td>
<td>1,190</td>
<td>744</td>
</tr>
<tr>
<td>Russia</td>
<td>432</td>
<td>696</td>
<td>418</td>
</tr>
<tr>
<td>India</td>
<td>6</td>
<td>686</td>
<td>371</td>
</tr>
<tr>
<td>Kazakhstan</td>
<td>40</td>
<td>634</td>
<td>475</td>
</tr>
<tr>
<td>Ukraine</td>
<td>429</td>
<td>520</td>
<td>312</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>1712</strong></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

* Only fully mechanized faces included.

Joy has been successful selling its equipment to customers that are market driven. These operators must produce coal at the lowest cost over the life cycle of the equipment in order to survive.

Installed base (see Table 2), as defined by Joy, refers to equipment in use at the time of a particular census. This figure is lower than machines in service which includes units under rebuild or idled, spare capacity. Installed base varies from month to month as a result of new faces starting up and old panels being worked out.
Table 2 - Installed Base

<table>
<thead>
<tr>
<th>Country</th>
<th>Number of Longwalls</th>
<th>Number of Shearers</th>
<th>Number of Joy Roof Supports</th>
<th>Number of Joy AFCs</th>
</tr>
</thead>
<tbody>
<tr>
<td>USA</td>
<td>65</td>
<td>52</td>
<td>35</td>
<td>31</td>
</tr>
<tr>
<td>Australia</td>
<td>30</td>
<td>8</td>
<td>28</td>
<td>22</td>
</tr>
<tr>
<td>UK</td>
<td>33</td>
<td>14</td>
<td>33</td>
<td>26</td>
</tr>
<tr>
<td>South Africa</td>
<td>8</td>
<td>3</td>
<td>7</td>
<td>7</td>
</tr>
<tr>
<td>Total</td>
<td>136</td>
<td>77</td>
<td>103</td>
<td>86</td>
</tr>
</tbody>
</table>

PERFORMANCE

The achievements of the world-class performers is presented below:

Upper big branch - performance coal

The achievements at A T Massey's Performance Coal Company in the USA are significant. This is A T Massey's first longwall operation that within less than three years achieved world class performance. At the Upper Big Branch Mine a complete Joy installation averages 620,000 t/month, with a best month performance of 640,000 tonnes. The 305m wide face is typically 1.83m thick and retreated 40.6m in one 24 hour period during April, 1997. The face equipment includes a JOY 4LS9 high voltage shearer with 725kW total installed power. The 175 shields constituting the set of roof supports each 1.75m wide with twin 800 tonne leg capacity. The JOY AFC is powered by three 522kW drives.

In addition, the advance achieved with JOY 14CM miners and 10SC32 shuttle cars in the “super section” longwall gate development remains well above industry average. The three entry super section has two miners and three shuttle cars operated by a single crew. The system is designed to eliminate time lost between cuts. When a cut is complete the mining crew leaves the continuous miner and moves to the second miner situated in another roadway. The work place is bolted and made ready for production. Three road gate advance averages 83m/shift and has reached a best of 152m/shift.

West elk - mountain coal company

In Colorado, the West Elk mine, operated by Arco's subsidiary Mountain Coal Company, achieved excellent shift and monthly production using a JOY 6LS3 shearer. Mining a 3.3m to 3.96m seam on a 290m face, they have produced 796,000 clean tonnes in a single month. The single face mine with a workforce of 280, has an annual production target in excess of 6 million tonnes.

Mountaineer mine - Mingo Logan

More recently in the eastern USA, Mingo Logan’s Mountaineer mine produced 969,911 raw tonnes in a month. This was achieved in a 1.65/1.8m seam where, during the month the face retreated 730m, with a daily best of 38.1m. The face equipment consisted of a JOY 4LS9 operating on a JOY conveyor with Ultratrac haulage. The supports were fitted with the RS20 roof support control system, and the face availability was recorded at 96.7%.

FMC Corporation

In Wyoming, notable production feats are being achieved through the use of longwall mining systems in non-coal applications. A longwall system consisting of JOY roof supports, AFC and shearer has been installed at FMC Corporation's trona mine in southern Wyoming and the 3.65m seam of hard (48.3MPa) mineral is being mined with a JOY 6LS6 shearer (See Photo2). The 190.5m face is using a set of JOY, 2X680 tonne shields, has a best month’s production of...
236,365 tonnes with a best shift of 6,365 tonnes. The use of longwall mining in the trona basin is increasing with Solvay’s recent decision to purchase a JOY roof supports and 6LS6 shearer.

**South Bulga - Cyprus**

Australia is the largest growth market for world class longwall systems. At Cyprus’s South Bulga mine, single month mine production of 493,453 tonnes has been achieved from a 200m face in the 2.4m Whybrow seam. JOY 940 tonne 2 leg shields with LI10 controls complement the JOY AFC, which is equipped with a twin 42mm chain. The 1000 mm wide AFC runs at 1.6m/sec and is powered by 2x750kW drives through 5665 Voith 1000BP couplings. South Bulga has achieved a 24,793 tonnes/employee-year, 2.5 times the Australian average.

**Matla Colliery - Ingwe**

In South Africa, the longwall face in 4 Seam at Ingwe’s Matla Colliery is equipped with a JOY 6LS5 shearer. The shearer set a South African production record of 343,649 tonnes in a single month in September 1997. The previous South African record was held by a JOY 4LS shearer.

**Daliuta - Shenhua**

High production equipment has been supplied to China. A JOY 6LS3 on a JOY AFC operating in a 4.0m seam at Shenhua’s Daliuta mine achieved a Chinese record by mining 14,907 tonnes in a two-shift period. The tough Jurassic coal in this shallow deposit is difficult to cut and a second Joy shearer and AFC combination, with more power, capable of extracting 5.0m has been installed at the neighboring Bulianta mine. With 610kW cutter motors and a total installed power of 1500kW the JOY 6LS5 is one of the most powerful shearers in the world.

China is the world’s leading producer of coal, mining about 1.4 billion tonnes raw coal a year. The industry can be divided into State owned and operated mines and a non State sector operated by a variety of entities (i.e. provincial governments, counties, towns, collectives and individuals). State owned mines produce about 40% of the country’s total output and account for nearly all the investment and mechanization. The non-State sector mines are smaller and less mechanized. There are new laws in place designed to limit the growth of this sector and to stop mining activities where they are dangerous or sterilize the reserve base. The problem for the authorities is that this non-mechanized, lower invested segment is producing the lowest cost coal.

In China 70 - 75% of electric generation is produced by coal fired utilities. This share of electricity production will continue for the next twenty years despite Kyoto initiatives and the Three Gorges hydro-electric project.

**CURRENT TECHNOLOGIES**

Longwall Shearers

The most significant development in the design of shearers was the introduction of the JOY multi-motor all-electric shearer in 1975. This innovation brought a new level of reliability and maintainability to the machine. The advantages of the modular, all-electric concept and the benefit of eliminating complex hydraulic circuits from the underground environment were quickly recognized. Since that introduction, advanced shearer manufacturers have adopted the concept.

Joy’s modular design consists of five main structural elements (see Photo3). The body of the shearer normally consists of three high tensile steel fabrications bolted together to form a slim main section with no under-frame. This design provides maximum under-body clearance for coal passage in a given seam thickness. The elimination of the under-frame also enhances underground transportation. The Joy design eliminated face-side access to the electrical controller section which means that normal maintenance can be carried out in a safe working environment. Two traction sections are bolted and doweled to each end of the controller case. Each traction section houses a haulage motor, primary traction gear case and
hydraulic system for moving the ranging arm and rotating the cowl. The controller case, which forms the center section, contains the electric control system including, vacuum contactors, transformers, SCR bridge or AC drive, microprocessors, control circuitry and data display screen.

The secondary traction gearcases (or down-drives) are bolted to the traction cases in an arrangement which permits the custom fitting of the shearer within the AFC and roof support envelope. A wide selection of down-drives can be fitted to the shearer to suit individual haulage preference.

High tensile steel ranging arm castings house the cutter motor and cutter gearcase. Joy manufactures its own gears using high performance, alloy steel. Custom design, coupled with exacting heat treatment processes, has made 6 million tonne design life a reality.

In the 1920s Joy developed a relationship with Reliance Electric to maintain the latest electric motor technology. This ongoing relationship has enabled Joy to combine the motor specific skills of Reliance with Joy's own application and design expertise. The incorporation of specially formulated varnishes and specific baking techniques have produced motors that are suited to the rugged mine conditions and machine duties.

The main driver of design development over the past twenty years has been the requirement for higher installed power (see Figure 1). Installed power has more than doubled in the last 15 years and high voltage (2,300V/4160V -60Hz, 3,300V/50Hz) electrical supplies are essential in the top producing faces. Total installed capacity on the JOY 6LS5 has reached 1610kW.

Recently there has been a trend in the application of AC haulage drives with on-board variable frequency controls. The JOY 7LS, scheduled for start-up in mid- March 1998 at Canyon Fuel's Sufco mine, and the 6LS5 for Moranbah are equipped with this feature. These AC drives are used for applications where there is a need for high cutting and flitting speeds. High speeds typically are required in narrow web operations or where uni-di cutting on long face widths is practiced. Uni-di refers to cutting in a single direction and then flitting back the entire length of the face before cutting the next pass.

Roof supports

Roof supports for high capacity longwalls are custom designed to meet exacting cycle requirements (up to 50,000 cycles) and operating requirements (over 1,200 tonne yield). Finite element analysis along with prototype static and dynamic load testing are essential in the design processes. Joy has introduced the design failure mode and effects analysis (DFMEA) process. This is a design analysis methodology that looks at possible failure mode and weights the severity and the probability of occurrence. These factors are combined to define an index for each component or sub assembly of the equipment. This approach is extremely beneficial in the prioritization of design efforts and resources.

Today's heavy duty supports are twin-leg design (see Photo 4), the diameters of which have increased to 450mm. Joy's first 450mm leg face is to be delivered in June 1998 to Moranbah North, a new mine, in Australia. Large leg diameters require that shields must be positioned on 1.75m support centers to accommodate the leg pockets. The growing demand for +1000 mm web cutting increases canopy length and this increases support rating to maintain densities. Longer, wider canopy dimensions add weight to each shield heavier which makes installation and face move logistics more difficult. Thick seam supports now typically weigh more than 25 tonnes. However, an advantage of wider supports is a reduction in the number of shields per unit face length and this increases the overall system reliability.

The hydraulic legs or jacks used to set the canopy to the roof generally are double telescopic to provide maximum open to closed height ratio. Joy legs are back extruded which enables large diameter, in-casing drillings for fluid flow. The elimination of welded stack pipes increases reliability and fatigue life, and the large diameter drillings improve flow and leg cycle times. The extruded design also reduces the leg constant for a given stroke and enables the leg to be positioned closer to vertical which improves performance. Other features include a variety of base lift arrangements to free the base from soft floor conditions and reverse force advance ram and relay bars for specific applications.

All the hydraulic functions of the JOY roof support are actuated through the Compak Valve, which consists of three sections; the solenoid manifold, main valve block and return manifold. The solenoid valves are the link between the
electronic control system and the function linked spool valves. The major advantage of the design is its ability to operate at pressures greater than 31MPa on a supply of 6 to 15 volts. This allows the solenoid to be activated on demand unlike other systems which require the hydraulic feed to be interrupted by a control solenoid. This improves cycle times and the ability to control multi-function activities. The valve block comprises up to 14 spool valves which provide a greater degree of reliability than the more contamination sensitive ball and seat designs.

**Armored face conveyors**

Armored Face Conveyors (AFC) serve a dual purpose; to convey coal and to provide a track on which the shearer can operate. As production requirements have grown and face lengths have increased, so too have AFC chain dimensions. Current high production AFCs (see Photo 5) utilize twin 42mm chain in 1000mm wide (raceway) pans with total drives rated at 1500kW. Such conveyors are designed to convey 4,500 tonnes per hour on faces up to 300m long. AFCs of these capacities require high voltage electrical supply (2300/4160V-60Hz and 3300V-50Hz).

Full load starting power requirements, on these high production faces, are considerable. To meet this challenge, Joy developed the JOY TTT (Turbo Transmission Technology) system, a microprocessor controlled fluid coupling which effectively provides a progressive start-up load (see Photo 6). The closely monitored flows and temperatures enable the unit to provide enhanced control and refined water management. In addition the system brings soft-start benefits of longer chain, sprocket and gearbox life, as well as reduced pan wear.

The off-load starting and load sharing characteristics of the system bring several benefits, including reduced electrical demand at start-up, the ability to use non matched motors, and reduced motor specification with associated cost savings. The double 562 turbo coupling increases and limits torque, improves thermal characteristics and is contained in a smaller envelop. The hydro-dynamic features are a function of the fluid circuit design and the precisely controlled fluid levels. These result in proactive load sharing and a controlled soft start.

The JOY AFC design uses cast steel sigma sections of exacting tolerances to eliminate the need to weld clevis and mating arrangements, and the high specification abrasion resistant upper deck plates enhance pan life and maintenance free fabrications.

The AFC-shearer mechanical interface through the haulage system is an important factor in optimizing operations. A variety of haulage systems have been developed to cope with greater haulage forces. Joy’s latest, Ultratrac 2000, the latest version of the proven Ultratrac haulage (+ 50 installations), is a forged rack and heavy duty sprocket system, with increased sprocket and trapping shoe life, capable of negotiating severe seam undulations.

Heavy duty AFCs utilize a single or twin, in-board chain configuration depending on capacity requirements and chain diameter. The sprocket designs and manufacture are a significant product differentiation with special consideration required for pocket design, machining and flame hardening. An important factor in achieving good chain life is an effective chain tensioning system. Joy offers two automatic chain tensioning devices. One mechanism relies on conveyor-chain slack in catenary to activate the tensioning rams, the other operates via load transducers on the rams which indicate variations in chain loading and cause the tensioning rams to extend, returning the chain to the nominal settings.

**Electronics**

The introduction of multi-microprocessor-based, embedded control systems on the JOY 1LS4 in 1980 was an important advance in shearer control technology. Since then, the increasing power of these microprocessors has enhanced their capacity and functionality. In addition to enabling cybernetic operation, these on-board control processing units (CPU) provide a variety of diagnostic features.

Joy is devoting an increasing amount of R&D investment to system integration through electronic standardization and advances in remote control techniques are enabling a cable-free environment. Cables in the underground environment are vulnerable to damage and the removal of these cables has increased system reliability. In the design of underground mining equipment, 'space saving' technologies also bring several benefits. Miniaturization has allowed smaller component design and these new, smaller components (e.g. valve banks and electro-hydraulic controls) generally can be placed in less vulnerable locations. These safer locations mean higher reliability and give operators ergonomic work environments.
resulting in greater safety and productivity. A second advantage of miniaturization is the ability to create space within a given design envelope to add new features. It also may be possible to make the case smaller and bring performance enhancements through redesign (e.g. thinner shearer body to permit more coal clearance).

There have been many technological innovations in shearer electronics, including optical encoders, serial data communication (which eliminates hard wiring in the controller case), face conveyor feedback loop, memory cut and the JNA control system. The JNA control system enables operating parameters to be changed using specific access codes. A shift supervisor can change operational inputs while an engineering manager has access to overload parameters and circuit timing. Using the JNA memory cut, a complete cut sequence including gate-end cuts can be programmed into an operator friendly system. This produces cleaner coal, improved machine operating efficiency, higher levels of automation and better roof and floor control. Cutting speeds, drum and cowl positions and undercut depth can be reconfigured via a menu roll-down option or this can be programmed into the JNA. This technology is being refined for those operators who want to view all the face data on surface in real time. Shearer data can be transmitted to the head-gate via either a pilot core or the power cores and from there to surface by telephone line.

A common specification today is shearer initiated support advance (SISA) which can be supplied in either infra-red or digital/serial data communication systems. The market trend is toward the digital/serial system since this also provides a means to identify shearer position on the face and is not a function of clear line-of-sight constraints. Both systems operate by sending a signal from the shearer to the roof supports to activate a selected shield sequence. Joy also has made progress in shearer steering technologies particularly in supplying a reliable ‘memory cut’ system. Joy’s first successful application of this having been on a 4LS shearer at R.J. Budge’s Wistow Mine in the Selby Complex. With it, the shearer operator creates an initial profile or template under manual control, and the machine then automatically replicates the profile on subsequent cuts until conditions change. The operator then updates the profile by manually cutting a new pass.

In roof support electronics, the JOY RS20 (see Photo 7) represents state-of-the-art, electro-hydraulic, control systems. In electro-hydraulic controls, as with most mining equipment features, a balance must be struck between proven technology and innovative design. The RS20 control system meets this criterion in the use of field tested microprocessors and software developed by mining experts. The flexible architecture used in the design of the RS20 also ensures expansion capability to meet future requirements. The benefits of the electro-hydraulic system are that the operators can remain in dust free areas while the system can operate close to the shearer which improves roof control.

The RS20 system consists of a set (one for each support) of intrinsically safe (IS) shield interface modules (SIM) linked in series and connected to the shield control center (SCC). The SIM has buttons to operate all the functions on that support as well as the support on either side of it. In addition it is possible to use the SIM as a support fault diagnostic aid for the support and the complete system.

The SCC is housed in an explosion-proof enclosure and provides the central automatic control facility and operator interface for the roof support control system. It consists of the sequence or control computer, color display, keyboard interface and intelligent power supply interface.

Integrated Longwall systems - the future

Shearers will continue to increase in power but the rate of increase will be slower than has been experienced in recent years. Haulage systems will trend to on-board variable frequency, variable voltage, AC drives. Electronic systems will continue to be developed to enhance automation. There have been several attempts to introduce self-correcting shearer steering systems. These systems rely on a sensor that detects the coal/rock interface. Two main types of sensors have been used in the industry. The most common system is based on natural gamma radiation detection and measures the difference in gamma radiation characteristics between coal and rock strata. These readings are calibrated to give coal thickness to the interface. One drawback of this technology is the loss of data or spurious readings in areas of broken roof conditions. In addition the technology does not function when a thick coal roof must be left or the overlying rock does not emit radiation (e.g. sandstone).
A second technology used to automate horizon control depends on either a sensitized pick or machine vibration analysis. This system can produce confusing readings in conditions where floor and coal are of almost equal hardness or where rock bands are present in the coal. Future steering solutions are likely to utilize a new enabling technology and eliminate operating anomalies through smart software.

A significant improvement in longwall performance will be achieved as a result of mechanically and electronically integrating the longwall face. If optimization of current equipment, as a system is to be attained, common protocol electronic communication between all the elements of the system will be necessary. That is, if the system bottle neck is the panel belt then the stage loader must be able to slow the AFC to match this constriction. At the same time, the AFC must be able to communicate with the shearer to control its cutting speed to match this rate and also to compensate for the shearer’s position on the face to eliminate surges. Communication between the AFC and the roof supports is necessary to achieve automatic face advance. Communication between the AFC and the roof supports must be refined to enable AFC steering. A continuous two-way communication between all the longwall equipment must be established and maintained, and an optimization logic must permit the face to produce the optimal tonnes under changing conditions. This type of three way dialogue might permit operational changes and allow the use of a sumping shearer, which eliminates the time lost at the gate ends.

Electronic control and automation of the coal face are the key to future advances and step changes in longwall mining productivity and performance. The achievement of 100% reliability through the life of the panel must be the goal.

When that target is reached through a combination of new technology, innovation and the use of redundant systems, the next step will be to fully automate the longwall face can be attacked.

In the UK automated faces have been sought since the early 1960s. These efforts usually have been undertaken with the aim of removing people from the face either as part of a safety/environmental directive or to meet economic targets. Both goals are valid; though, both miss the major advantage in attaining a truly automated face.

When a longwall face is made 100% reliable and automated, people will be required on the face for face installation and limited work between shifts. Eventually it will be possible to eliminate the roof supports walkway and this will radically reduce the leg-to-tip distance and give significant improvements in roof support design. The shields will be smaller and lighter, and the legs can be positioned closer to the vertical for better mechanical effect. The reduction in size will bring improved performance at reduced cost, shorten face installation time, and reduce face-move transport issues.

AFCs with capacities up to 7,000 tonnes per hour will be needed for longwalls up to 400m long. This will require individual transmissions of 1500kW. Although Joy’s current soft-start technology can handle this power, new gear box designs will be required. In time, the development of variable speed, variable voltage, AC drives will eliminate couplings, and allow AFC speed to vary with the shearer and production needs.

AFC improvements can be realized with fundamental changes in materials and conveyor design. At present, coal is not conveyed by the AFC (in the manner of a belt conveyor) but pulled and pushed along the AFC deck plate by steel flights bolted to ever larger combinations of single and twin strand chain. Friction between the flights and the deck plates, particularly at the pan joints, requires significant power to overcome the resistance and to move the chain and flights. The use of materials such as nylon and carbon fiber composites has resulted in greater strength and lightness in other industries and could present an opportunity for mining equipment designers. In addition, materials with sufficient strength and lower coefficients of friction need to be considered. A rethink of the task at hand and how it might be accomplished by carrying the coal on a strong, lightweight, flexible, sprocket-driven mat within the AFC race could reduce power requirements. Research is required to find ways to capture the AFC energy wasted when no coal is being conveyed.

In the United States, environmental pressures will place a premium on the extraction of low sulfur coal primarily found in low seam deposits. Technology must be developed which will allow these thin seams (1.0 - 1.2m) to be mined economically. Obviously this will mandate a fully automated face.

The ultimate technological key to improved longwall performance will be the development of an integrated electronic system which will control all the elements of the longwall face in a self optimizing mode. Such a system will be reliable to aero-space standards and require zero maintenance. The face will be 100% automated requiring no workforce on the face.
during cutting, and all functions will be controlled and monitored from the gate-end, surface, or any location selected by the user.

CONCLUSIONS

Longwall mining will be the dominant underground coal mining method worldwide for the foreseeable future, and productivity improvements will continue to be realized. These are likely to be the result of technological improvements in the design of the electro-hydraulic and mechanical features of the various elements making up the longwall system. These design improvements will focus on reliability and capacity. System improvements gained from the electronic and mechanical integration of the longwall system components will be realized. Improvements in the reliability of equipment and system integration will lead to the fully automated face, and this in turn will permit a complete redesign of the longwall system. These new system designs will bring greater reliability, productivity and lower costs to mine operators.

However, of equal and perhaps greater impact will be the development of a new relationship paradigm between the equipment manufacturer and the coal mine operator. This new paradigm is the result of the enormous pressures and changes which the coal industry has had to endure. The most significant pressure is the price squeeze. During the last twenty years the average price of coal has increased only 4% against 51% for oil and 211% for natural gas. The second major change is the shift in focus from individual pieces of equipment to complete systems. Individual pieces of equipment have reached stand-alone availability in the high 90% range but system availability, which is a compounding of these individual availability figures, typically is much less than 80%. Hence, system integration rather than individual product improvement is the key to competing in the future.

The new relationship paradigm will create a seamless existence between user and supplier, and will focus on the system life-cycle costs. The customer-supplier relationship will become more anticipatory, less reactionary and place greater strategic focus on shared interests. This more holistic approach (i.e. Life-Cycle Management) is evolving in the mining industry. Paying on a cost per tonne basis, rather than being invoiced for discreet sales or services, complements this approach to business and is growing in acceptance. This new approach provides a genuine flexibility and the means to better match costs to the revenue stream.

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Surface Assisted Continuous Underground Mining

R D Peterson

ABSTRACT

Longwall mining has historically been considered an application for only deep reserves. This has changed in the recent past as longwall has become competitive with surface mining. Many resources marked for stripping now lie in the domain of longwall. Today, the best longwall mines compete with strip mines operating in even the lowest stripping ratios. Longwall mining has become more widely applicable than it ever has been in the past and it is now generally considered the method of choice in situations where high levels of coal production are required. But the best way to use longwall mining has become less certain with the opportunities that highwall access provides. Wall-to-wall mining is the use of retreating longwall developed from and retreating toward a highwall. As a surface mine reaches its ultimate highwall, the opportunity for longwall punch mining or wall-to-wall mining is obvious. There are several clear operating advantages to operating a longwall in this way and where highwalls are available, the best economic decision will generally favor planning a direct approach. Yet wall-to-wall is not applicable in every situation even when highwalls are available. Moreover, the advantages of wall-to-wall mining do not always justify the cost of trenching or box cuts to access coal. However, many tactical advantages of operating from the highwall can be realized with the use of specially equipped blind drilled or raised bored shafts. This paper discusses novel techniques for a shaft assisted mining system which permits the addition of conveyor structure and mine systems from the surface to the face for advance and retreat mining with longwall, shortwall or room and pillar panels.

INTRODUCTION

Longwall mining is considered the most productive and cost effective method of underground coal mining used in the world today. The reason longwall systems are so productive is simple. They eliminate common and primary interruptions to cutting coal.

Reasons for longwall's success

Four primary functional systems must be available for any underground mining system to mine coal:

- cutting;
- roof support;
- haulage; and
- ventilation

When all of these systems are provided at the mining face, coal mining can take place. When any one is interrupted, coal mining must cease. Longwall systems prevent interruptions to the above systems and thereby permit more time for cutting and mining coal. Shearers designed to cut coal at the face travel up and down the face and remain in continual contact with it. Powered roof supports eliminate the delays having to do with the ordinary support of the roof above the coal face. AFC haulage systems provide uninterrupted haulage for clearance of coal away from the shearer. Longwall face ventilation is
provided with flow-through ventilation which eliminates many of the inefficiencies with leakage and re-circulation common in room and pillar panels. In longwall ventilation, the primary ventilating current is coursed directly past the mining face so there is no need for curtains, fans or tubing.

**Longwall's co-dependency upon continuous miner (CM) development**

The rate at which a longwall can produce coal is limited by the rate of development which must precede it.

While this fact may seem obvious, it is none-the-less a critical element to successful longwall mining. Moreover, it is commonly overlooked in short term planning. As mines in the US have pushed the envelop in both longwall production and panel length, shortcomings of the development system have become critical. In recent years there have been several instances where development has not been completed in time and it has become necessary to make special arrangements to "hammerhead" CM development from opposite directions in order to get it completed in time for the longwall move.

**Development methods**

Place change mining has approached its limit of effectiveness in the US as it has now become a frequent constraint upon longwall retreat productivity. This, together with the need to improve room and pillar mining has urged the improvement of continuous haulage systems and the use of on-cycle bolting. This has made it possible to take longer cuts to reduce interruptions to cutting from moving the miner from place to place. Continuous haulage in the US has made higher rates of advance possible today than ever before.

Slugish development systems have become a focal point in Australia as well. The standard systems used for development in Australia 10 years ago are now widely considered inadequate for today's mines. These systems typically use a CM to cut single cuts in excess of 100m in length. The primary shortfall with this system is its use of a serial cut/bolt cycle with extraordinarily long cuts and extended shuttle car travel distance. The numerous interruptions to cutting from haulage and bolting are so great with this system that a complete re-evaluation is now taking place throughout the Australian coal mining industry. In an effort to improve advance rates, continuous miners, equipped with on-board and satellite bolting systems, have been used with marginal success. To be fair, these systems generally have not been used until the situation has become so dire as to require impossible advance rates after it is too late to avoid costly longwall outages which result in mine closure.

The most progressive Australian mines have begun looking toward the adoption of place change mining as a method for improving and sustaining high advance rates and have adopted the place change cutting sequence common in US mines.

It is ironic that while both the US mines and those in Australia really face the same development rate limitations the trend appears to be in opposite directions with respect to place change mining. At the same time that place changing is becoming unsuitable for the high retreat rates of the US mines, it is still a quantum leap for the Australian mines. This, combined with the fact that place change mining can be done with standard equipment makes it a viable improvement. This appears to be the driving the movement toward place change mining in Australia.

The key to high advance rates really comes back to the four primary functional systems of mining mentioned above. As long as roof support, haulage and ventilation are provided, the system is available for cutting and mining coal. Place change mining is more successful than the systems which have traditionally been used in Australia because place changing coordinates cutting and bolting cycles. Concurrent operation of these cycles, in separate headings, avoids their interference with each other. Place changing evolved from the notion of coordinating parallel or concurrent activities in separate headings. The coordination of the blasting, loading, haulage and support cycles of conventional mining is what led to the concepts behind place change mining. The development systems of the Australian longwall mines, on the other hand, evolved from the British road heading systems for development with advancing longwalls. In those systems, it was only necessary to drive a single heading ahead of the advancing longwall face and mining efficiency of the heading advance was considered of only minor importance as it rarely interfered with the production of longwall coal.
As mines strive to maximize development advance rates with place change mining, its limits are constrained by time lost in moving the miner from place to place. In order to improve advance rates more cuts must be taken. This is not a linear relationship. As more cuts are taken, the unproductive time spent moving the miner from place to place accumulates. Consequently, the total productive cutting time during the shift is limited by the accumulated moving delay. Thus when viewed with respect to the variables we can control and influence, we can assert a mathematical relationship between total shift production and the time it takes to move the miner from place to place.

The time delay variables with respect to place changing are interrelated. The time available for cutting is equal to the available shift time less the time required for moving, assuming other delays have already been accounted for. The total time required for moving is a function of the number of moves and the number of moves is a function of the number of cuts, which is again a function of the available time. In other words, miner move delays decrease the available cutting time and the number of moves that can be made during the shift. The following describes the relationship between shift production and miner move time all other variables constant.

\[ P = \frac{Tc \cdot S \cdot R}{Tc + RM} \]

Where:
- \( P \) = Shift Production Tonnage;
- \( Tc \) = Tonnes per cut;
- \( S \) = Available Shift Time (Other Delays having been deducted);
- \( R \) = Cutting Rate; and
- \( M \) = Avg. Miner Move Time

Table 1 illustrates the relationship using example cut volumes. Fig. 1 is a graphical illustration of Table 1 extended to the point where only one cut can be taken during the shift. It shows the asymptotic relationship between shift production and move time. Notice that production is maximum at \( t = 0 \) and minimum at \( t = total \ available \ shift \ time \) where only one cut can be taken. This is not surprising. It is obvious that keeping the miner cutting coal for more time will result in higher production at the end of the shift. The most productive number of moves is zero since each move represents an interruption to production. Hence, if all the other delays can be minimized, the ideal system is best organized with the miner kept in one heading to avoid unessential moves. Thus, there is an incentive for reducing the number of moves and a definite need for continuous haulage and a support system that allows the miner to remain in one heading where it can cut continuously.

**Table 1 – Place change mining production limits as a function of miner move time**

<table>
<thead>
<tr>
<th>Example Cut Volume Parameters</th>
<th>Cut Width m</th>
<th>Cut Length m</th>
<th>Cutting Height m</th>
<th>Coal Density</th>
<th>Tc Tonnes/cut</th>
<th>R Cutting Rate Tpm</th>
<th>Time per cut Min</th>
</tr>
</thead>
<tbody>
<tr>
<td>5.20</td>
<td>10.00</td>
<td>2.50</td>
<td>1.35</td>
<td>175.50</td>
<td>10.00</td>
<td>17.55</td>
<td></td>
</tr>
<tr>
<td>Available shift time = S</td>
<td>240</td>
<td>240</td>
<td>240</td>
<td>240</td>
<td>240</td>
<td>240</td>
<td></td>
</tr>
<tr>
<td>Number of cuts/moves = C</td>
<td>13.68</td>
<td>10.64</td>
<td>8.71</td>
<td>7.37</td>
<td>6.39</td>
<td>5.05</td>
<td></td>
</tr>
<tr>
<td>Move Time/cut = M</td>
<td>0.00</td>
<td>5.00</td>
<td>10.00</td>
<td>15.00</td>
<td>20.00</td>
<td>30.00</td>
<td></td>
</tr>
<tr>
<td>Shift Production = P</td>
<td>2400</td>
<td>1868</td>
<td>1529</td>
<td>1294</td>
<td>1122</td>
<td>886</td>
<td></td>
</tr>
</tbody>
</table>
Mines in the US have found that continuous haulage systems permit taking longer cuts. Longer cuts reduce the number of moves and the total cutting time lost. In the trona mines of Wyoming USA, for example, continuous haulage units are used to take cuts of 100m in length. The haulage system not only removes coal or ore from behind the miner but also provides a stationary platform for the installation of bolts. Hence, the system really eliminates many of the delays for both roof support and haulage at the mining face thereby enabling high shift production tonnage.

**Fig.** - Relationship between shift production and move time
Surface assisted underground mining methods

The following points pertain to the functional operation of longwalls and continuous miners and are of particular importance when designing the best mine plan for any new longwall project:

- **Retreating longwalls** are the most productive application of longwall mining. However, longwall retreat requires panel development ahead of it. Hence, retreating longwall is really co-dependent upon continuous miner productivity and longwall retreat rates are limited to the rate at which panels can be developed ahead of time. Maximum rate of longwall retreat is a directly proportional to the rate of development.

- **Continuous miners** can cut 20 to 30 tonnes per minute and sustain 10 to 15 tonnes per minute on a regular basis if they are permitted to cut without interruption. This has been proven in coal mines in the US using continuous haulage.

- **Roof Bolts** can be installed on cycle with the mining and directly behind it at the rate of one to two bolts per minute with on-board bolting machines. On-board and satellite bolting systems are available and can be mounted on the mobile boot end and on the continuous miners to permit the installation of bolts directly behind the miner. A full pattern of bolts can be installed in the face area and permit the continuous miner to advance at 0.32 m/min without stopping. This has also been proven in US coal mines.

- **Direct access from the surface** permits the elimination of haulage delays since conveyors can be fed in directly from the surface to stay with the continuous miner. This is the principle behind continuous haulage and Addington’s Addcar highwall mining system, operated in the US and Australia.

Since the continuous miner is a bottleneck to production with traditional development systems, any improvement to its rate of advance will improve longwall productivity. More importantly, if the mining system design will permit continuous CM cutting, high rates of production can be achieved on development. When this is so, development work can be scheduled in series with retreat rather than in parallel with it and a cost-effective single section mine can be planned.

Wall-to-wall mining is a method of developing and retreating longwall panels off the highwall which is particularly suited to application of these concepts.

**Wall-to-wall mining with tandem miner development**

Fig. 2 illustrates a plan view sketch and Fig. 3, a cross section of a wall-to-wall Surface Assisted Underground Continuous Mining system operating from the highwall. Essentially the structure for the conveyor system is fed in from the surface directly to the rear of the miner so that the tail loading section stays with the tail of the miner at all times. The tail loading section is part of the mobile boot end which is used to pull belt through the structure and off the gravity belt storage unit located on the surface. Two headings are mined in tandem using a CM unit in each heading. In one heading the miner cuts only in straight alignment. In the other heading, provision is made for cutting the cross cut for ventilation.
Fig. 2 – Plan view wall to wall surface assisted Gate Road development
Conveyor belting is stored in a gravity belt storage unit located on the surface. Likewise, mine systems power, water, air, hydraulic and lubrication, are attached to and fed inby from the surface through the conveyor structure. Pipe, cable and hoses for these systems are fed through a compartment fitted to the bottom of the conveyor structure for this purpose.

**Shaft assisted continuous underground mining**

Similar to the wall-to-wall system, Shaft Assisted Continuous Underground Mining (SACUM) takes advantage of the ability to provide mine systems and infrastructure from the surface to the face to permit continuous haulage. The difference is that SACUM uses shafts or boreholes for connection with the surface instead of an open highwall portal. Fig. 4 illustrates the plan view of a SAPM section with tandem continuous miners. Fig. 5 shows the cross section of one CM development heading and Fig. 6 shows the retreat of a longwall or shortwall mining face using the system.

Notice that coal is transferred from the panel conveyor to a mainline conveyor underground. Coal is not conveyed up the shaft. Conveyor technology is not yet available which will handle longwall scale production tonnage up a vertical shaft. Hence, the system requires that coal is conveyed to the surface through an established set of mains. This is the primary difference between SAPM and the wall-to-wall system. In the wall-to-wall case, conveyance is made directly to the surface via the highwall portal.

In both cases, the idea is to provide for development of the underground openings from the surface so high rates of advance can be achieved. The monorope conveyor structure fed from the surface combines the high production capabilities of highwall punch mining with continuous haulage to permit on-cycle bolting and sustained cutting. The combined elimination of delays results in the ability to cut coal continuously throughout the shift and the achievement of high rates of advance.

In some instances, it may be possible to combine wall-to-wall and SACUM concepts making the maximum best use of the highwall for directly accessible panels and using shafts for access to the others. In these cases, it may be wise to dedicate an area in the open cut highwall or end-wall for portal access for the development of a main corridor for coal clearance. When no highwalls are present, box cuts will suffice.
Fig. 4 – Plan view shaft assisted underground panel development
Fig. 5 – Cross section shaft mining system for cm development

Fig. 6 – Cross section shaft mining system for LW retreat
OPPORTUNITIES FOR NEW MINES

From the graph of Fig. 1, it can be seen that there is considerable potential for improvement with continuous miners. The key is to keep the CM cutting coal. This is the goal of wall-to-wall and SACUM.

Indeed, the term continuous miner, in the way that the equipment has been traditionally used, is a common misnomer. The cyclical way it is used rarely permits continuous mining. Place change mining only adds to the discontinuity by requiring movement of the miner from place to place.

At mines in the US, CM units with continuous haulage behind them, have proven to sustain rates higher than 3000 tons per 10-hour shift. Two objectives are achieved when high rates of production like this are reached in development:

- the length of time spent on development is considerably shortened; and
- the unit cost of development coal is considerably reduced.

When these objectives are achieved, the presumption that longwall equipment utilization is of foremost economic importance may be no longer valid. The economic issues surrounding the optimal mix of capital and labor for the new mine warrant re-evaluation.

Re-evaluation of the labor/capital mix paradigm

The high level of capital investment required for longwall equipment generally causes management to overstress the importance of ensuring that longwall equipment operation takes priority over other production activities. However, as development rates increase and the unit costs of development coal become closer to those of longwall coal, manning levels and the coordination of development and retreat operations should be re-examined. Such re-examination is especially warranted for new mines planning the use of the systems described here.

The assumption that longwall equipment should stay productive ahead of other priorities is a paradigm in planning as it masks economic opportunities that come about with improved development. The prerequisite to keep the longwall producing continually is only justified when the costs of idling the longwall are greater than the costs of running it. Hence, the assumption only holds true when there are large differences in productivity levels and unit costs. Since large differences have traditionally been the case, the assumption has generally been valid and in this case scheduling should strive keep the longwall in operation at the expense of efficient labor utilization.

However, when the productivity level of CM development approaches that of the longwall, as it will with the systems described here, and when costs of labor are high, as they are in Australia, economics will favor conservation of labor. When this is the case, the assumption is no longer valid and it becomes more attractive to plan new mines with reduced manning levels. This can be done by scheduling development in series with retreat rather than in parallel with it.

When high CM production is attainable, development labor manning or census levels can be reduced to only those necessary to run a single section mine. This is accomplished using the same crews for retreat as used for development. In other words, the timing of development and retreat coincide in series at the expense of equipment utilization but in favor of reduced manning.

Efficient use of capital balanced with an efficient use of labor will always obtain optimal financial performance. It is important to understand how the mix relates to the bottom line when planning a new mine because this is the most convenient time to set manning levels. The costs of severance are high and will generally prohibit reductions in manning levels after start-up.

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Series development

Series development is the scheduling of production labor to operate development and retreat mining sections using the same crews. By scheduling production in this way, the following advantages are realized:

- all production is sourced from a single section;
- ventilation is only required for one mining section and no overcasts are required;
- all mining systems are provided to a single mining face, hence less system down time;
- maintenance activities are focused on one section;
- only one crew is required in the mine at any given time;
- only one supervisor is required for the mine at any given time;
- transportation is provided with a reduced mobile equipment fleet;
- total manpower requirements are reduced.

Organization Chart

Hypothetical SACUM Mine with Series Development
Manpower requirements with series development

The attached organization chart illustrates the reduced manning requirements for a SACUM operation. Notice that Contractors are used to facilitate the set-up of mining systems to avoid delays. The dashed lines indicate alternate positions. All production manpower used on development to run the tandem pair of miners is used on retreat to run the longwall.

Hypothetical example

The following example, for a hypothetical new mine, examines how series development can improve the economics in each of the following two cases:

- Case 1 – the place change mining section scheduled in parallel with longwall retreat; and
- Case 2 – continuous haulage with tandem miners operating in each heading as with the W2W and SAPM methods described.

Assumptions:

1. Cost of capital = 15% p.a.
2. Capital Longwall Face Costs = $A45,000,000;
3. Life of Longwall Face = 5 years;
4. Labor Costs = $A100,000/man yr.;
5. Longwall Production = 15,000 tonnes/shift;
6. Longwall Operation = 323 days/year X 1.57 s/day = 507 s/year.
7. Longwall Panel dimensions = 250m wide, 3.75m high, 6000m long;
8. Total driveage required per panel = 19,575m/4 = 6525m²; and
9. Development Cut Dimensions = 5.2m wide, 3m high
10. Coal Density = 1.35 insitu

Case 1

- Place Changing (PC) development rate = 30 m/s (632 tonne/shift);
- interruption of longwall for set-up = 0 shifts (parallel sections);

Case 2

- Tandem Advancing Miner (TAM) units = 2000 tonnes/shift/CM 95m/s per CM unit or 4000 tonnes/shift,190m/s total driveage/shift;

2 Assumes 4 gate sets with 100m XC and 3 setup rooms required per 3 panels. Equivalent length of set-up room = .5* face length =375m
• interruption for set-up = 11 operating days;
• Reduction in census for serial development/retreat = 100 people (2 CM sections with 4 crews each plus outbye and support personnel);

The following data are calculated:

• Longwall retreat rate = 6010m/year ~ 1 panel/year;
• Equivalent annual cost of longwall capital including depreciation = $13,425,000/year;
• Cost of Labor for 100 people = $10,000,000/year;

Hence,

Case 1 - Calendar delay for Series PC development = 6525/30 = 217 operating shifts or 138 day/year;

Case Cost of Capital related to delay period = 138/323*$$A13,425,000 = $$A5,744,760.

Similarly,

Case 2 - TAM development time/panel = 6525/190 = 34 operating shifts or 22 days/year;

Case 2 - Calendar delay for Series TAM development = 22 + 11 = 33 days/year;

Case 2 - Cost of capital related to delay period = 33/323*$$A13,425,000 = $$A1,371,594.

The example illustrates the following points:

• there is an economic and functional relationship between the speed of development and the scheduled utilization of capital equipment;
• Even standard place change development, when utilized in Australia can be improved financially with series development. Case 1, with the reduction of 100 people, is improved with series development by 10,000,000 - 5,744,760 = $$A4,255,240/yr; and
• The economics improve at the higher development rates of Case 2. Series Development improves Case 2 by $$A10,000,000 - $$A1,371,594 = $$A8,628,406 by virtue of reduced manning - assuming equivalent capital for Case 1 and Case 2.

In addition to the economic scheduling of labor, additional related tangible and intangible economies accompany the lower manning numbers too. In some instances, as with most new mines in remote areas of Queensland, it is required that accommodation is constructed for everyone on the payroll. Savings related to reduced manning are real and will have a big impact upon the financial evaluation because they require up-front capital. In addition to fewer accommodation units, fewer

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3 This is a reasonable assumption since the number of development sections and reduced mobile equipment capital costs with the SAPM system will likely be offset by additional capital required for shafts and special conveyors.

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pieces of transportation equipment are required for fewer people. Intangible benefits include those associated with a smaller work force such as increased camaraderie team work, and communication.

CONCLUSIONS AND RECOMMENDATIONS

The following conclusions can be drawn:

- Modern longwall mining is more productive than most other mining methods and requires fewer people to operate.
- Surface mines are loosing market share to longwall mines because longwall methods have become more cost effective.
- The most cost effective way to take advantage of longwall mining from a surface mine's highwall favors the direct approach of wall-to-wall mining because of the many operational and safety advantages it offers.
- Continuous haulage is provided with the direct approach when provisions are made to keep the tail loading section of the panel belt with the miner at all times.
- Where wall-to-wall mining can not be used, Shaft Assisted Continuous Underground Mining may be an alternative which offers many of the same advantages of wall-to-wall.
- Tandem Miner Units will provide continuous haulage to each of two development headings thereby taking full advantage of the direct access and eliminating the need to move the miner from heading to heading.

Recommendations

The ability to cut continuously with continuous miners justifies a re-evaluation of the capital/labor mix paradigm when planning a new mine. It is recommended that evaluation of new mining projects should consider focus upon ways to improve development rates so that higher rates of production can be realized with higher advance rates. If this is done to the capacity of the CM systems it may not be necessary to perform development and retreat at the same time.

The ability to cut continuously with continuous miners will warrant a complete re-thinking of how best to organize extraction activities and whether or not the purchase of the longwall is the best use of investment capital justified given the potential of high capacity shortwall systems.

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Transforming Mining - A Framework for Dramatic Changes in Performance

W M Knowles

ABSTRACT

The mining industry has been slow at adopting the latest management philosophies from other industry sectors, such as manufacturing, and in retrospect, with good cause. With the plethora of three-letter-acronyms such as TQM, JIT, TPM and now, BPR each touted as the panacea for profit improvement, you wonder if you must choose one or attempt to embrace them all! In truth, each new insight into organisational behaviour bears some investigation. This paper looks at the underlying principles and elements of successful change strategies, and how several mining companies, including coal mines, in Australia, Chile and Canada, are applying these principles to their operations. These examples illustrate the need to think radically, and to view mining as an overall process delivering value to the customer rather than a collection of functional silos. They also show that often, a continuous improvement initiative is not enough when a fundamental redesign, a paradigm shift, is needed for a performance breakthrough.

THE CHALLENGE

The coal mining industry is again facing a time of uncertainty and challenge with many operations scaling back operations in the face of weak coal prices and high costs. Unfortunately, the outlook for the future is not much better:

- Global competition is increasing, particularly in the steaming coal markets. By the year 2000, six of the largest export coal mines will be in Indonesia, Columbia and Venezuela. These mines are expected to have combined exports of almost 65.8 MT.

- Costs are high. In comparison to its competitors, costs at Australian coal mines are among the highest in the world driven by high labour and other site costs, and are much higher than the low cost producers in emerging countries where FOB cash costs are in the order of US$22-23 per tonne.

- Commodity prices are static and falling in real terms; since 1980, coal prices have declined by an average of 4% per year. Over the next several years, supply is likely to keep up with demand for both coking and steaming coal providing little incentive for upward price moves. This is despite expected strong growth in demand for steaming coal due to the number of new thermal power stations coming on line in Asia.

- Returns to shareholders are low. A survey for the NSW Minerals Council revealed an average operating profit in 1995/96 of $2.61 per tonne; this equates to a 6.2% return to shareholders (Coopers & Lybrand, 1997). By comparison, the average return in the Australian mining industry is 8.8%. Forecasts for 1996/97 project a loss even though governments and employees still make exceptional returns. The long-term trend is illustrated in Fig.

On the positive side, growth in the thermal coal trade is expected to remain strong and opportunities exist for Australian coal producers to expand production and increase exports. This requires, however, that the industry is competitive with other suppliers, particularly those in Indonesia and the United States. It also requires that the return on funds employed be sufficient to attract the capital investment needed to develop and expand operations.

1 Coopers & Lybrand Consultants, Sydney
Industry

These developments pose a challenge, and it is clear that significant improvements in productivity and profitability are required. At the same time though, traditional improvement approaches are either not delivering lasting change or are merely allowing operations to keep pace with the leaders. This is particularly true of Enterprise Agreements where promised results have, in many cases, yet to be achieved, and of technical advances where equal access to technology quickly means that, everyone has access to the same benefits. Productivity improvements, due to technology, are on average 5-9% per year.

The question is therefore “How do we achieve a fundamental change in the way we do things to provide superior returns to our investors and secure long-term job security for our employees?”

Strategies for change

The answer unfortunately is not simple. As people probably suspect, there is no “silver-bullet” to miraculously solve the problem. We need instead to look at how work is performed across functions in order to make these operations more logical, effective and efficient. At the same time, we must also address the human aspects through effective leadership, a clear vision and targets, and individual and team development. Thus, by addressing the business in a holistic sense, we can bring about the improved performance and productivity required for a sustainable future.

This holistic strategy borrows on the concepts of business process reengineering (Johansson et al, 1993), change management and, more recently, corporate transformation (Gouillart and Kelly, 1995). However, in borrowing these concepts, it has been more to explain and provide a framework for our experiences and observations on what works and what doesn’t work in the mining industry rather than to provide a cookbook recipe for change. The elements included within this strategy are as follows:

- Achieve motivation and commitment;
- Create the vision;
• Build the measurement system;
• Develop a detailed understanding of the coal supply chain process;
• Redesign the work processes and infrastructure;
• Develop teams and individuals; and
• Align reward systems.

Implementation is not a sequential process but is achieved by working simultaneously, although at different paces, on all elements. Before we discuss these elements and implementation in further detail, it is worthwhile to review some of the successes mining organisations have had applying these concepts.

• A coal mining company has recently targeted improvements in its production, mine planning and maintenance processes, and is forecasting an increase in production of over 30% in 18 months without increasing the workforce.
• A iron ore mine reduced train cycle times by 20% resulting in deferring a planned $150 M capital investment in new consists; the capital was spent instead on upgrading the fleet of locomotives.
• A copper mine re-engineered the sequential, iterative and time consuming process of new mine development, to a collaborative, concurrent process achieving a "time-to-market" capability of 30 months.

The issues, actions and techniques applicable to each element required to achieve these benefits are discussed in further detail in the following sections with illustrations drawn from examples in the mining industry.

Achieve motivation and commitment

It is understood that achieving anything significant, including a fundamental improvement in mine productivity, requires motivation and commitment not just of management but of the entire workforce. This requires a combination of top-down leadership and bottom-up involvement to create a groundswell of change led by middle managers and directed/coached by senior managers.

First, it requires the active involvement of senior management in the organisation with their staff to arrive at a common understanding of the need for change and the way ahead. This is the most challenging step as middle management often has the most invested in the status quo. They’ve developed the systems and procedures under challenge, and are in the position they are in because of current systems. It therefore requires concerted effort, through workshops, meetings and one-on-one sessions, to gain their commitment to the planned changes. At one operation, we facilitated a series of workshops over a week long period involving Managers, Superintendents and selected Supervisors to evaluate the issues in the coal supply process from drill & blast through to coal processing and train loading. Once the essential issues were defined, actions were identified, prioritised and responsibilities assigned. These workshops followed a detailed diagnostic and allowed the staff to discuss and debate the issues and thus begin to own the change process.

Second, mechanisms are required to engage the workforce in the process, and in a two-way dialogue on what’s required and why. In the past, organisations, such as Hamersley Iron, have employed continuous improvement teams to begin focusing people on looking at processes, customer requirements and quality/cycle time issues. More recently, a coal mining client of ours has been successfully using a series of workshops, as part of its certified agreement implementation process, to focus work teams on their code of behaviour, team skills, customer requirements and improvement opportunities.

Both of these activities are required. Many TQM efforts have failed because they lack guidance from the top, that is, teams focus on trivial issues, generate impossible to implement recommendations or fail to adequately tackle controversial issues. Conversely, management workshops and team building fail because the people who are key to the changes, the workforce, are not involved.
Create the vision

The second element is creating a compelling vision that sets some ambitious stretch targets that become the organisation’s reason for being. The vision should be bold enough to provoke some strong reactions and have some substance and meaning for people. Pepsi’s vision, for example, is very simple: Beat Coke. In the mining industry examples of effective visions may be to:

- Be the lowest cost producer in NSW (if you are at the high end of the cost curve);
- Increase production by 30% in two years with the same labour and resources;
- Cut rail cycle time by 50%; and
- Radically change the mine development process.

These visions are much more tangible and relevant than ones often seen in corporate boardrooms or on noticeboards, such as, “being the preferred supplier of our customers”. Even so, they often need to be translated into simple themes that are directly relevant to a work team. For example, for maintenance team the vision may be “zero breakdowns” whereas for an overburden team the vision may be to “keep the trucks moving”. These themes contribute directly to the corporate vision but focus the team on the critical or high leverage areas within its control.

Build the measurement system

If the vision is important then we need to keep score and help drive us toward the end objective. It would be extremely difficult to become a scratch handicap golfer if we didn’t keep score to see the impact of changes in our golf swing or to reinforce the benefits of practice on the driving range or putting green.

It is therefore obvious that an effective measurement system is required in an operation to track progress against the vision, to reinforce positive behaviour and to highlight the impact of changes. This measurement system must do three things:

1. It must break the vision down into specific quantitative objectives and milestones so that set targets must be achieved by specific dates;
2. It must align the measures in the organisation with the vision; and
3. It must translate the high level objectives and milestones into specific measures for the work teams that reflect their visions or themes and the factors over which they have control.

Consequently, the system involves a set of inter-linked scorecards or dashboards moving from the corporate or strategic level to the teams. Fig. 2 illustrates the relationships between the vision and measures with the outcomes at the various levels in the organisation. At one operation, we have designed a system based on these concepts. The keys to the success of this system have been:

- gaining the operators’ and supervisors’ trust in the accuracy and consistency of the data, particularly that collected by the mine equipment monitoring system;
- involving the supervisors in recording, tracking and discussing the performance trends with their teams on a daily/weekly basis; and
- selecting measures and setting performance targets that are linked to the expected outcomes defined in the strategy.
DEVELOP A DETAILED UNDERSTANDING OF THE COAL SUPPLY CHAIN PROCESS

One of Stephen Covey’s habits of effective people is “beginning with the end in mind”, that is, start with an idea of what you want to achieve. Imprecisely defined outcomes and expectations is a common flaw with many improvement programs. Thus, they lack focus and efforts are spread across a wide-range of activities diluting the effectiveness of the initiative and quickly leading to complaints that the initiative is consuming too many resources and not producing any gains.

We, therefore, need to understand the processes within the coal supply chain and the factors that have the greatest leverage on the outputs, costs and quality of the operation. This analysis defines or calibrates the gap between what is currently being achieved, and what is technically and practically achievable – in other words the process capability. This element also provides the technical basis that under-pins the vision, and provides the route-map that illustrates how the vision can be achieved.
Fundamental to this element is a re-assessment of work methods and practices to challenge the status quo. This is achieved by mapping the important production processes, from site preparation to train loading, as well as the main support processes such as maintenance, mine design/scheduling, and order entry and logistics. The process maps identify the inputs/outputs, controls, mechanisms, and issues with each activity. The processes are analysed to understand the costs, operating cycle times and delays. From this information, internal benchmarks for equipment or machine performance are developed, and the overall process capability is defined. An example of a hauling process is illustrated in Fig. 3.

For support processes, such as maintenance, the costs, effectiveness and importance of each sub-process or activity is analysed to understand its value to the organisation and to define how well it is currently being performed. Those activities which provide high value to the organisation but whose performance is below expectations are obvious targets for improvement.

![Diagram of Equipment Utilisation and Productivity](image)

**Equipment Utilisation**

**Equipment Productivity**

Fig. 3 - Example hauling

External benchmarking can also establish what is possible. There are, however, pitfalls with benchmarking. It is difficult to find similar operations, i.e., strip ratios, geology, infrastructure, and thus the results are always open for debate. Benchmarking a small sample of organisations within the same industry may also lead to a false sense of what is considered “best-practice”. For example, one small mining operation considered its maintenance performance well above average because it had “benchmarked” itself against other small operations within the same company. Actual performance was, in fact, well below average.

**Redesign the work processes and infrastructure**

Many of the activities discussed up until now prepare the ground for change. The critical activities though are those leading to changes in work practices, systems and structures that deliver lower costs and higher productivity. These activities are often difficult to get underway due to organisational inertia caused by:
1. the scale of the task in terms of the amount of work to be done and the magnitude of the gap; and

2. the lack of familiarity of people with managing by fact and in systematically changing things.

The first part of the solution is to begin by identifying and then prioritising specific, concrete actions that address the high leverage areas. Each action should be capable of being implemented within a short period of time, e.g., 4-6 weeks, and should have tangible outcomes that are measurable and relate to the overall objectives. As the initial planning activities are likely conducted by a small group, the resulting actions need to be assigned to individuals or work teams so that detailed action plans can be developed. As part of this process, it is worthwhile working through with those responsible for the task the rationale behind the action, the expected outcomes and deliverables and the broad strategies to be used. To facilitate this process, a simple model is used that first defines the current or “as-is” situation, second explores the desired or “to-be” state, and third defines the broad steps and tasks required to get there.

As an example, one of the key drivers in a truck and shovel operation is the time available and used to move overburden. At one site, a significant source of lost time was shift change and crib. An action was therefore initiated to develop an effective procedure to allow the shovel and a minimum number of trucks to work through crib by staggering the crib times of the crew. The procedure was developed and tested in one crew and then rolled out to the other crews. Consequently, productivity through crib has increased to nearly the same levels at other times during the shift.

The second part of the solution is to provide people with a simple, systematic process and with some basic improvement tools and techniques. At one operation, an improvement process was designed using the following steps: analyse, plan, implement, and monitor. To assist people in understanding the process and their role, we developed a set of guidelines for each step that described the purpose, the preparatory work, outcomes, and tools and techniques available. Specific timetables were established with each work team outlining the meeting schedule, and supervisors were then coached through the process over a 6-week period.

While this element involves substantial bottom-up activity, management support and guidance are required to ensure that the work teams remain focused on key issues and arrive at practical solutions that challenge the way things are done. For example, one of the work teams recommended two alternative practices for spotting trucks at the shovel because they were unable to resolve a conflict between the practices in use by two different crews. This was clearly not the intent, and the Superintendent directed the team to reassess its recommendations and collect further data on the differences in the two practices.

Develop teams and individuals

Another area that receives little attention in many improvement processes is the need to develop the capabilities of the people within the organisation. The changes resulting from the processes described above involve dramatically different ways individuals and teams work and operate. In addition, the improvements are driven and led by line management, and not through some parallel process. This not only involves the people in developing the solution who often have the most detailed knowledge of the process and inherent issues, but also provides the means of engaging people in the process to build motivation and commitment.

Managers and supervisors require new skills as their focus is no longer directing and controlling but leading and coaching. Operators and maintainers require problem solving and team membership skills. These are in addition to machine or process specific skills, such as, the skills needed to proficiently operate an overburden shovel. Training and development, for both management and the work force, are required. Of the two, training and development of management is the most challenging and with arguably the highest payback. The focus of most programs in the industry, however, is the operator or tradesperson. For a start, proficiency standards for supervisors and managers are required that address more than just the technical or statutory elements of their role. From these standards, education processes may be designed to suit the individual needs.

Align reward systems

We act as we are measured and rewarded, and thus our reward systems must support the corporate vision. In many cases, these systems are in direct conflict with our vision. For example, although most organisations support planned and preventive maintenance, the people most often praised are the breakdown fitters who get things going after a failure. We
therefore need, at some point, to look at how people are rewarded and whether these systems are in fact aligned with the vision, business objectives and processes.

Within the coal industry, some practices stand out that do not necessarily support the changes that are required. These are:

- the weekly production bonuses that are based on absolute production volumes instead of profits, or annual plan or strategic targets; and
- overtime, while allowing flexibility, is often abused and encourages wasteful practices.

There is no doubt that considerable effort will be required over the next few years to develop systems within the industry that link compensation and rewards to performance.

MANAGING THE CHANGE

While the strategy outlined above provides the framework for change, implementation requires effective program management including:

- a strong focus on results to link project objectives with business goals, and to set the criteria for success
- a flexible, layered plan that is developed with members of the team and which fixes high-level milestones and intermediate goals, but allows some flexibility in task execution
- sound organisation to adequately define roles and accountabilities and to communicate effectively what’s required and why
- effective coordination and control the project to balance and commit resources, and to integrate tools for planning with reporting, and to formally monitor and report progress including taking action in time to achieve schedules
- mechanisms to evaluate the effectiveness of the plans, tactics and systems to adapt and improve capabilities

Above all, the actions and plans must be seen as an integral part of day to day operations. Therefore, all the activities on site must be integrated within an overall mine business plan. This allows the resource commitments of the often-competing initiatives to be assessed and actions prioritised based on the organisation’s strategy.

CONCLUSION

The coal mining companies today, like many others, face substantial challenges. To address these challenges and realise the opportunities that exist, organisations require new ways of doing things, and need to evaluate and adopt tried and tested approaches used in other industries. This paper proposes a framework and set of principles to significantly increase productivity, reduce unit costs and make better use of existing physical plant and equipment in order to improve the return on funds and provide for a sustainable future.

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Gateroad Development - The Correct Method, the Correct Equipment?

N Gow¹

INTRODUCTION

Longwall mining in Australia is approaching maturity. Longwall production from 'state of the art' Australian longwall mines now rivals similar US based longwall operations. Regrettably however, Australian gate road development rates in general fall short of the required development to longwall production ratios that are required to sustain annual longwall production capabilities. By contrast, most US based longwall mines achieve the required development to longwall production ratios to sustain their longwalls. This is despite many having to drive three heading longwall panel gate headings to meet regulatory requirements.

In Australia the focus in recent years has been on the improvement of roadway support capabilities of continuous miners. The goal has been to integrate coal cutting and loading with roadway support. Notable products resulting from this focus have been the Joy 12CM 30, Jeffrey 2048 and more recently the Voest Alpine ABM20. Other innovations in this arena have been designed and some have been trialed as prototypes. These machines have included the Joy Sump Shearer and Long-Airdox KB2 miner. Despite these improvements, development rates generally still lag behind those needed to meet improved longwall capabilities.

The US coal industry, on the other hand, has generally focused on using existing high capacity continuous miners in place - changing gate road panels together with purpose built high capacity roadway support machines. This system has generally provided adequate development to longwall production ratios in US mines.

As more Australian mines seek improved development to longwall production ratios, some operators have opted to introduce place-changing methods into gate road development. Many operators are considering this option whilst other operators seek to improve an existing more conventional gate road development system.

This paper attempts to demonstrate, through modelling of non-identified actual mine scenarios, the opportunities and options available to mine operators when considering a choice between place-changing methods as compared to continuing with integrated mining and roadway support methods.

As mentioned above, considerable development has occurred in integrating roadway support into the routine operation of the continuous miner. In parallel with this, continuing development of higher capacity continuous miners, suitable for place-changing, has proceeded in the US and these have been made available in Australia.

Up until recently, far less attention had been given to improving coal clearance from the back of continuous miners. The shuttle car was the principal method of carrying continuously mined coal and still largely remains the most popular, despite its aging technology (approaching 30 years) and ongoing inadequacies.

Over the intervening years, several attempts have been made to supersede the need for shuttle car coal clearance. Some operators have experimented with the introduction of “mobile boot end” technology to eliminate the need for shuttle cars, but with only limited success.

Earlier attempts to introduce continuous haulage into Australian coal mines failed because of deficiencies in equipment design and conveyor belting durability, the concept being subsequently abandoned. A bridge conveyor type continuous haulage system has recently been reintroduced into the Australian coal industry. The industry awaits with considerable interest the outcome of its introduction.

Senior Marketing Manager, Long-Airdox Australia Pty Ltd
The recent introduction of battery coal haulers into gate road development has shown promising results. This technology, although relatively new to the Australian coal industry, is well proven in the US, having been used since 1984.

The second part of this paper focuses on coal clearance alternatives for gate road development. The author, from his personal experiences expresses a view as to the relative merits of each alternative, and productivity predictions from recent modelling of non-identified actual mine scenarios.

**THE CORRECT METHOD**

At the outset it needs to be said that there is no categorically correct method. The choice between the place-changing method and the integrated mining and roadway support method primarily depends upon two major factors.

1. **The Depth of Cut**
   
   This is normally determined by geotechnical considerations, particularly as the completed excavation must conform to a specification that supports longwall extraction. Cut depths in Australia range from a maximum of 15 m down to 2 m.

   For the purpose of the supporting modelling, cut depths were set at 6 and 10 metres respectively.

2. **The Pillar Dimensions**
   
   Pillar dimensions have a direct effect on-
   
   a) The percentage of delay directly attributable to changing out haulage units.
   
   b) The percentage of delay directly attributable to flitting the continuous miner to the next working place.

**Model outcomes**

For the purpose of supporting modelling, pillar dimensions were set ranging from 60 m x 30/40 m to 204/250 m x 30/40 m respectively.

As shown in Figs. 1 and 2, the model clearly demonstrates that the place-changing system offers superior productivity outcomes, compared to the integrated mining and roadway support system, for battery haulers up to pillar lengths of 150 m. The result for pillar lengths over 150 m was that the integrated mining and roadway support system offers a superior productivity outcome as compared to the place-changing system.

Shuttle cars were not modelled beyond 150 m as pillar lengths over 150 m generally cannot be serviced by shuttle cars without the requirement of surging. Surging normally results in less than satisfactory outcomes for many varied reasons.
Fig. 1 - Un-a-Hauler productivity analysis (with 6m cut)

Fig. 2 - Shuttle car productivity analysis (with 6m cut)

As shown in Figs. 3 and 4, the opportunity to access a 10 m cut-out further improved the place-changing system's advantage over the integrated mining and roadway support system.
The use of battery haulers allowed access to the place-changing system's superior development productivity as compared to the integrated mining and roadway support system in pillar lengths of up to 200m. Pillar lengths over 200m resulted in the integrated mining and roadway support system being more productive than the place-changing system.

The use of shuttle cars provided productivity superiority in the place-changing system as compared to the integrated mining and roadway support system for all pillar lengths up to 150m, the limit of reach for shuttle cars without surging.
Comment

Detailed modelling suggests that operators can improve gate road development rates in mines through the introduction of the place-changing system of roadway development.

Pillar lengths of up to 150m can be more efficiently driven using the place-changing system, with a 6m cut, and shuttle car coal haulage.

The introduction of battery haulers will allow more efficient drivage of up to 150m length pillars and even longer pillars, of up to 200m length, using the place-changing system and extending the cut to 10m.

THE CORRECT EQUIPMENT

Equipment selection is an arduous and time consuming task. The capital cost of development equipment is considerable and many operators have suffered the disappointment of less than satisfactory results from the introduction of new technology.

The following section looks at the various coal haulage alternatives available to complement existing improvements available in current continuous miner variants.

The general options available in coal haulage equipment are-

1. Shuttle cars,
2. Mobile boot end,
3. Battery haulers, and
4. Continuous haulage

Shuttle cars

Shuttle cars as shown in Fig. 5, have provided the mainstay of coal haulage from continuous miners for approximately 30 years. Their payload capacity ranges from 7 tonnes to 12.5 tonnes.

The use of shuttle cars in gate road development is well understood by coal mine operators and therefore requires no further explanation.

The major advantage of shuttle cars and therefore their continued use stems from the longtime knowledge and experience of mine operators in this equipment. Their disadvantages of sub-optimal payload, poor operator ergonomics, inflexibility of operation and high operational costs are well known to mine operators. Despite this, they remain the dominant coal haulage vehicle used.

Mobile boot end

Mobile boot ends, similar to that shown in Fig. 6, have had limited penetration into the coal industry since their Australian introduction.

The mobile boot end is positioned behind the continuous miner and receives coal directly from the miner discharge boom. The mobile boot end then transfers coal production to the panel conveyor, which runs through the mobile boot end. The mobile boot end, which is self-propelled, moves forward following the continuous miner, and conveyor structure is added to extend the permanent belt whenever necessary.
The use of the mobile boot end is designed to be a continuous process. In practice, safety factors have limited the access of the mobile boot end to being a continuous process. Most mobile boot ends still in the field are currently being used as stationary boot ends.

The advantages of a mobile boot end primarily stem from its ability to provide a high capacity continuous coal clearance system.

The disadvantage of the mobile boot end generally lies in two areas.

1. Discipline is needed in operation of the unit to prevent panel conveyor problems and therefore prevent resultant loss of conveyor availability.
2. The severely constrained ability to re-supply the continuous miner with required materials for roof support and maintenance. Operators have designed narrow access supply vehicles and/or provided the ability to reverse the direction of the belt. The limited ability to re-supply the continuous miner can inhibit and to some extent, negate the advantage of its continuous coal clearance ability.

Battery haulers

Battery haulers, as shown in Fig. 7, whilst relatively new to Australia, have been extensively used in the US coal industry since 1984.

Fig. 7- CHA818 Un-a-Hauler

The advantages of battery coal haulers stem from a number of factors-

1. They have superior payload (11 tonnes to 18 tonnes).
2. They are not constrained in route and/or distance by trailing cables.
3. They have improved operator ergonomics (as compared to shuttle cars).
4. They have proven to be reliable.
   
   The have utility use outside coal haulage.

There are four major disadvantages of battery coal haulers.

1. They are limited in continuous operation due to their stored energy source (batteries).
2. Their application is limited by the severity of roadway gradients.
3. They require underground charging facilities to operate in distant locations distant from the surface.
4. They require a disciplined battery management to ensure optimal performance.
Continuous haulage

A typical bridge type continuous haulage system is shown in Fig. 8.

The system extends and retracts by tramming along side the belt conveyor installation and transported coal is transferred to the panel conveyor via the RFM tailpiece. The tailpiece is capable of taking retraction of the entire length of the system plus the continuous miner.

Only one of this type of continuous haulage is in operation in the Australian coal industry, and it is currently being used in a place-changing multi-heading bord and pillar production panel.

Its use in gate road development is contemplated.

Fig. 8 - Continuous Haulage Unit

The advantages of this type of continuous haulage are seen as it

1. being capable of high haulage rates 600 TPH to 1800 TPH.
2. being able to be integrated into place-changing systems.
3. eliminating downtime caused through changing out battery haulers and/or shuttle cars.

The disadvantages of this system are seen as:-

1. the high capital cost of the system,
2. the system is best suited to angled cut throughs,
3. the belt conveyor road needs to be sufficiently wide to accommodate the panel conveyor and the bridge conveyor sections (normally 6.0m width required),
4. floor conditions in the conveyor roadway are critical to the success of the system,
5. re-supply of the continuous miner is impeded by bridge conveyor for the integrated mining and roadway support system of mining, and

6. a disciplined management of the operation of the system is essential to ensure its success.

COMPARISON OF RELATIVE EFFICIENCIES MINING SYSTEMS AND EQUIPMENT

The successful mine model

Table 1 is a requisite model indicating ultimate development needs to service a "typical state of the art" longwall mine. The mine has to achieve economic key performance indicators of 3.0 million ROM TPA and output per man year of at least 15,000 ROM tonnes.

Table 1 - Requirements for the typical 'State of the Art' modern longwall mine

| DEVELOPMENT MODEL REQUIREMENTS FOR THE TYPICAL "STATE OF ART" MODERN LONGWALL MINE |
|---------------------------------|---------------------------------|
| **ECONOMIC KEY PERFORMANCE INDICATORS** | **SETTING MINE PRODUCTION PARAMETERS** |
| Annualised Production | Annual ROM tonnes divided by output per man year = Workforce Size Workforce Size Limits Mine To:- |
| Output per Man Year | 3,000,000 Divided by 15,000 - 200 persons 1 x longwall unit plus 2 x development units |
| 3.0 Million ROM Tonnes } eg. South Bulga design |  |
| 15,000 ROM Tonnes } parameters |  |

**CALCULATING DEVELOPMENT REQUIREMENTS**

**Parameters**

- Tonnes per meter of development
- Tonnes per 1 meter longwall shear
- 2 x heading gate roads pillar dimensions

**Calculations**

1 x metre of gate road panel advance = (100+100+25)/100

Therefore the ratio of development tonnes to longwall tonnes is:

Gate road development tonnes @ 3,000,000 tpa & ratio 1:17.6 is:

Add is a further 3280m for main road development:

Calculate new development ratio: 3,000,000 divided by 11,534 x 21.18

Calculate annual production weeks as:

Calculate required weekly production as:

Calculate weekly development tonnes as 71,429 divided by 12.28

Calculate the number development unit shifts per week (p/w) as:

Calculate the required metres per shift as:

COAL98 Conference Wollongong 18 - 20 February 1998
Calculate the time required to complete each pillar as:

\[
\text{Calculate the number of weeks to complete pillar @ 14 shifts p/w as:}
\]

\[
\text{Add effect of each pillar service extension @ 6 x shifts or 0.4 weeks as:}
\]

\[
\text{Calculate service extension compensated required development metres/shift:}
\]

\[
\text{Calculate Effect of Development System Utilisation}
\]

The above calculation assumes that the development system is 100\% available

\[
\text{Calculate availability adjusted development requirement using 70\% availability.}
\]

\[
\text{NOTES}
\]

The above development model indicates the required shift development rates to support longwall production and achieve a combined development plus longwall annualised production of 3.0 million tonnes ROM.

Both development units would be required to achieve 17.45m per shift each for 14 shifts per week for at least 42 weeks per annum.

The model requires that two development units achieve at least 17.45 metres per shift for 14 shifts per week and for at least 42 weeks per annum. Only a minority of Australian longwall mines have achieved this level of development. The majority of Australian longwall mines would attain average development rates considerably less than this benchmark.

Opportunities for improvement

The author suggests that a choice of the most appropriate development system together with a system-matched suite of development equipment will provide the needed quantum improvement. To this end, some comparisons are presented to indicate options and their relative efficiencies.

A direct comparison between shuttle cars and battery haulers is shown for both the place-changing system (10 metre cut) and the integrated mining and roadway support system in Figs. 9 and 10 respectively.

In the modelled place-changing gate road development system, battery haulers offer efficiency improvements over shuttle cars ranging from 14\% down to 4\%. Battery haulers also provide opportunities to achieve acceptable development rates in pillar sizes exceeding 150 metres which is not achievable using shuttle cars.

In the more conventional integrated mining and roadway support system for gate road development battery haulers offer efficiency improvements over shuttle cars averaging 16\% to 17\%. Battery haulers also provide the opportunity again, to achieve acceptable development rates in pillar sizes exceeding 150m, which is a not achievable using shuttle cars.

The model demonstrates that battery haulers provide the opportunity to extend pillar length to 250 metres without efficiency penalty, compared to shuttle cars hauling coal from the drivage of 60m and 100m pillars.
Fig. 9 - Shuttle cars vs Un-a-Haulers place changing development

Fig. 10 - Shuttle Cars vs Un-a-Haulers Integrated mining and roadway support system (Conventional)
Continuous haulage

Modeling of continuous haulage for two heading gate development has been restricted to pillar lengths not exceeding 60m. The reason for this restriction is to keep the total length of the continuous haulage system to no greater than 100m, to enable reasonable management of the system in the confines of a two heading layout.

It is noteworthy to examine the modelled results from the introduction of continuous haulage as shown in the table below:

<table>
<thead>
<tr>
<th>Pillar Dimensions</th>
<th>Place-Changing</th>
<th>Conventional</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>6m cut</td>
<td>10m cut</td>
</tr>
<tr>
<td>60m x 40m</td>
<td>69%</td>
<td>---</td>
</tr>
<tr>
<td>60m x 30m</td>
<td>---</td>
<td>80%</td>
</tr>
</tbody>
</table>

The indicative comparison of possible annualised savings through the replacement of shuttle cars by battery haulers in a particular mine application. The comparison is shown in Fig. 11.

The particular mine drives 2 x gate roads using a Voest Alpine ABM20 and shuttle cars.

Mobile boot end

In relation to the mobile boot end, the author has experience in the use of this equipment in single entry development.

Gate road entry development rates superior to that achievable by shuttle cars were experienced. However, regular shift development rates exceeding 6m caused continuous miner re-supply problems, which often impeded the use of the system.

Mobile boot end technology can provide improved results, particularly in single entry work, providing some compatible re-supply system is developed and integrated into the total mining system.

CONCLUSION

It is believed that coalmine operators have the opportunity to further consolidate improvements in gate road development rates.

Whilst significant technology advances have been made in integrated continuous miners, little attention has yet been given to improving coal clearance from the continuous miner.

Battery haulers provided efficiency gains across a spread of pillar sizes ranging from 110m x 40m to 250m x 40m. In an annualised meterage requirement for that development unit of 5,000 metres, efficiency gains converted to savings ranging from $0.76 million pa to $1.95 million.

This paper alludes to the opportunity for mine operators to improve development rates through the introduction of better-matched development systems and more efficient coal clearance systems currently procurable.
Fig. 11 - Comparison of annual savings Un-a-Haulers vs Shuttle cars in gateroad development
An Integrated Mine Development and Supply System

T Muguira

INTRODUCTION

In the face of intense global competition and other business pressures on coal miners, ongoing quality initiatives and continuous process improvements are needed to enhance business performance. By viewing a minesite in terms of key processes, rather than departments, and employing innovative technologies and better applying organisational resources, there exists enormous potential to achieve reductions in process cost and time.

The paper identifies and selects one critical coal mining process, namely the supply system, to think about how productivity improvements might occur and what changes might be employed to enhance overall system performance. While the supply system is only one of few processes existing at a minesite, the paper addresses only the supply process, but includes interaction across traditional interdepartmental boundaries. The paper has been prepared for a wide audience and is based on experience and observation at several underground mines however some of the concepts are a result of “dreaming” in the face of global competition.

BACKGROUND

Brambles Coal Services (BCS) has conducted a review into the potential for improvement of the supply and materials handling functions of thirty underground coal mines. The review identified some of the issues that adversely affect underground coal mining operations as follows:

1. development advance rates must satisfy longwall production;
2. unreliable supply infrastructure to support development can affect the ability to improve metres advance rate;
3. internal resources are stretched and not able to consistently address effective process improvement issues;
4. employees focus on tonnes with limited regard for cost control; and
5. some minesites are not fully aware of the root causes of the problems resident in their logistics processes and therefore no strategy improvement formulation or implementation is occurring

A significant number of respondents made comments that related to operational logistics and ineffective activities in the supply chain. Table 1 lists some high ranked problems that pertain directly to existing mine supply systems.

While every mine is unique, with individual / specific issues and needs, BCS has proposed a flexible Underground Supply System (USS) solution that can be tailored to a particular minesite. The USS is not directed toward issues of labour relations, mine conditions, equipment availability and panel layout, but rather applies to supply logistics and materials handling issues.

Brambles Coal Services, Newcastle
Table 1 – Typical mine site observations / problems / issues pertaining directly to existing mine supply systems

- Coordination of deliveries (stock control and double handling) issues;
- Minimum bulk handling of high usage items: stonedust, salt, diesel, soluble oil, ballast, concrete;
- Wastage: damaged pallets, broken drums, torn bags, gear returned to surface is trashed;
- Increasing requirement for greater quantities of stone dust spreading underground;
- Desire for "cassette" type storage / transport of strata support materials;
- Transport cycle times significant and increasing;
- High demand on Eimco's for underground materials handling;
- Foreign materials in coal stream (timber, packaging, steel); and
- Injury potential due to manually handling of strata control material

UNDERGROUND SUPPLY SYSTEMS

General
The Brambles Coal Services USS is not just another efficiency project that can be dumped on a minesite. It is an activity based process that delivers customer value and provides a mechanism to achieve regular overall process improvements. The primary objectives of the USS are to increase mining advance rates and reduce the cost per metre/tonne of mined coal, by addressing and improving minesite supply logistics. It is recognised that to succeed in the analysis and implementation of any solution a partnership relationship is required between BCS and the mine such that the proposed processes changes can be effectively designed and implemented and then have performance measures applied.

Approach
Analysis initially involves a joint understanding of operational activities, progressing through an assessment of specific issues that inhibit performance improvement due to inefficient supply and materials handling. An integrated solution based on the elements of the USS follows the joint party analysis that involves all organisational departments.

The framework given in Fig 1 is indicative of the staged approach of the analysis. The framework forms the basis of the supply system continuous improvement program, highlighting the need to understand the process activities, identify critical areas for review and measure progress against predefined performance indicators. A benefit analysis designed to provide a comparison between the present system and its associated costs, and the new integrated process solution and its associated costs, is also developed, such that potential savings can be quantified.

Fig 1 provides a framework to study the entire supply chain from a supplier's facility to the point of use at the mine. Typical areas that may be considered include:

- Planning and monitoring individual processes / procedures;
- Ordering and reordering processes (how are they communicated?);
- Supplier's packaging methodologies;
• Transport to minesite;
• Interaction / communication between supplier and minesite;
• Receival procedures at minesite;
• Surface handling / rehandling methodology at minesite;
• Storage facilities (bulk and otherwise);
• Materials transfer from surface to end user (including end user waste and the minimisation of double handling and manual transfer of supplies);
• Relationship between materials handling and people injuries;

High cost / usage materials must be studied within the context of individual supply chains, then studied with a more holistic view. Synergies in the management areas of, ordering format, receival procedures, storage, handling, packaging, transfer methods, damage and wastage should also be considered.

It is imperative that the supply system be viewed as a process and not in terms of departments, such that redesign from beginning to end, employing whatever innovative technologies and organisational resources necessary, can occur. The overall project is timetabled generally as follows:

1. Design (prepare functional specifications);
2. Project Management of the new process (follow through the functional specification and schedule tasks);
3. Training and document preparation for users;
4. Equipment installation (hardware);
5. Commission system (hardware and systems) ; and
6. Ongoing review and ongoing improvement (against performance measures).

While this sounds like significant effort, the framework, when applied to a defined timetable, leads to a planned and integrated solution that will lead to waste reduction, reduced injury, better utilization of resources, predictability and ultimately lower cost per tonne. For example the recent benefit analysis highlighted areas where implementation of a proposed USS at a single mine provided an ongoing financial benefit of 7.3 million dollars per annum in addition to intangible benefits like:

• Reduction in heavy vehicle movements over travel roads by 144,000 kilometres;
• Five percent improvement of wastage / damage;
• Improved safety in material handling;
• Enhanced performance measurement in panels;
• Potential to improve development rates without additional labour; and
• Improvement of planning and control of materials, people and machines both underground and on the surface.
Materials Flow

The materials flow schematic proposed at for high usage/cost items is given in Fig 2. The system involves an off-site store where strata control cassettes are salvaged, cleaned and refilled. The off-site store also prepackages conveyor belting and structure, pipes and fittings, hoses and mining cables such that these items are delivered to the central on-site store, in a manner where they can go directly underground. The central on-site store area is designed to support high usage mining consumables, while minimising disposable packaging that traditionally enters the coal production stream. In addition to the central store area, the minesite surface supports a main store that supplies traditional items such as, engineering spare parts and consumables, plus a bulk materials storage facility. Bulk minesite surface stored items including oils, stonedust, ballast, diesel and cement powder are supported by bulk supply underground transport equipment that is custom built to suit unique minesite conditions. Stonedust, ballast and concrete enter the mine through strategically located boreholes where they can be delivered close to the point of usage, again utilising custom built underground transport equipment.

The minesite materials flow depicted in Fig 2 supports an underground storage area for cassette rehandling however the activity is surplus for mines that have direct vehicle access from the surface. Delivery of supplies from the underground store is by custom built transport equipment, that mechanically interfaces with development and longwall mining equipment.

Information flow and the enablers for process improvement

While the materials flow diagram forms the basis of what is seen physically on site, it is the enablers of process improvement (process procedures, mechanical designs, computer based control) that provide a holistic approach to integrate all of the supply and material handling activities.

Process procedures means a functional specification detailing exacting process requirements and how information flows across and between various mine departments and to suppliers. Mechanical designs means tailor made engineered equipment that allows for effective distribution of supplies. Computer based control means establishment of an integrated information technology platform that links all mine departments, to provide a legitimate management information system that supports supply logistics.

The desire to build information systems across functional boundaries is not new and there are many examples of cross functional information technology “solutions” that never flourished. In addition, there exists many minesite examples where information technology “solutions” failed or simply did not perform. Given this scenario, it is fundamental to understand what process procedures and computer based control means, at minesite level. Graham (1972) indicated that innovation in the use of information technology and communication must be combined with how information is used and structured. Graham’s observation of the systems analyst, who had responsibility for designing computer applications, was first expected to redesign the process. As with the USS, one must focus on an agreed functional specification that addresses the differing needs for information, from all users, prior to the application of information technology.

Information flow depicted in Fig 3, and the subsequent management of that information is the basis of a successful USS. Information flows freely across and within departments to the key planning areas via a centralised communication facility. The forward planning ability of each production unit helps in the coordination between supply (includes the supplier, off-site store, on-site store) and production departments to allow for effective transport coordination and prioritising of value chain activities. The purchasing department has a thorough knowledge of current and forecast activities as opposed to having to rely on intuition and historical performance data alone.

Fig 4 represents the integrated information technology solution that supports the USS. It is built around the jointly generated functional specification and includes the following philosophies:

1. It is a holistic approach covering the entire supply and material handling aspects across all departments. It includes the three process enablers of process procedures, mechanical designs and computer based control;

2. Supply process is driven by panel performance;
3 Containerisation of supplies / materials;
4. Off-site cassette restocking;
5. Bulk handling of materials / ballast, stonedust;
6. Integrated surface material handling through a centralised facility;
7. Underground transfer facility for interim storage of some materials (only necessary at some mines);
8. Mechanical reprovisioning of continuous miners;
9. Data communications to panels and longwall;
10. Delivery scheduling;
11. Interface and shared information with existing surface computing facilities.

The ability to receive performance data from production panels will not only drive the supply function, but allows for geological information, machine availabilities and the measurement of other production based performance indicators.

CONCLUSION

The USS is an activity based process built on a framework of continuous improvement. It involves a holistic approach that cuts through inter-departmental barriers in order to not only formulate a supply strategy, but to redesign the entire mine supply process, then implement the process and effectively monitor its process.

While every mine has some unique characteristics, the USS provides a flexible solution that can be tailored to a respective minesite. USS implementation can be efficiently timetabled in six overlapping stages, namely, design of the functional specification, project management, training, installation, commissioning and ongoing review.

The USS is but one of a few key processes on a minesite, and its implementation can provide both positive tangible and intangible outcomes that lead to a lower cost per metre advance or tonne of coal.

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Report on Hydrogen Sulphide Experience at Southern Colliery

M Ryan¹, T Harvey², J Bride³ and M Kizil⁴

ABSTRACT

Hydrogen sulphide (H₂S) as a seam gas has occurred in several locations in the German Creek seam at Southern Colliery. This paper overviews the techniques developed to quantify the H₂S content of the seam, overviews research into controlling and reducing H₂S emissions from the seam, and details the procedures used to mine through the H₂S zone in 702 longwall panel. Learning points from research and mining experience are reviewed for application in the next longwall block.

INTRODUCTION

Southern Colliery is operated by Capricorn Coal Management Pty. Ltd. and is situated inland from the coastal cities of Mackay and Rockhampton Qld (See Fig. 1).

Fig. 1 - Locality map

¹ Mine Manager, Southern Colliery Operations, Queensland
² Senior Mining Engineer Underground, Shell Coal Pty Ltd, Queensland
³ Director O&B Scientific, Victoria
⁴ Department of Mining, Minerals and Materials Eng., The University of Queensland
The colliery is an underground longwall operation, mining the German creek seam, the lower most major economic seam of the Bowen Basin's Permian German Creek formation.

Hydrogen Sulphide (H,S) gas was first detected in the open cut workings adjacent to the proposed entries for Southern Colliery in October 1987.

Subsequent development of the mine (1988) encountered H,S for the first 800m of mains and during the development and extraction of the first longwall block (601) (Smith, Phillips and Byrnes, 1990) (See Fig. 2). Significant H,S was not found again until the development of the gate roads for the 701 longwall block in January 1995 (Ko Ko and Ward, 1996). The H,S zone was not mined in 701 longwall panel as poor roof and weak floor conditions required the longwall face to be relocated outbye of the H,S zone. In June 1996, H,S was encountered again during development of the 702 panel maingate. Development, drilling and testing has delineated a continuous H,S zone through at least four longwall panels (701 to 704) (See Fig. 3).

![Diagram of German Creek and Oaky Creek ML 1831 and ML 1832](image)

**Fig. 2 - H,S Zones in Oaky Creek and German Creek Collieries**

Other significant occurrences of H,S have been found in the Bowen Basin, at Oaky Creek Mine 15 km west of Southern Colliery and at Collinsville 300 km to the north. Minor occurrences have been found at Newlands, Crinum, Gregory and Gordonstone mines (See Figs. 1 and 2).

This paper reports on the experiences of mining through the H2S in the 700's district of the mine.
EXPLORATION TO DEFINE H₂S ZONE IN 701 LONGWALL PANEL

A surface drilling program of eight holes coring the German Creek seam commenced in February 1995. The coal was analysed using drum tumble and silver nitrate tests to quantify the H₂S content of coal. In June 1995, during the development of 701 maingate B heading, H₂S was encountered between 18 and 23 cut-throughs. Rib samples of coal were taken and tested. The results of these samples showed minor quantities of H₂S, despite deputy's reports indicating high levels.

In August 1995, it was decided to conduct further drilling programs from the surface and from underground to define the extent of the H₂S zone and to facilitate a trial chemical infusion of the zone.

Underground drilling program for 701 longwall

The drilling program started in September 1995 drilling horizontal holes using a ProRam from 701 tailgate at 22 cut-through. These holes were not successful; one hole reached its target, the other holes terminated in the roof or floor close to the collar. In October 1995 a Diamac 260 rig was used to drill five 170m holes in a fan pattern from 701 tailgate at 22 cut-through. Core samples were analysed for H₂S content. The drilling program and associated tests gave an indication of the levels of H₂S present and the extent of the zone (see Fig. 3).

It was discovered that the Drum tumbler, silver nitrate tests and delays between sampling and testing were providing varying and questionable results.
INITIAL RESEARCH INTO QUANTIFYING AND REMOVING OF H₂S FROM COAL

Infusion by hydraulic fracture method

In October 1995, five vertical holes were drilled from the surface, into the 701 longwall panel. Staff from CSIRO Petroleum conducted permeability and stress measurement tests. This was followed by a trial infusion, injecting zinc chloride (ZnCl₂) with a fluorescein dye tracer using the hydrofrac method. This proved unsuccessful as post infusion drilling showed no evident reduction in H₂S levels, or any trace of the 35 000 litres of the dye solution (Ko Ko and Ward, 1996). Had the area been mined then a better assessment of the success of hydrofrac could have been made.

Spray technology

After the failure of the infusion trials, efforts turned to developing a chemical spray to absorb H₂S from the atmosphere. Laboratory tests were conducted using a range of chemicals in US in the oil industry for the removal of H₂S from drilling mud. Efforts were made to source a chemical used in China (Peng et al, 1992) without success. However literature research revealed the importance of pH levels in neutralising H₂S.

The pH has to be high enough to ionise the H₂S thus enabling it to be removed by oxidation. Fig. 4 shows the equilibrium of the aqueous system, H₂S, HS⁻ and S²⁻ with relative concentrations versus pH (Garrett et al, 1979). Following initial discussions with Shell, CSIRO, ICI and mine personnel a test rig was set up by the CSIRO. The aim was to simulate a H₂S contaminated mine roadway, and to test the effects of different chemicals, varying pH, varying flow rates, and varying spray droplet size on H₂S contaminated air in the test rig (See Fig. 5). Three series of duct tests were undertaken producing some positive results, the best giving a 91% reduction in H₂S levels (see Fig. 6). The initial test used sodium hydroxide to control pH and sodium hypochlorite to oxidise H₂S. This test proved effective, however the pH of the solution at 12.4 was unacceptable for the mining environment. Varying spray droplet size between 50 and 150 micron produced little difference in the effective removal of H₂S. Later tests replaced sodium hydroxide with a buffer solution to keep the pH below 10 and these produced acceptable results. However tests without hypochlorite, using buffer only, reduced the effectiveness of H₂S removal by more than 50%.

![Fig. 4 - pH effect on equilibrium of sulphur species in aqueous solution (Garrett et al, 1979)](image-url)
The tests proved that levels of H₂S in the atmosphere could be controlled with economic quantities of chemical sprays, however the best results were achieved at pH levels above those acceptable in the mining environment. The most successful chemical, sodium hypochlorite, was potentially more corrosive on face equipment than the H₂S itself.
Drum tumbler

A Drum tumbler system with the ability to constantly sample gas during coal breakage was designed and manufactured by O&B Scientific (see Fig. 7).

The system rotated a 255 litre drum constructed from High Density Polyethylene (HDPE), end for end about a central stainless steel shaft. The drum tumbled the sample at 20 RPM for 60 revs. The period of rotation produced coal breakage representative of the size of coal on the armoured face conveyor (AFC). The test enabled the prediction of the volume of H$_2$S released into the atmosphere from a given sample.

DEVELOPMENT OF 702 MAINGATE

H$_2$S was encountered during development at 16 cut-through 702 maingate in June 1996. A monitoring program was set up to compare the predicted and actual amount of H$_2$S released. The predicted H$_2$S released was determined from face and rib samples gathered during mining operations and tested using the drum tumbler. The actual release was determined by logging coal production, measuring ventilation quantities at regular intervals, and monitoring the H$_2$S levels in the return at 15-seconds intervals. Reasonable correlation was found between the results of predicted and actual H$_2$S released (Harvey, 1996). A similar monitoring program was set up when 703 maingate intersected the H$_2$S zone in June 1997.

ACARP PROJECT

A research project, “Maximising Coal Production in the Presence of H$_2$S Seam Gas” jointly funded by ACARP, Oaky Creek and Southern Collieries, was set up to investigate:

![Diagram of Drum tumbler system](Fig. 7 - Drum tumbler system)
- Occurrence of H₂S
- Prediction of H₂S release;
- Storage Mechanisms;
- Mine ventilation system and control measurements;
- Mining options;
- Permeability; and
- In seam chemical neutralisation.

The project is being staffed by a research team from the University of Queensland, Departments of Mining, Minerals and Materials Engineering, Earth Sciences and Chemistry.

The project was started with mine funding to enable monitoring of Oaky Creek Longwall 8 as it mined through a H₂S zone in October/November 1996. Samples were taken from the ribs, at 10m intervals in both headings of the H₂S zone and from the face during production. These samples were tested for H₂S content using the modified drum tumbler. Sub samples were sent to University of Queensland for further analysis. The data from H₂S sensors on face, ventilation and production for each shift was recorded and analysed to determine the actual H₂S release in litres per tonne.

At Southern Colliery, in April/May 1997 a surface and underground drilling program was conducted to determine the extent of the 702 H₂S zone and to investigate the 703 zone. Analysis of the drum tumbler results from this program and from previous rib sample data was correlated with actual H₂S release from continuous miner development at Southern Colliery and from Oaky Creek Longwall 8. The results of this analysis were used to produce a contour model of the predicted H₂S release in 702 and 703 longwall panels (See Fig. 3).

UNDERGROUND INFUSION FROM HORIZONTAL HOLES

In July 1997 it was decided to conduct a trial infusion with a buffer solution of sodium carbonate and sodium bicarbonate in an attempt to reduce the H₂S emission from 702 longwall. Another attempt at infusion was made because:

- Infusion provided a pro-active approach to reducing H₂S emissions;
- mining the infused area would enable effective evaluation of its success; and
- research into chemicals to absorb H₂S enabled appropriate infusion chemicals to be selected.

Nine holes were drilled using a ProRam, from “C” heading in 702 maingate, between 16 and 17 cut-throughs. The holes were approximately, 6 m apart, 90 m in length and at an angle of 45 degrees to the main cleat direction (See Fig. 3). The holes via a shut off valve were fed into two separate manifolds, connecting alternate holes back to a pump and 8000 litres storage tank. Initially underground water supply was connected to the system and pressure and flow rates recorded to establish the permeability of the seam. Over a period of 14 days approximately 50 000 Litres of buffer solution was pumped into the seam, at a maximum pressure of 1800 kPa, using different valve configurations and various pump and flow-back sequences to ensure the maximum saturation of the seam in the time provided. During the flow-back process the return fluid was sampled and H₂S content. Preliminary analysis of results indicates that approximately 18 000 L of H₂S was taken into solution during infusion, which represents approximately 20% of the measured H₂S release from the infused zone.
MANAGEMENT PLAN

A management plan was developed at Southern Colliery with the primary aim of preserving the health and safety of those working in areas affected by H₂S. In order to achieve this, the plan was designed to:

- Prevent any persons from being exposed to concentrations of H₂S above 10 ppm in the general body of air;
- prevent the maximum concentration of H₂S anywhere in the mine from exceeding 200 ppm;
- protect mining equipment from H₂S corrosion; and
- maintain adequate production.

The ACARP research team worked closely with mine operators during the extraction of the 702 H₂S zone. Their purpose was to monitor emissions and worker exposure levels each shift and prepare data for feedback to operators each day.

VENTILATION

Ventilation in the panel was conventional antitropal on the face and homotropal in the maingate conveyor road. Compressed air venturi fans were placed in an exhausting vent duct system from the BSL to outbye of the pantechnicon. The system was designed to duct H₂S-laden air generated in the BSL to outbye of the pantechnicon into the homotropal conveyor road (Fig. 9).

MINING PROCEDURE

The rate of cutting coal was used to control the release of H₂S. When H₂S levels approached 10 ppm the shearer haulage was stopped to reduce H₂S emissions.

Some of the procedures that were put in place to limit access to the face and reduce risk and exposure to H₂S were:

- All people on the face were to be located on the intake side of the shearer, when the shearer was cutting,
- all people inbye of the last accessible cut-through on the intake roadway were required to carry a face mask at all times,
- face masks were worn by all persons on the face line when the armoured face conveyor was conveying coal and/or the shearer was cutting coal, and
- personal H₂S monitors were carried by the deputy, the chock operator and the shearer operator.

Several alarms, both visual and audible were situated in the face area to warn when H₂S was approaching pre-determined levels. These alarms were set at the following levels:

- The H₂S monitor on the maingate drive was set to give a visible alarm at 10 ppm;
- the power to shearer was cut off if the tailgate H₂S monitor recorded a level of 200 ppm; and
- coal cutting was stopped if the monitor in the homotropal conveyor road reached 100 ppm.
**H₂S DETECTION AND MONITORING**

**Face samples**

During the mining of the H₂S zone in 702 longwall panel, a total of 153 coal samples were taken from the face for testing. The data from these samples was used to produce a contour map of the H₂S zone (See Fig. 8). Samples taken approximately 20 cm from the intersection of infusion holes and face showed a 75% reduction in H₂S content compared with samples taken over a metre away from holes.

**Fixed detection systems**

Electro-chemical H₂S sensors (AMR) were used to continuously monitor H₂S gas concentration levels within the longwall ventilation circuit. Sensors were placed at either end of the longwall face (See Fig. 9), outbye in the longwall homotropal conveyor return, in the tailgate return and at two locations along the main trunk conveyor system.

**CORROSION**

Corrosion of materials due to H₂S in moist atmospheres is well known. Damage that occurs to steel is minimal compared to damage caused to copper and some copper alloys. The most noticeable damage experienced at Southern Colliery was to the coating (92% copper and 8% tin) on chock legs. Field and laboratory tests undertaken by consultants (ETRS, 1997) on behalf of CapCoal have shown that the coating is attacked by H₂S, leaving an outer coating of copper sulphide, underlain by layers of tin oxide and tin. This coating, when mixed with coal dust, forms a rough crust on Chock Legs. Damage to chock leg seals can be attributed to this crust. To reduce the corrosion potential, the chock legs on 702 longwall were coated with raw solsenic oil and wrapped in plastic. This method worked well provided the wrapping remained intact.

To check the corrosion risk to electrical components, copper strips were placed in electrical boxes on the face area and along outbye conveyor roadways. Each strip was weighed and numbered prior to installation. After mining through the H₂S zone these copper strips were checked. Those placed in flameproof boxes in the face area were unaffected. However minor corrosion was found on strips in the outbye belt starters.

![Fig. 8 - H₂S contour based on face samples correlated to measured H₂S release per shear](image-url)
PERSONAL PROTECTIVE EQUIPMENT

The specifications of personal protective and monitoring equipment issued and used during mining in H₂S zone is given in Table 1. Face masks and filter units were accepted by operators and proved to be a viable protective device for the situation.

Personal monitors gave repeatable results and proved reliable if charged correctly.

LEARNING POINTS AND COMMENTS

Exploration and testing

The detection of H₂S zones during exploration drilling has proven difficult due to the size and shape of zones. The ACARP research program includes a study of indicators in coal that may improve detection.

Drum tumbling of core or rib samples can give a good estimate of the H₂S released during mining. A reliable estimate requires the results to be correlated with actual release from mining in a H₂S zone in that seam and the samples must be tested on the same day that they are collected. The ACARP project is investigating the bonding between H₂S and coal and release mechanisms of H₂S from coal.

Fig. 9 - Longwall access areas and face ventilation for 702 H₂S zone

Control of emissions

Initial attempts at infusion and using hydrofracing were unsuccessful due to use of an unsuitable chemical and the inability to determine the extent of infused zone in coal. Subsequent infusion tests using parallel holes in seam and a buffer solution did reduce H₂S emissions. The extent of this reduction is still being evaluated and a more extensive infusion program for next panel is being considered.
Atmospheric control of H₂S levels by sprays has been shown the potential to greatly reduce H₂S levels, however the chemicals used are too corrosive for the face environment. A test using buffer solution to wet coal as it entered the Beam Stage Loader (BSL) gave encouraging results and this concept is being evaluated for control of H₂S release on (AFC) and maingate for the next panel.

Mining experience

The workforce accepted the operating procedures, protective equipment and monitoring equipment. This was in no small part due to the previous experience of mining H₂S affected coal in the mine, the training given to the workforce and the daily feedback of exposure levels and face emissions. This feedback enabled longwall operators to gain confidence in their ability to control H₂S emissions. Overall productivity in the zone was reduced by about 20%.

Sources of H₂S during mining

Data on H₂S sources has not been fully evaluated but initial indications are shown in table 2.

<table>
<thead>
<tr>
<th>Source</th>
<th>Range</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shearer</td>
<td>50% to 70%</td>
</tr>
<tr>
<td>AFC (depends on location of H₂S zone)</td>
<td>5% to 10%</td>
</tr>
<tr>
<td>Maingate corner</td>
<td>10% to 15%</td>
</tr>
<tr>
<td>Desorbed from face</td>
<td>5% to 10%</td>
</tr>
<tr>
<td>BSL and longwall conveyor</td>
<td>10% to 20%</td>
</tr>
</tbody>
</table>

The most difficult gas source to control is the AFC. Three options available for emission control in the next longwall panel are:

- To infuse the face with buffer solution;
- To wet coal on the face and AFC with buffer solution;
- To install a curtain along the face to segregate air.

A fourth option of reversing the ventilation is not currently approved under Queensland legislation.

Minor amounts of H₂S were recorded in outbye conveyor roadways but these were less than expected. A duct and hood arrangement was installed at the longwall conveyor transfer as a precaution.

Ventilation

The use of homotropal ventilation of longwall conveyor and an exhausting fan duct system on BSL proved successful. The use of an electric fan and smaller duct system to replace the venturi fans is being investigated for the next panel.

ACKNOWLEDGEMENTS

The authors acknowledge the contribution made to the control and study of H₂S at Southern Colliery by staff from CapCoal, Oaky Creek, Shell Coal, CSIRO and The University of Queensland and associated researchers. The opinions expressed in the paper are those of the authors and not necessarily those of these organisations.
### Table 1 - Personal Protective and Monitoring Equipment

**WARNING DEVICE (ENVIRONMENTAL MONITORING UNIT)**

Zellweger (neotronics) - Analitics minigas - 4 sensor gas unit, measuring:

- O₂ - high and low
- CH₄ - % by volume
- CO – ppm
- H₂S – ppm

**RESPIRATORY PROTECTION / EYE PROTECTION**


- Clear polycarbonate visor which offers full eye and face protection;
- Less than one ppm leakage when fitted correctly; and
- Full training given including negative pressure testing and individual practical use of mask and filters.

**FILTERS**

The two filters used are:

1. **Sundstrom 210/310 - Particle filter Class P3 - high standard filter**
   - Dust, viruses, bacteria, asbestos, smoke, aerosols;
   - Conforms to - AS 1716 – 1991; and
   - Licence No – 0766

2. **Combined with Sundstrom - Gas filter B2E2 High standard filter**
   - Class 2 filter when used with a full face mask and the 210/310 particulate filter;
   - Offers highest level of protection for apparatus of this type;
   - Conforms to - AS 1716 – 1991;
   - Licence No. – 0766; and
   - Filter exceeds Australian licensing requirements for H₂S.

Full face mask with this protection is approved for use up to 100 times TLV level or 5000 ppm which ever is lowest.
FINE COAL DWATERING BY COMPACTION IN BINS EXHIBITING RELIABLE DISCHARGE

A G McLean

ABSTRACT

This paper firstly reviews the requirements for container and hopper design for reliable discharge. Discussion is then directed to the evaluation of the consolidation stresses acting in both in both the cylinder and hopper sections during both initial and flow conditions. Here the vast difference, especially in vicinity of the bin outlet, between the initial or filling and flow conditions is highlighted. This vast difference has major design and operation implications. A further major implication is that all bulk solids must dilate to flow. The opportunity to simultaneously attain reliable discharge and partial moisture removal is also examined.

The selection of design parameters to maximise the consolidation stresses in both the cylindrical and hopper sections is detailed. In regard to the latter, it is recommended fine coal bins be discharged using low friction lined transition hoppers with large outlet spans. Such outlets will discharge fine coal in a relatively compacted state. This compaction state should exhibit minimum water retention and short term moisture uptake.

The multiphase attributes of fine damp coal flow is then considered. Here it is noted, as a consequence of the dilation in the hopper negative or suction interstitial fluid pressures form. It is noted such pressures generate an adverse pressure gradient which significantly retards the discharge. One technique to conveniently eliminate the adverse pressure gradient is to install low pressure high volume air sparging to the hopper. It is suggested this air sparging will have the added benefit of effecting partial moisture reduction. This technique should prove far more reliable than post feeder compaction using relatively complex mechanical systems.

INTRODUCTION

General

To date fine coal bin storages have been largely designed to achieve maximal storage capacity at minimal cost. Such bin storages have resulted in generally adverse bin operation characterised by relatively small live storage capacity and unreliable discharge or in extreme cases complete flow obstructions. To allay these operation deficiencies greater attention has been devoted to designing bins for reliable discharge. This reliable discharge is possible by applying the principles of modern bulk solids handling design. However, due to increased economic pressures coal producers are increasingly faced with the problem of storing increasingly finer and wetter coal into bin storages (typical of larger capacity). Furthermore their customers are demanding dryer coal exhibiting favourable flow properties. Noting the coal industry is faced with these acute demands, it is appropriate to examine the design and operation of coal bins to simultaneously effect both reliable discharge and partial dewatering of fine coal products including fine reject material. The possibility of achieving these seemingly counter opposed goals will first be discussed in reference to gravity discharge bins. This initial discussion will highlight that reliable discharge of fine wet coal requires proper attention to the multiphase nature of the stored material and awareness that all particulate materials must dilate to flow in and more importantly from converging channels. The same will also highlight that existing bin designs generate minimum opportunity for simultaneous dewatering and reliable discharge. In fact it will be shown that some existing bin geometries completely overlook the opportunity to effect partial dewatering simultaneous to reliable discharge. Discharge from such bins, when compounded by poor operation practices, typically exhibits high moisture content. Most notable of such situations is rainfall collection in partially discharged funnel flow bins. On the contrary it will be shown...
that partial dewatering is possible by installing air sparging systems into fine coal storages. The latter part of the discussion will highlight the potential of bin feeder system designs to effect both reliable discharge and partial dewatering.

- In this discussion typical bin geometries will be adopted namely a bulk solid container comprising a cylindrical section atop a converging section, Fig. 1. The latter converging section may be either of axisymmetric or planar form. It is also expected the reader is familiar with the concept of mass flow channels. Such channels are now appropriately much used because of their superior flow characteristics (Arnold et al, 1981), especially when storing and discharging fine cohesive coal.

In regard to dewatering, it is expected water will be displaced from the interstitial particle spaces whenever the particulate bed is subject to high consolidation pressures with decreasing interparticle spacing. On the other hand a solids bed is referred to as dilating whenever the interparticle spacing is increasing. Such dilation occurs when bulk solids experience decreasing magnitude consolidation stresses.

Requirements for reliable discharge

Reliable discharge of fine coal from a converging channel or hopper requires:

- the flowing coal is yielding throughout the bulk
- slip is occurring along the walls
- existence of an arched stress field (Arnold et al, 1981)
- opportunity for the flowing bulk solid to dilate
- the magnitude of the gravity generated body forces must exceed any interstitial fluid phase adverse pressure gradients (Reed, 1973; Johanson, 1979; McLean, 1984).

Fortunately, procedures now exist to design converging channels to generate reliable discharge of cohesive bulk solids subject to (near) incipient flow conditions. Basically design of the hoppers for reliable design involves selecting (Arnold et al, 1981):

- the outlet span (B) to be sufficiently large to cause collapse or yielding of the cohesive bulk
- selecting the hopper wall slope (α) to be sufficiently steep to attain slip along the walls
- hopper walls with low sliding friction characteristics
- hopper details to ensure the transport volume increases in the direction of flow (ie diverging feeder side skirts, flow regulation gate stress relief, provision of overhangs at joint lines, etc.).

CONSOLIDATION STRESSES IN BINS

General

Equations for predicting the consolidation stress variation with depth in typical bins or bulk solid containers for both initial and flow conditions are now well known (Arnold, 1981). Such variations, for a typical bin geometry, are depicted in Fig. 1.
Parameters Governing the Extent of Consolidation in Bins

It can be well appreciated that the magnitude of the consolidation stresses to which the flowing bulk solid is subject depends on:

- solids bulk density and compressibility noting for fine coal the variation of bulk density with consolidation is adequately described by

\[ \theta = \theta_0 \left( \frac{\sigma}{\sigma_{10}} \right)^b \]  

(1)

or in terms of specific weight \( \chi \)

\[ \chi = \theta g \]  

(2)

where

- \( \theta \) bulk density, \( \text{kgm}^{-3} \)
- \( \theta_0 \) characteristic empirical bulk density at stress level \( \sigma_{10} \), \( \text{kgm}^{-3} \)
- \( \sigma \) major consolidation stress, kPa
- \( \sigma_{10} \) empirical major consolidation stress, kPa
- \( b \) empirical exponent, -
- \( \chi \) bulk specific weight, \( \text{kNm}^{-3} \)
- \( g \) gravitational acceleration, 9.81 ms\(^{-2}\)

- internal friction angle (\( \delta \))
- boundary friction angle (\( \varphi_{gw} \))
- cylinder diameter (\( D \))
- cylinder height (\( H_c \))
- form of the channel (ie whether axisymmetric or planar flow)
- stress field factor (\( K \))
- whether the bin contents are inactive (filling, initial or peaked) or active (flow or arched)
• the time of storage
• bulk solid moisture content
• solids and fluid phase specific gravities
• permeability to fluids flow

Cylinder consolidation stresses

In the cylinder the consolidation stresses are adequately predicted by the much used Janssen variation given by

\[ \rho_1(z) = \chi z_0 \left(1 - e^{-z/z_0}\right) \]  

where

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>(\rho_1(z))</td>
<td>the major vertical consolidation stress, kPa</td>
</tr>
<tr>
<td>(z_0)</td>
<td>parameter defined by (z_0 = D/4 K_j)</td>
</tr>
<tr>
<td>(D)</td>
<td>channel inscribed diameter, m</td>
</tr>
<tr>
<td>(\lambda)</td>
<td>boundary friction coefficient = (\tan \theta_w)</td>
</tr>
<tr>
<td>(\theta_W)</td>
<td>boundary Coulomb friction angle,°</td>
</tr>
<tr>
<td>(K_j)</td>
<td>stress field factor relating the orthogonal stress components</td>
</tr>
<tr>
<td>(z)</td>
<td>depth from the effective surcharge level, m</td>
</tr>
</tbody>
</table>

An examination of equation (3) suggests the consolidation stresses are large whenever the cylinder variables \(\chi, H\) & \(D\) are large and \(\theta_W\) is small.

Consolidation stresses in hoppers

Since most bin storing fine coal involve a combination of a cylinder atop a hopper or converging section it is appropriate to examine the consolidation stress variation with depth. Obviously this much used arrangement facilitates convenient concentrated discharge, usually through a central symmetrical outlet, of the contents to downstream unit processes. However, much to the surprise of most coal operators, whenever a converging channel is used to discharge bulk solids the stresses acting on the discharging material in the vicinity of the hopper outlet are low in magnitude (typically 3 - 10 kPa) as shown in Fig. 1. In fact the variation in the consolidation stresses in the converging section of a bin is adequately described by equation 6.3.5(3) in (SAA, 1996) namely:

\[ \rho_1(z_h) = K \left(\frac{\gamma h_h}{(j - 1)} \left(1 + \frac{h_h - z_h}{h_h}\right)\right) \]  

where

<table>
<thead>
<tr>
<th>Symbol</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>(\rho_1(z_h))</td>
<td>consolidation stress at depth (z_h) in the hopper, kPa</td>
</tr>
<tr>
<td>(h_h)</td>
<td>height of the hopper from the apex to transition, m</td>
</tr>
<tr>
<td>(z_h)</td>
<td>depth below the cylinder to hopper transition, m</td>
</tr>
<tr>
<td>(\gamma)</td>
<td>unit weight of bulk solid, kNm(^{-3})</td>
</tr>
<tr>
<td>(p_{vit})</td>
<td>container transition mean vertical pressure, kPa</td>
</tr>
<tr>
<td>(j)</td>
<td>stress field parameter, defined in Clause 6.3.5 (SAA, 1996)</td>
</tr>
<tr>
<td>(K_{hf})</td>
<td>normal pressure ratio for hopper based on powder mechanics principles</td>
</tr>
<tr>
<td>(\delta)</td>
<td>effective angle of internal friction,°</td>
</tr>
<tr>
<td>(\alpha)</td>
<td>hopper angle, °</td>
</tr>
</tbody>
</table>

\[ K = 0.5*(1+k_{hf}) \left(1 + \sin(\delta)\right) \]  

\[ \rho_1(z_h) \]  

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For initial conditions in the hopper it may be noted the stress field parameter $j$ exhibits a magnitude less than unity and in fact approaches zero when incompressible bulk solids are stored in channels exhibiting minimal boundary flexibility. On the other hand during flow $j$ is large and positive with typical values exceeding 3. Such values for $j$ cause eqn (4) to reduce, in the vicinity of the hopper outlet, (ie for $z_h$ large) to

$$\rho_1(z_h) \propto K \left( \frac{\gamma (h_b - z_h)}{(j-1)} \right)$$

Equation (6) predicts the well known radial stress field variation for converging channels. It is now appropriate to examine the implications associated with eqn (6). The most significant implications, relating to the magnitude of the consolidation stresses at the outlet of a converging channel during flow are:

- strong dependence on the outlet span, $B$, (ie $B = 2 (h_h - z_h) \tan(\alpha)$);
- relative independence of the extent of consolidation effected in the container cylinder;
- strong dependence on the local magnitude of the bulk density;
- strong dependence on local channel parameters via the factor $j$ including wall slope ($\alpha$), wall friction ($h_b$), channel form ie planar or axisymmetric (via factor $c_h$ in SAA(1996) and a lesser extent on the effective angle of internal friction ($\phi$)).

**OPPORTUNITY FOR PARTIAL DEWATERING AND COMPACATION**

**Cylinder**

The existence of large consolidation stresses in the vertical bin section suggests the possibility to compact the bulk solid and hence mechanically remove some water from the same by installing suitable permeable walls in the vicinity of the cylinder hopper transition as depicted in Fig. 2. However, to date this opportunity has largely being ignored due to the hitherto unavailability of suitable low cost porous non blinding materials, high capital cost and for other reasons. However, the increased availability of sintered low friction wear resistant porous material suggests increased opportunity now exists to exploit this gravitational consolidation for dewatering. It may be noted the actual quantity of water removed would be dependent on the time of storage and be promoted by application of low magnitude vacuum pressure differentials across the permeable membrane. It may be noted since the permeability to water flow decreases with increasing depth in the cylinder it is usual to observe upward moisture migration (provided the particle density exceeds that of the fluid phase). In cases involving storage of extremely wet coal installation of bin top surface drainage facilities may also be warranted.

![Fig. 2 - Dewatering using permeable bin walls at the bin transition](image)

The existence of the large consolidation stresses at the base of cylindrical containers further suggests the opportunity for dewatering using live bottom bin arrangements as depicted in Fig. 3. To date such bin arrangements have received
minimal application due to high capital cost and maintenance difficulties. A number of operation concerns have been allayed somewhat with the current generation of live bottom bin discharge units. In these arrangements the actual moisture reduction would be effected via permeable membranes situated in the container base. The extent of actual water removed or minimal water uptake on discharge may also be enhanced by incorporating a compaction zone at the discharge end of the extraction device.

**Fig. 3 - Live bottom bin storage and dewatering system**

Typical bin cylinder hopper combination

Regarding the channel form, it is found planar channels (as a result of their more favourable flow characteristics (Chamberrlain, 1986)) generate stresses approximately double those occurring in axisymmetric channels (Arnold et al., 1981).

Noting this compaction of discharging fine coal can be maximised by utilising transition hoppers with large outlet spans. With the availability of wide belt or apron feeders such arrangements, refer Fig. 4, should receive increased application for handling fine coal products and delivering the same in relatively compacted states. Necessary design details for wide large capacity belt feeders are presented in the papers by Winkler (1973) and Bridge and Carson (1987).

**Fig. 4 - Schematic of a typical transition hopper wide slot feeder combination**

MULTI PHASE FLOW CONSIDERATIONS

A further implication of equations (2) & (4) is that all bulk solids must dilate to flow. In fact this dilation of the hopper contents, in the case of bins with poorly selected hopper design parameters, may completely nullify the compaction effected to the bulk in the cylinder section. A further consequence of this dilation and the low permeability, especially to water flow of fine coal, suction interstitial pressure occur within the hopper during flow 3, 4, 5, 7 & 8]. Such suction or vacuum pressures, refer Fig. 5 severely retards the flowing bulk to the extent flow obstructions occur.

The actual extent of retardation can be accessed by considering the flow as subject to an effective gravitational acceleration ($g^*$) (McLean, 1984 and Carson, 1987) of magnitude defined by...
where $g_0$ is the actual gravitational acceleration (9.81 m/s$^2$), $\rho$ the bulk density of the flowing solid and $\frac{dp_f}{dz_h}$ the local interstitial pressure gradient.

A not so obvious implication of equation (7) is that for low density bulk solids even a low magnitude adverse pressure gradient will cause cessation of flow. Sluggish flow is also observed whenever fine low permeability bulk solids are discharged from bins with impermeable hopper walls. Here it is considered inappropriate to commit further discussion to the evaluation of the interstitial pressure gradient. However, it is most appropriate to consider what techniques can be employed to eliminate the existence of adverse interstitial gas pressure gradients.

The simplest and possibly the lowest cost technique is to supply low pressure high quantity aeration to the hopper contents (Lobb, 1995 and Lazzari, 1991) as opposed to supplying product deteriorating water injection. The extent and pressure of the aeration should be sufficient to generate a gradually varying monotonic pressure distribution exhibiting a favourable pressure gradient at the hopper outlet as shown in Fig. 6. In addition to effecting flow promotion this aeration or air sparging has the potential to simultaneously effect moisture reduction. The extent of this moisture reduction can be enhanced by preheating the air flow (particularly in situations where waste heat is available), increasing the air volume and increasing the air to coal contact time. Here the effective contact or residence time may be increased by effecting recycling of the hopper contents. A further advantage of recycling is the maintenance of the bin, especially the hopper, contents in an activated state. In this activated state passive or arched stress fields are maintained in the vicinity of the hopper outlet.

Fig. 5 - Interstitial void pressure variation generated during discharge of fine damp bulk solids

Fig. 6 - Air sparging during discharge to supply necessary dilation volume and effect moisture reduction
Here it should be noted the air sparging must be effected using low pressure high volume air supply during discharge. In no case should expensive high pressure high volume air supply be used as ratholes or air channelling will simply occur within the coal mass and in most cases coal flow will cease. It is usually most convenient to locate air injection devices at or immediately below the hopper transition. It should be noted air injection without bin dispatch is largely ineffective due to the high consolidation stresses applied to the hopper contents. Notably the applied consolidation stresses, during initial or filling conditions, causes the permeability of the hopper contents to be exceedingly low. This more or less eliminates low pressure air permeation through the same. The extent of air sparging may be increased by supplying the additional air via internal hopper fitments in combination with a wall permeation shelf or band. Even under the worst case scenario where the majority of air flow short circuits some beneficial moisture reduction would be effected along the actual and somewhat limited coal air interface. Here it may be noted use of internal hopper fitments ie hopper in hopper arrangements or inverted cone inserts also exhibit a flow promoting effect.

It must be noted here it is not the intention to completely fluidise the hopper bin contents as such fluidisation is grossly expensive as both the pressure and volume of the supplied air are large in magnitude. Such air supply variables contrast markedly to dilation air parameters. As one expects the air flow through the hopper contents under low intensity aeration would highly non uniform. This spatial variation results from the fact the air flow, in cohesive damp particulate beds, occurs essentially via channels formed in the bed and along boundaries of adjacent cohesive shear blocks. It is expected the bulk density of the actual shear blocks so formed would remain relatively high in response to the maximum consolidation experienced by the flowing coal. Here flow of these shear blocks is greatly enhanced by the air flow into the hopper. Most importantly this fragmented air flow even when low in magnitude will eliminate the formation of adverse interstitial fluid pressure gradients. Reiterating despite this flow being small in magnitude it is almost impossible to generate practical low pressure air flow through the hopper contents whenever the hopper contents are subject to a peaked or initial stress field.

A further advantage of air sparging this flow promoting effect is self correcting. Notably a sluggish flowing coal will be exposed to a longer period of air flow so facilitating increased moisture reduction of the flow bulk. This partial surface moisture reduction may generate significant improvements in the flow properties. However such increased aeration times must be well within the limits determined by the spontaneous combustion characteristics of the stored coal and which does not impart any significant loss of calorific value or coking property.

**OTHER TECHNIQUES TO EFFECT COMPACTION**

Before closing this discussion it is appropriate to briefly discuss other possible techniques to effect compaction of fine coal discharges from bins. Such compaction is possible using a number of mechanical devices which subject the flowing coal to compaction. These devices, when used in different situations particularly when handling dry materials, may be used as a flow promotion devices. Possible compaction devices hence include vibrated bin activators, large diameter screw feeders with compaction zones (Fig. 7) and moving hopper side wall systems (Fig. 8) to name just a few.

![Fig. 7 - Bin incorporating a combined feeding and compaction zone large diameter screw feeder](image)
Techniques to effect compaction using a screw feeder, refer Fig. 7, include decreasing the flight pitch or diameter in combination to increasing the hub diameter (Carson, 1987 and Bridge, 1987). Other techniques include omitting a screw flight, plug formation (missing flights at the end of the screw) and the use of a beach or contraction in the screw casing transport volume (Carson, 1988; Womack, 1987; Stuart-Dick, 1991). Here it may be noted the simplest technique to generate a beach is to arrange an inclination in the screw casing base. In all cases the transition must be smooth and impart no ledge or obstruction to the flowing fine coal.

In regard to the moving hopper side wall system, as shown in Fig. 8, the transport volume below the hopper outlet proper (selected for flow reliability to have a large span) only experiences a slight and controlled reduction in the transport volume so imparting controlled compaction of the discharging material.

As one observes compaction by these post hopper discharge systems is associated with retardation of the flowing material. Such retardation of coal flow is rort with danger as over compaction, system blinding and flow obstructions can easily occur. Unfortunately, once a flow obstruction or coal plug occurs flow can only be reinitiated by removing, usually manually, the hopper contents and obviously the blockage. It is also expected these systems would attract very high initial and maintainance costs.

![Fig. 8 - Moving hopper side wall feeding and compaction system](image)

**CONCLUSION**

This paper initially highlights the vast difference in bin consolidation pressure distributions between the initial and flow conditions. This discussion, especially of the consolidation pressures in the cylinder, suggests the opportunity for partial moisture removal from the bin contents. The extent of which can be maximised by suitable selection of bin parameters.

A further consequence of the predicted consolidation stress distributions is that bulk solids must dilate to flow. This dilation, in hoppers with inappropriately selected design parameters effectively nullifies the compaction imparted in the bin cylinder. The same also causes the formation of adverse interstitial pressure gradients which severely retard the flow of fine damp coal. Fortunately, these flow retarding two phase effects can be easily and at relatively low cost be eliminated by injecting low pressure high volume air into the discharging hopper contents. This flow enhancing air flow has the potential to simultaneously effect partial moisture reduction. The same is also shown to attract reliable and stable flow promotion in contrast to other mechanical based flow promotion and compacting devices.

The favourable flow characteristics associated with stable low friction lined transition hoppers feeding onto to wide belt feeders warrants widespread application for the reliable discharge of fine damp coal products. This application can be reinforced by selecting optimal hopper design parameters selected for each storage situation on a case by case basis and by installing air sparging facilities. Such improvements to the storage container will generate considerably enhanced container discharge characteristics without attracting excessive initial and maintainance cost. It is planned by the author, in association with a successful progressive engineering project company and a coal mine operator, to validate the claimed benefits in a future ACARP funded industry collaborative project. This project will also identify the benefit of...
optimising complete storage container design when handling adverse fine damp coal products. This application is in response to ongoing fine coal handling difficulties within the industry as reflected in the recently published ACARP research priorities (ACARP, 1996) an improved coal handling. Obviously this planned project will aim to prove this technology on a scale consistent with reliable coal flow, typical actual plant throughputs subject to actual operating conditions at minimal financial risk to the individual coal producer.

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Computational and Experimental Investigation of Spiral Concentrator Flows

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ABSTRACT

Spiral concentrators are used globally in the fine coal processing industry to segregate particles, by gravitation, on the basis of density and size. Consisting of an open trough that twists vertically downwards about a central axis, a slurry mix of particles and water is fed to the top of the concentrator. Particles are then separated radially as they gravitate downward. Since their introduction to Australia in the 1940's, the generic design has evolved largely by laboratory trial-and-error investigations of different prototypes. However, this approach has proven expensive and optimal designs have not been necessarily developed. Accordingly, Computational Fluid Dynamics (CFD) analysis has been used recently as an alternative method of investigation, to assume as is envisaged, a role in the design process. To date, CFD models have progressed to simulations of turbulent fluid flow on current production spiral designs, and are continuing to be adapted for inclusion of particles at realistic feed concentrations. To be able to use the models confidently however, laboratory experiments must also be performed to validate the predictions during the development stage. This paper reports the current findings of an ongoing CFD and experimental program applied to one spiral unit. Satisfactory quantitative agreement has been achieved for the fluid and particulate flow characteristics, and although further validations are appropriate, the model already possesses significant potential for use as a reliable predictive design tool.

INTRODUCTION

Spiral concentrators consist of an open trough that spirals vertically downwards in a helix configuration. Fine 100-1500 \( \mu m \) coal and waste rock particles are fed at concentrations of 30-40\% by volume, and are segregated radially on the basis of density and size as they gravitate downward. Ideally, light suspended particles (coal) travel to the outer trough regions whilst heavy particles (quartz) settle and move inwards toward the central column. To optimise this process, historical evolution of the design has been almost exclusively based on empirical trial-and-error developments of the appropriate geometry. However, this approach has proven to be expensive and is hence somewhat prohibitive.

Recognising that a fundamental understanding of the flow physics could lead to the development of a predictive model in the design process, Holtham (1990) investigated by experimentation the fluid flow patterns and particle movements on the LD9 spiral unit. This study has since formed the basis of an improved and continuing experimental program (Golab et al., 1997). In parallel with these programs, modellers this decade have turned from using predominantly analytical approaches (Holland-Batt, 1989) to Computational Fluid Dynamics (CFD) analysis which requires much less empirical input. Following the preliminary investigations of laminar fluid flow on the LD9 unit (Jancar et al., 1995; Matthews et al., 1996), the models have been extended to include both turbulence within the fluid phase and dilute particulate additions (Matthews et al., 1997; 1998).

Attempts to understand the mechanisms of spiral concentrator operation have been restricted by the complex character of the flow. Even when considering the fluid phase without inclusion of particles, the flow possesses a free surface, is thin with depths < 1 cm typically, and displays laminar to increasingly turbulent behaviour radially outwards with maximum velocities reaching 3 m/s. Moreover, a secondary circulation current in a plane perpendicular to the mainstream flow direction travels outwards near the free surface and back inwards toward the central column near the trough base. The complexity is further magnified by the mutual interaction of the fluid flow and particles, by interactions between the particles themselves, and by the obvious constraints imposed on the flow by the geometry.

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In this paper, the current experimental procedures and computational methods of investigation, used to quantify the relevant properties of the flow, are briefly described. To assess the reliability of the predictive model, validations for the LD9 unit at industrial flow rates are presented for the fluid flow. Numerical results for particulate additions at dilute concentration (to examine the influences of the hydrodynamic processes) and concentrations where particle-particle interactions become significant are then given. The practical significance and potential worth of the model are finally discussed.

INVESTIGATIVE PROCEDURES

Experimentation

Experimental investigations have been performed on the LD9 unit at industrial flow rates of 4, 6, and 8 m³/hr. Holtham (1990) measured flow depths using a computer controlled multi-point depth gauge equipped with conductivity probes. Flow rates were used to estimate mainstream velocity components for eight radial sections across the trough, and by the injection of dye tracers, the secondary flow was identified and measured. Golab et al. (1997) have measured instantaneous fluid velocities on the LD9 unit using Particle Image Velocimetry (PIV). In the PIV procedure, rhodamine particles are injected into the flow and illuminated with a laser beam, pulsed on-and-off at 1 ms intervals. Particle movements are recorded by videotape, the images digitised, and the velocities and positions of the tracers calculated by computer. In order to introduce the laser sheet to the flow, a 30° section of the original fibreglass trough is removed and replaced with transparent acrylic (Fig. 1). Due to the curvature of the trough, five separate windows are required through which the laser beam is transmitted and, by adjusting the position of the laser within each window, velocity measurements can be made over most of the trough region.

To investigate the effects of the hydrodynamic processes on particulate transport, Holtham (1990) measured first the discharge positions of quartz particles fed at dilute (0.3% by volume) concentration and fluid flow rate of 6 m³/hr. Three narrow size distributions with mean diameters of 75, 530 and 1400 μm were considered, and the percent recovery to each of 8 radial streams was calculated. Experiments using quartz with a broad (100-1000 μm) grain size distribution were also conducted at a volume concentration of 15%. This was done primarily to assess the significance of the Bagnold forces, as high volume fractions of particles (up to 50%) were found to accumulate in the inner region. Currently, the PIV technique for the fluid flow is being adapted to examine particles with material densities of that of coal and quartz.

Computation

The CFD model solves the steady-state Reynolds-averaged Navier-Stokes equations using the commercial software FLUENT. The Volume of Fluid (VOF) and RNG k-ε turbulence formulations are used to simulate the free surface transport and laminar to turbulence transition across the trough (Matthews et al., 1996, 1997). The LD9 computational domain is given in Fig. 2. At the inlet, the volume fraction of the water phase is arbitrarily “guessed” and the velocities specified to give a desired flow rate. The domain is divided into 35° sections with the outlet solution specified as the inlet conditions for the next downstream sector until fully developed flow conditions are obtained. Four walls bound the domain and the flow is essentially a duct flow that includes an interface between water and air. The mesh contains
respectively, 20 x 39-46 x 208 control volumes in the mainstream, depth-wise and radial directions, with cells clustered toward the spiral base.

Fig. 2 - LD9 computational domain with reduced cells for clarity

There are two main methods used to model particle flows using CFD. In the Lagrangian method, particles are treated as discrete entities and their trajectories calculated as they proceed through the fluid flow field. This approach yields detailed descriptions of the flow of individual particles, and is a more fundamental procedure to describe the acting hydrodynamic forces (pressure gradient, buoyancy, drag and virtual mass) and collisions with a wall. Alternatively, the Eulerian approach regards the particle cloud as another continuum able to interact and inter-penetrate with the fluid flow. Suited to particle flows at high concentration, interactions between particles are accounted for and mean particulate velocities and volume concentrations are calculated throughout the flow domain.

Matthews et al., (1998) have examined particle flows at dilute feeds using the Lagrangian method. Although these processes are believed to be the main contributors to particulate separation, two-way coupling effects of fluid-particle interactions and the interplay between particles have yet to be investigated. These additional factors have now been examined using the Eulerian approach, and the results of a preliminary investigation are presented in this paper. At the present time, location of the free surface must be assumed in the analysis although an approximate profile can be employed from the fluid flow solution. By making an adjustment for the high accumulation of solids that typically arise in the inner region, this limitation is not too severe as measured profiles are reasonably similar, particularly at small feed concentrations (Holtham, 1990). A fully predictive Eulerian-VOF free surface model in FLUENT will be available by mid 1998.

EXPERIMENTAL VALIDATION OF MODEL USING THE LD9 UNIT

CFD simulations of the fluid flow have been published previously (Matthews et al., 1997; 1998) and are found to be in satisfactory agreement with the experimental data (Holtham, 1990; Golab et al., 1997). In Fig. 3, the fully developed transverse profile with predicted and measured depth comparisons normal to the trough are given. The simulated mainstream velocity distribution is also depicted in the top part of Fig. 3. The discrepancy in the outermost region reflects clearly detectable air entrainment, currently unaccounted for by the model, which has the effect of “bulking” the flow. Caused by turbulent eddies escaping above the free surface and entrapping air as they return as droplets to the medium, the entrainment is likely to increase the water depth by 15-20% for the characteristic Froude numbers encountered in the outer zone (Matthews et al., 1998).
Fig. 3 - Predicted fluid profile at 6 m$^3$/hr (top) and depth comparisons with experimental data at 4 and 8 m$^3$/hr (bottom)

Perhaps the most encouraging aspect of the fluid flow simulations has been the satisfactory comparisons of mean mainstream velocity with the PIV measurements at arbitrary depths and radii within the flow. Predicted and measured mainstream velocities at 6 m$^3$/hr and 1 and 5 mm depths are given in Fig. 4. Also given in Fig. 4 is the classic structure of the secondary flow, first identified and measured using dye injections (Holtham, 1992). The PIV measurements and computer simulations reveal that the secondary flow is transient in nature with velocity fluctuations of at least the same order as the mean values. Characteristic values for the mean secondary motion are generally an order of magnitude less than the mainstream velocities and vary from 0.01-0.3 m/s.

Fig. 4 -Top: comparison of simulated and measured mainstream velocities at depths of 1 and 5 mm at 6 m$^3$/hr; bottom: Secondary flow in the outer trough at 8 m$^3$/hr
The development of a fully predictive, robust and accurate model for the fluid flow has been critical. Indeed, the evidence suggests that the detailed flow character primarily drives the separation processes (Holland-Batt, 1989). Having established a reliable model for the fluid phase, the next step is to examine the particulate flow at dilute concentration, based on the acting hydrodynamic forces using the Lagrangian method. Although experimental validation is not yet available, the nature of a particle-wall collision is assumed to obey the impulse equations describing sliding upon impact (Sommerfield, 1992). Sliding collisions are believed to predominate over non-sliding contacts as typically impact angles are small, and particle diameters much larger than the laminar sub-layer thickness (Matthews et al., 1998).

The Lagrangian procedure consists of injecting 100 particles of the same density and size uniformly across the trough on the free surface of the flow. The trajectories are then calculated until the radial positions of hydrodynamic equilibrium are reached, typically after 3-4 turns of the helix. Analyses were performed for 100-1500 μm coal and quartz particles with material densities of 1450 and 2650 kg/m³, respectively. The results suggest that the LD9 concentrator is able by hydrodynamic processes to separate particles in the somewhat narrow size range of 200-500 μm. Within this range, the classic pattern is observed with coal particles migrating to the outer regions whilst quartz accumulates at the small radii (Fig. 5). At < 100 μm, both coal and quartz are predicted to remain in suspension and move to the outermost zone; above 500 μm both particle types are found to accumulate within the inner regions.

At the higher flow rate, the upper extent of particulate separation is extended from 200 to 500 μm as magnitudes of the secondary velocity are significantly stronger; at 4 and 8 m³/hr, the predicted maximums of u are 0.13 and 0.22 m/s, respectively. However based on the hydrodynamic processes alone, no apparent single definitive radial cut exists between the desired product and waste material (Fig. 5). For example at 8 m³/hr, optimum separation of 200 μm particles would be achieved at -0.20 m from the central column. Conversely, poor separation at 0.20 m would occur for particles of 500 μm diameter, for which segregation should be performed at ~0.06 m radius according to the analysis. Hence, if it is accepted that coal concentrators are designed so that the hydrodynamic processes are the dominant factors controlling separation, then the simulations suggest that the LD9 performance could be improved.

Differences have been found when the Lagrangian radial distributions for quartz are compared with the measurements obtained by Holtham (1990) at dilute 0.3% volume feed. First, Holtham discovered that approximately half of the smallest particles of < 100 μm diameter accumulated within the inner region, although the rest migrated to the outer radii as predicted by the model. Second, particles with progressively larger size above 500 μm migrated outwards to accumulate predominantly within the central portion of the trough. Accordingly, the Eulerian method was employed to simulate...
Holtbam's experiments of 75, 530, and 1400 μm quartz feeds at 0.3% volume concentration and fluid flow rate of 6 m³/hr. Significantly improved comparisons with the measured radial distributions of particles have been found, and are depicted in Fig. 6.

![Graph showing comparisons of measured and predicted Eulerian radial distributions of 75, 530, and 1400 μm quartz particles fed at 0.3% volume feed and fluid flow rate of 6 m³/hr.](image)

Fig. 6 - Comparisons of measured and predicted Eulerian radial distributions of 75, 530 and 1400 μm quartz particles fed at 0.3% volume feed and fluid flow rate of 6 m³/hr

Considering first the finest particles of 75 μm diameter, the Eulerian model correctly predicts that a large proportion of particles move to the inner region. This reflects an even injection of particles with depth whereby those near the trough base in the low velocity region move inward by gravity and the secondary flow. Conversely, all particles in the Lagrangian analysis were placed initially on the free surface. At diameters above 500 μm the particles are predicted to move progressively further outward toward the central region of the trough. This trend has been measured in the experiments of Holtbam (Fig. 6), occurs under industrial operating conditions (Holland-Batt, 1989), and can be attributable primarily to Bagnold's dispersive force that is more prominent at greater particulate diameters and volume concentrations (Holtbam, 1992).

**PRACTICAL SIGNIFICANCE OF MODEL DEVELOPMENT**

Currently, CFD is not used in the design process of coal spiral concentrators. The accurate and robust nature of the model however, suggests that CFD could potentially assume, at some level, a significant role. For example, the detailed fluid flow solutions of transverse profiles and velocity distributions (without even considering particle inputs) may be of use to an experienced design engineer. The ability to extract detailed information above that available through experimentation, can also lead to further fundamental insights into the behaviour of the flow. Perhaps more importantly, because the hydrodynamic processes are believed to be the dominant factors driving separation, the Lagrangian analyses can be used as a preliminary measure in assessing the potential performance of any given prototype. Such information would probably be useful without considering the effects of particle-particle interactions, which the Eulerian analysis suggests, should be considered in the complete analysis.

Fundamental information of the fluid flow properties and particle trajectories can be attained much faster computationally than experimentally. To obtain a fluid flow solution for an arbitrary geometry requires 48-72 hours using a single processor of a 120 MHz UNIX server. A further 12 hours are required to perform the dilute particulate flows for a range of particle types using the Lagrangian method. In contrast, many weeks would be needed to achieve experimentally the same detailed information for the fluid flow using the procedures outlined in this paper. Moreover, the computer hardware employed in the investigation is by no means state-of-the-art, so that continuing rapid technological developments are likely to see costs of available computers decline significantly. Indeed, comparative fluid flow solutions presented 18 months ago (Matthews et al., 1996) required two weeks of processing on a HP 7000 series work station. It is then feasible that projected solutions in the near future will be affordably accomplished within several hours.

Although the fluid flow predictions are generally excellent and the particulate simulations encouraging, further improvements to the model and experimental validation are necessary. The most notable numerical addition will be combination of the VOF and Eulerian models to fully simulate the free surface particulate flows at realistic feed concentrations. Experimental validation will focus upon the particulate flow, namely the measurement of particle velocities, dynamics of particle-wall collisions (including restitution and friction coefficients) and modes of particle
transport. By continuing to implement realistic physical aspects of the flow into the model, and as the costs of hardware continue to decline, CFD will gain greater potential for use in the design process. Indeed, there may already be economic benefits in using the current state of model development, reported in this paper, as a design tool at some level. Ultimately, it is envisaged that the CFD model will eliminate most of the competing designs during the prototype development stage, to perform physical testing on one or two preferred configurations for the given application.

CONCLUSIONS

This paper has reported the current findings of a continuing experimental and computational study of spiral concentrator flows applied to the LD9 unit. The aim of this study is to develop and validate a CFD model that can be used in the design process. Accurate results for the fluid flow characteristics have been simulated and the model is robust enough to be used for arbitrary geometric configurations. Particulate analyses have been performed at both dilute and high concentrations to examine the influences of the hydrodynamic processes and particle-particle interactions. Although further extensions to the model and accompanying experimental validation are required, particularly with respect to the particulate flow at realistic feed rates, predictions of the model are encouraging. Indeed, the current state of model development is likely to be of some use at the design level, and this will become increasingly the case as future costs of computer hardware decline and improvements to the model are implemented.

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Improving Fundamental Stockpile Management Procedures

P Keleher¹, D Cameron¹ and M Knijnikov¹

INTRODUCTION

Coal Quality management and the control of the flow of coal through complex mining preparation and transport phases of standard mining operations has assumed greater importance over recent years.

Considering the history of the Australian industry from 1970, it is significant to note the increase in production levels and the inferred increase in focus on quality control - both of which drive the management of product quality into a position of greater importance. Quality management is one fundamental of the industry coming under increased pressure.

HISTORY

The Australian coal industry has grown from a saleable coal production level of 45 mt per annum in 1970 to 192 mt per annum in 1996.

![Australian Coal Production Graph](image)

Fig. 1 - Source Australian Black Coal Statistics

Emerging from that increase is the need for greater focus on stock control. Issues such as optimum stockpile size, stockpile turnover period, stock level fluctuation and timely stock management have all assumed greater significance.

The practices of the past are no longer sufficient to cope with the needs of today’s industry where a changing environment of higher quality standards, more sophisticated quality control and total quality management is driving the quality issue to being one of the more pressing aspects of the coal mining industry. An aspect requiring a review.

Further evidence for the increased pressure on quality management is the gradual reduction in saleable coal as a proportion of total coal production. Saleable coal production comprised 87% and 80% of raw coal production in 1970.

¹ Carbon Consulting International Pty Ltd, Queensland
and 1996 respectively. One of the contributors to this change has been the increased pressure from customers for a cleaner, more consistent and better presented product.

Compounding the situation, quality specifications are now required on more parameters and to a greater level of precision.

As a result, the perceived importance of quality in the coal industry has increased dramatically over the past 27 years. In 1970, ash, energy and other basic parameters were almost alone in consideration of quality indications. (Fig. 2)

Gradually the concentration of parameters has increased not only for the metallurgical coal market but also for the thermal market. As coal utilisation technologies evolve so too is the need for greater precision in the management and tracking of coal evolving and with the imminent introduction of gasification and new steel making technologies this trend is set to increase.

![Fig. 2 – General Observations On Quality Requirements](image)

**THE COSTS**

Historically, quality issues have been addressed using standard visual and recording techniques. The Quality officer has routinely visited key stockpile and production points and recorded relevant data. These techniques have been improved and adapted to accommodate change but in general the same tools are applied today for the same purpose. Meanwhile the requirement for a precise tracking tool has become more acute.

Obviously there are shortcomings to the existing methods, and these shortcomings emanate from sources including the heterogeneous nature of coal, increased production levels and more stringent specifications.

The obvious costs are;

- Loss of quality definition on stockpiles;
- Lack of precise quality control; and
- Continuous sampling to track parcels of discreet quality.

Each of the above causes either direct or indirect costs to an operation.
A less obvious but very significant cost is the adoption of a cautious and reactive approach to cargo preparation. The Quality Control Officer, in the absence of reliable timely data, ensures the cargo is well within specification. The loss of yield, inefficient machinery use and loss of valuable management time sacrificed to short term reactive measures are rarely quantified, but are recognised by industry as being substantial.

Even with the abundant downstream analysis, the final result can still be mediocre as depicted on Fig. 3

![Fig. 3 - Costs presently associated with cargo assembly](image)

Coal markets however, are characterised by the need for a uniform product to particular specifications. Inability to meet these specifications results in financial penalties or cargo rejection, depending on the severity of the quality non-conformance.

Knowing the location of individual parcels of product with their own individual quality characteristics is a fundamental requirement of stock management.

**A SOLUTION**

In response to this fundamental need, industry has employed a variety of tools and technologies for particular applications.

These include:-

- High speed computers;
- Increased sophistication in database software;
- Improved radio communications;
- On line analysers and weightometers; and
- Reliable real time surveying techniques.

Individually, each of these benefit the quality recording methods in specific applications. However, individually they do not provide a real time quality management tool to meet the tracking needs for complex stockpiling situations.

By combining the above tools and techniques the tracking of quality through a coal flow system in real time becomes a reality.

QMASTOR© is such a system.
SYSTEM OVERVIEW

Essentially QMASTOR© has three elements:

a) A Remote Positioning System

Differential GPS technology is used to monitor the movement of coal through all 'active' stockpiles using satellite receivers placed directly on the working plant (loaders, trucks, dozer). The receivers automatically transmit the position of the machinery, and consequently the coal parcel, to a central computer via a radio telemetry link. This 3-dimensional positional data is transmitted many times per minute to facilitate the location of coal with precision appropriate to the task.

b) A Central Computer

The central computing system emulates a customised 3-dimensional stockpile model and is comprised of several individual elements including a relational database, machine tracking software, stockpile mapping, 3D visualisation routines and production reconciliation tools. It has the capacity to receive a variety of data types including both planning and production information. This data is received either as static files or real time data strings. For example, quality data from an on stream analyser can be fed to the system, matched with the GPS data and compared with the master production schedule thus providing a continuous real-time quality profile of the required stockpiles.

The status of stockpiles can be reported either on screen or on a report format by:

- sub zone;
- longitudinal or lateral section;
- individual stockpile composites.

c) An Optimisation Model for Coal Reclamation

Coal being reclaimed from stockpile must meet pre-defined quality specifications. The optimisation model automatically determines a reclamation schedule to meet these specifications in the most economic fashion based on the value and quality of available coals. As coal is reclaimed to the blend, monitoring via DGPS continues, maintaining an up to date status of the various stockpiles. A schematic of the Total System Configuration is shown in Fig. 4

![Total System Configuration](image-url)
Whilst QMASTOR© is a product in its own right, it can also be integrated with other quality management tools, providing a real-time update facility for the planning of coal flow systems.

**ADVANTAGES**

The provision of accurate stockpile information allows management to make well informed timely decisions. Specific advantages include

- Confidence that a cargo is not only in specification but also optimised for quality and cost parameters;
- Reduced operating costs associated with survey and traditional quality control methods;
- Stockpile tracking on a real time basis;
- Facility to analyse and backtrack machine usage;
- Immediate survey control; and
- Reduction/elimination of downstream sampling and analysis.

The need for volumetric surveys (either ground based or aerial) is eliminated by the continual GPS record of the stockpile surface. QMASTOR© techniques achieve a level of control unmatched by industry alternatives.

The need for double handling of product due to quality uncertainty is eliminated and the need to sample is reduced. In addition, the user can clearly demonstrate to their customers a dedication to total quality management and state of the art quality control; parameters which may soon be standard in sales contracts.

**APPLICATIONS**

QMASTOR© can be applied in any material flow system where there is a need for accurate and timely stockpile control. For the coal industry this could be:

1. at the port - to ensure optimum use of coal whilst consistently meeting specification
2. at the washplant - where clean coal stocks require careful maintenance to ensure client needs are met
3. prior to the washplant - for raw coal blending into the plant to maximise yield.

Other applications in the coal industry may include raw coal haulage and blending prior to the washplant. Outside the coal industry potential clients in the transport and bulk materials industries are investigating the suitability of the system to their particular needs.

**RESULTS**

Fig. 5 demonstrates the advantage of QMASTOR© in comparison to a conventional method of tracking stock.

1. **Background**

   To compare QMASTOR© with a conventional technique, a population sample of 30 vessels is selected. The method relies on a simple approach of comparing quality characteristics analysed by either the conventional or the QMASTOR© method and comparing those results with the superintending results through automatic sampling of the vessel. The superintending results are taken as correct.
The Conventional method is based on pre-shipment analysis. During the transportation and stockpiling from the clean coal area to pre-shipment area regular samples are taken from the mobile equipment.

Interpretation of the results is based on standard statistical techniques.

2. **Analyses of conventional result.**

In the diagram, the curve with large dots represents the differential on ash basis between the conventional technique and the automatic sampler. As can be seen from the diagram, approximately 23% of vessels had a differential more or equal to -0.35. Similarly, 3% of vessels have -0.2 differential result, going up to 18% for -0.1, and so on. For this case the distribution is almost random, and the curve is roughly approximated by a simple line at 0.10.

3. **Analyses of QMASTOR® results** The QMASTOR® results are represented by the curve with asterisks. The curve approximates the Normal distribution of probabilities. For example, differential zero results between QMASTOR® and automatic sampling are achieved for 24% of vessels. In contrast, the theoretical probability equals 26%.

Shown as background to the QMASTOR® curve is the theoretical histogram for the precision of <0.35 and >0.35. For this normal distribution, the rule of 3 sigmas (standard deviation) is applied. For example, comparison of QMASTOR® and conventional method probabilistic results for 1 sigma gives ~68.27% of vessels and ~33.3% respectively.

4. **Comparison of results reached by QMASTOR® and conventional methods.** The Australian standard on methods of analysis and testing of coal and coke gives figures on repeatability and reproducibility of 0.15 and 0.25 respectively. In probabilistic terms, QMASTOR® improves this result 2 times (~68.27% and ~33.3%). Similarly, QMASTOR® gives considerably improved results across the whole spectrum.

**CONCLUSION**

Fig. 5 clearly demonstrates, QMASTOR® has the following advantages over conventional methods of stock management:

- It reduces the variance of shipped product;
- It provides confidence in the planning of cargoes quality; and
- It is a controlled management tool (as demonstrated by the Normal probabilistic distribution) which gives the operator control over quality.

QMASTOR® provides companies with a system to consistently strike quality targets with accuracy and confidence. It also forms a basis for continual improvement of not only the quality of the product, but also the efficiency of the total operation. This is a versatile product and its application will assist in resolving one of the coal industries fundamental challenges.

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QMASTOR - *

CONVENTIONAL METHOD - O

NORMAL DISTRIBUTION - [shade]

Fig. 5 – Comparison of Results
Joint Industry Planning Platforms for Coal Export Supply Chains

R Bridges¹, N Buckley², J Goodall³, D Seeley⁴

ABSTRACT
Improving the performance and reducing the costs associated with export logistics chains is critical to the competitiveness of export coal mines. The fundamental practices associated with the use of export logistics chains made up of mine, trucking, rail and port operations are being challenged by the advent of third party operators on rail systems and the use of the Internet.

Whilst individual mines can improve their processes to drive down their mining costs, they face major challenges in their endeavour to improve the performance of export logistics chains and reduce the significant logistics costs of moving coal from the mine to export ships, via the shared infrastructure of rail systems and ports. There is an increasing realisation that global competition is not only between mines but between coal export regions that are defined by their rail system and ports infrastructure.

The development and use of a joint industry planning platform for the export logistics chains of the Western Australian Grain Industry has demonstrated that an industry facing significant restructuring and increased competitiveness can achieve major throughput and cost reduction gains when stakeholders in export logistics chains share key planning information using the Internet and state of the art planning tools.

Joint industry planning platforms for export logistics chains are being considered or are at initial stages of development for a number of Australasian coal export logistics chains. This paper reviews the development of a joint industry platform for the WA Grain industry and reports on the state of development of similar planning platforms for the export logistics chains of the Illawarra, Hunter Valley, SE Queensland, Blackwater / Moura, Goonyella and Mt Isa / Townsville export coal regions.

This paper addresses the key components of joint industry planning platforms, the key information that should be shared, the use of the internet and information servers, and the contractual structures required to enable stakeholders of an export logistics chain, who are competitors or potential competitors, to work together to improve the competitiveness of a coal export region.

"Simple schedules for complex coal supply chains,"

INTRODUCTION
Improving the performance and reducing the costs associated with export supply chains is critical to the competitiveness of export coal mines. The fundamental practices associated with the use of export supply chains made up of mine, trucking, rail and port operations are being challenged by the advent of third party operators on rail systems and by the use of the Internet. Moreover, there is an increasing realisation that global competition is not just between mines but between coal export regions that are defined by their rail system and ports infrastructure. Hence as Ohmae (1995) has pointed out, these supply chains form the estuaries of entire economic regions. Further, Ohmae (1990) makes the point, that it is these regions which are actually in competition in a borderless economy.

A client of a coal supply chain wants the most effective supply of high quality product at the best price possible, from anywhere in the world. This client will spread their risk over a number of suppliers in order to avoid over-exposure to any single one. The various risk factors typically include:

¹ InterDynamics Pty Ltd
² Queensland Rail
³ Westrail
⁴ PhD, InterDynamics Pty Ltd
• regional politics (as it affects stability of supply);
• industrial relations (track records of steady supply, e.g. Hunter Valley);
• weather influences (e.g. havoc caused by El Nino);
• changes in rail/road infrastructure (affecting reliability and performance);
• performance of the ports involved (e.g. Newcastle, Vancouver BC.
• reliability of mine production; and
• the nature and the development stage of the ore deposit.

The coal buyer’s requirements are shaped by the buyer’s own clients, and a provision of steady supply, say to power stations. This supply will be made as economic as possible, without extra burdens such as overly large stockpiles. The buyer considers orders for specific quanta of time, what shipment sizes, shipping delays, cargo assembly delays, etc. are like within that quantum, e.g. a month. Hence, they minimise the drain on their financial position by taking into account their risk covers and working capital.

The buyer will negotiate a variety of contracts with its suppliers, contracts which take into account the following:

• long or short term nature;
• available pricing;
• necessary lead times;
• roughly uniform supply flow; and
• a positive on-demand arrangement with minimal cost penalties.

From the client or buyer’s whole-of-business perspective (Fig. 1), they wish to know the answers to the following questions about a given export chain:

• How long does it take to move the ore from mine to port?
• In a given campaign, how many tonnes/day can be supplied at port, under what assumptions about the transport infrastructure and port loading?
• What is the nominal capacity of the chain on a daily basis?
• What assumptions occur around the perceived performance of a chain (e.g. that ships are delayed at least ten days at the port of Newcastle)?

Such being the case with the clients of a coal supply chain, it behoves the stakeholder in a given chain to determine how their own supply and production process serves these clients, and to find a behaviour and performance which will effectively provide the best revenue by taking the client’s needs into account. This entails as well the transport performance of the entire supply chain, and it is herein that some of the best opportunities lay for improved competition. These improvements are with the opportunistic synergies which exist with other stakeholders, and it is here that benefits of joint industry planning platforms come to the fore.
Joint industry planning platforms

The development and use of joint industry planning software for the export supply chains of the Western Australian Grain Industry has demonstrated that an industry facing significant restructuring and increased competition, can achieve major throughput and cost reduction gains using this approach. Such joint industry planning is made possible when stakeholders in export supply chains share key planning information using the Internet and state-of-the-art planning tools. This of course applies equally to coal export.

Joint industry planning is based upon the capability to visualise and understand how our organisations and systems work as a whole. Hence, there is a natural flow from a concern for the effectiveness of various parts of an organisation to the effectiveness of the whole-of-business. Once focus is brought to bear on export supply chains, this concern naturally extends to the performance of the whole-of-regional-industry. Again the opportunity to open up to a wider perspective beckons, by considering the supply chain relationship to its client base, onto the relevant global commodity market.

Joint industry planning software for export supply chains are also being considered or are at initial stages of development for a number of Australasian coal export supply chains. This paper addresses the key components of joint industry planning platforms, the information provisions, and the contractual structures required to enable the stakeholders of an export supply chain, who are often competitors or potential competitors, to work together dynamically. This enables them to improve the competitiveness of a coal export region. In addition, it describes the various capabilities which joint planning platforms make available.

Hence, the construction and use of joint industry planning platforms involves a gradual extension of awareness of the business milieu, through the whole-of-system visualisations which software can support (Fig. 2).
Once a whole-of-system view of the operations of the supply chain becomes available in software, stakeholders are enabled to view the whole-of-business relationships between them in action. Then opportunities for effective information sharing and for specific dynamic stakeholder relationships in the planning platform, become clear amongst themselves. Next, access management is applied to the key resource, often the rail infrastructure or a port infrastructure, and a variety of forward capabilities of the entire chain, become truly possible. When an effective means to steward this global view of the export chain is achieved, then successful world competitiveness becomes realisable and sustainable.

Fig. 2 - Whole-of-system planning

PERFORMANCE IMPROVEMENTS IN THE SUPPLY CHAIN

Rail system managers and planners can have difficulties to determine the actual capacities of complex rail networks because of different assumptions of what to include in their calculations, and a lack of tools which adequately accounted for dynamic variations. It may not be straightforward to determine what kinds and quantities of rolling stock should be included for consideration, what mine production regimes and working practices should be assumed, what interfering traffic is assumed as background, and what equipment and rail maintenance to account for. Furthermore dynamic variations due to irregular production, irregular ship arrivals, equipment breakdown, and track speed reductions for maintenance are part of the real operations of the railroad, and significantly impact available capacity. It is for this reason that software tools have been developed for various railroads and other companies, which provide a realistic reference for the productivity of the current system. Bell (19) has summed up these matters as follows: '...we are now able to monitor all aspects of the chain that are relevant to our business. As an example, this has allowed us to achieve much greater flexibility and increased utilisation from our locomotive and wagon fleets.' — Michael Bell, Westrail

Reference capacity models

Reference capacity models (RCMs) have been developed for Queensland Rail's coal and minerals export chains, using animated management tools (Seeley et al 1995) which provide a benchmark for the current throughput capacity of operations, against which all potential improvements can be assessed. Often the RCM is used to identify the performance bottlenecks and the constraints which need the most attention that are implicit in the current system. Such identification focuses upon how resources can be best applied to yield immediate improvements in overall throughput. Bell (19) explains that "The tools can be used to assess the capacity requirements of the chain by identifying its constraints and bottlenecks. This enhances the overall logistics performance of the industry and allows us to make informed decisions when planning transport capacity." — Michael Bell, Westrail
Fig. 3 is a schematic of the major components in a reference capacity model for mines, railroads and other commodities. In the planning tool the four quadrants specify the current system as it exists or is being contemplated. The central function both specifies the nature and specifics of the demand on the supply chain, and the available tactics for managing how the demand will be met.

The four quadrants and central activity perform the following sketched functions:

- **Services** - It is here that the major service components provided by the organisation's core competency is specified. For the rail component of a supply chain this would mean the specification of the contractual requirements to the mines serviced by the supply chain. This is a description of the way the service provider mobilises and deploys its resources, via its dispatch rules, and defines other elements, such as background traffic, around which it has to operate. The organisation's history and culture is revealed in this quadrant, via the specification of these rules.

- **Resourcing** - In this quadrant, the availability of the various resources in the system are detailed; for example, the track accessibility slots, the availability of rolling stock via schedules and the rostering of crews. It covers long, medium and short term resourcing profiles via these schedules.

- **Productive Flows** - In this quadrant, the sequence of process steps which make up the productive flows and internal processes of the business are mapped. For example, the haul cycle steps required for a consist to go to the mine, load, travel to the port and unload, and then return would make up one of the productive flows. This "process mapping" data covers cycle details, sequencing rules and routing information.

- **InterModal Transfers** - It is here that storage buffering and handling delays will be specified when commodities are transferred from one sub-system to another, or from one mode of transport to another. For example, the details of the unloading process at the port would be one of the intermodal transfers so specified. This information also covers scheduled and unscheduled availability of these inter-modal resources. It is differentiated from the process quadrant due to the tendency for intermodal points to be located at system boundaries, and also because intermodal transfer points tend to be points of high capital value, often becoming determining constraints.

- **Demand Management** - In this central component the actual nature and details of the demand requirements on the system are detailed. A good part of this demand profile follows on from current contractual obligations, while the remainder is often the capability to handle "spot" opportunities. Here is where the decisions on how the available resources will be utilised in order to meet those demands are determined through robust scheduling and contingency planning. The customer sits at the heart of this component - customer profiles are segmented.
here, cascading from the future to the present, via future demand projections, converted to annual demands, then converted into Monthly/Weekly/Daily, as appropriate.

The detail of how this set of inputs is translated into the software by our underlying Planimate™ software platform originating in Australia, (Seeley and Warren, 1994) are beyond the scope of this paper, but how they work can be seen by viewing the software in operation. Outputs are also divided in a similar fashion.

The application of reference capacity models

With these models, such phenomena as the spread of system congestion can be reduced, work practices can be changed to relieve pressure on a key constraint, and improvements made to haul cycles which are critical to throughput. Expensive capital investments can often be avoided or significantly delayed by implementing policy changes which improve performance. Least cost outcomes can truly be found without looking to expenditures on more capital resources as a way out.

Such performance improvements are very important, however, more significant opportunities open up when rail movements can be coordinated with commodity suppliers such as mines, and with loading/unloading activities at the ports. This is typically done by regularising production flows and their take-up by the logistics network. This approach enables managers in the various stakeholders to take a more proactive planning approach, by gradually progressing through a series of steps from reaction to significant planning horizons. In this manner, both mines and rail can improve their competitiveness through the identification of dynamic coordination opportunities which enable them to improve utilisation of assets, and improve the competitive positions of the various service providers on the rail system.

In this manner, real advantage for all concerned emerges by focusing improvements across the entire supply chain. It is clear then that improving the performance and reducing the costs associated with export supply chains is critical to the competitiveness of export coal mines. This takes on added urgency when external competition and the realities of the international marketplace are taken into account.

World competitive performance

Whilst individual mines and railroads can improve their processes in order to drive down their running costs and capital burden, they face major challenges in their endeavour to improve the performance of export supply chains. Individual mines and other commodity suppliers are intimately constrained by the overall performance of the supply chain. Global competition will not yield simply to the biggest provider, but rather to the combined synergy of the stakeholders in a given region, which can function as a coordinated unit in order to meet client requirements. It is a more effective way to be a clever provider. Such providers will reduce the significant system-wide logistics costs of moving coal from the mine to export ships, via the shared infrastructure of rail systems and ports.

What follows in this paper, is a description of how the empowerment of supply chains has been done, and is being done, with the assistance of a powerful software platform, called Planimate™ (Seeley et al 1995), which enables all of the stakeholders to view a common "Big Picture" of the current situation, and to evaluate any number of potential scenarios which would effectively meet future demands. The implementation steps are straightforward but are greatly assisted by a pragmatic trust of the other stakeholders, the will to make internal political adjustments within each organisation, and a good software platform which provides dynamic visualisations and an effective distribution of information.

The time when a mine could consider only optimising the costs of its own internal operations and price negotiations with the rail provider are effectively over. Medium and long term survival and profitable exploitation of the mineral resource, is now intimately tied to the performance of the entire supply chain and its ability to compete in a world market.

Visualising the whole-of-business

What is meant here by the term "a visualisation of the Whole-of-Business" is not merely the bottom line of some accountant's static analysis, or a pie chart from some sophisticated but unproven spreadsheet financial model. Rather, it is an animation of the entire supply chain which enables people to see the various interactions which occur between client and provider, intermodally, and with respect to external demands on the overall chain. This animation also delivers in-depth analysis of the behaviour of the whole logistics system. This kind of visualisation is a powerful tool for achieving agreement on how the supply chain actually works overall, what is the key determining constraint (see "Capacity

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Assessment and Planning”), and upon what issues stand out for immediate attention. Moreover, everyone can see how the system works and achieves its numbers. Our Planimate™ software platform delivers such visualisations quickly and dramatically.

When whole-of-business animations also provide excellent reporting and graphics of key performance indicators, stakeholders are enabled to see a much greater perspective from which to make decisions then when they are only focused upon their own internal measurements as in Seeley and Griffith (1994b). Stakeholder mines are very autonomous companies which prefer a lack of commercial regulation. Typically, a number of opportunities are rapidly seen which can promote their own self-interest in their own individualised manner. Some of the typical opportunities are:

- **Looking over fences** - The ability to see how one’s own performance affects or is affected by neighbouring stakeholders in the chain; dynamic collaborations where mutual benefits are possible, almost leap out from the screen.

- **Deeper Understanding of Capacity** - Whole-of-system understanding enables each to notice what enhances, and what limits their use of resources, providing practical alternatives which keep costs minimised.

- **Including the Client Loop** - The ability to see the relationships between the throughput productivity of the supply chain, its impact upon the marketplace, and the resultant demand requirements placed on the chain by its world clients.

- **Schedule Harmonisation** - There are many possible benefits from harmonising the production schedules of the mines with the consist schedules of the railroad, and with the situation at the ports.

In using the software platform which delivers this whole-of-system picture, individual stakeholders can greatly profit from being able to alternate their perspective from their own self-interests back and forth with a perspective which reports on the overall interests of the entire supply chain and a global view. Such global views by keeping their relevance to individual stakeholders, avoid any undue influence from dominant positions, such as a railroad or a port authority.

For example, a mine might explore how a more continuous supply of labor can be achieved in its environment, then watch what affect such a policy change has on the overall performance of the supply chain. Only in this manner can stakeholders such as mines, truly appreciate the interplay of their actions within the synergy of the regional industry. Such synergy emerges from dynamic collaborations chosen for their benefit to the self-interest of both parties, as opposed to any enforced cooperation or rules of equity.

However, the practicality of working with a number of stakeholders, each with their own self-interest and management, does not indicate the requirement of some overall chain authority, directing the actions of the stakeholders. Rather it becomes far more useful to think in terms of sustaining a global perspective which includes an individual planning capability that acknowledges the unique business style and processes of that company. An individual focus can then be established and sustained by the contractual relationships from which effective global performance will naturally emerge from the best efforts of each of the stakeholders pursuing their own interests. This theme is taken up in a later section on Stakeholder Relationships. Our understanding and tactics for addressing the issues for the entire supply chain is presented next, under the term "supply chain management".

**Supply chain management**

In this section, key understandings of an export supply chain are discussed, including how all of the pieces come together to function as a unit, and working with the dynamic concept of the haul cycle taken from train graphs, can incur significant savings. A mine depends upon the other players in the supply chain in order to meet its client’s demands effectively. This is then followed by brief discussions of our two main tools for getting the most productivity from the supply chain, capacity assessment and robust scheduling.

**The logistics network**
The components of the supply chain, including producers, productive flows, primary transport, intermodal transport, storage, depots, cargo assembly, and shipping stem, can be viewed as all one network (Fig. 4). That is, the chain is a network consisting of various transport (process) flows alternately connected by buffered interfaces. The software modelling capability includes entities for directly modelling each of these components and their dynamic behaviours.

Transport flows - A single mode movement of a commodity between two buffers; such movements can be considered single step in a larger movement called a route which follows a particular sequence of steps; these route often return to their origin, and are labelled route cycles. Generally speaking, each flow will have a transport rate as a commodity volume/per time unit.

Modal buffers - These are storage locations which handle modal changes in an overall transport flow, such as truck to rail, mine to rail, rail to off-loader. They are particularly useful for handling changes in transport rates between flows. For example a high rate in requires a buffer in order to provide a low rate out. In this case, an in-flow provides an input burst to the system. In the opposite case, a build-up in the buffer is required in order to release commodity at a higher out-flow rate, and more fully utilising the output modality. Variability in transport flows across the network, actually generates temporary periods of substantial rate changes at various locations.

Buffer costs - Modal buffers, when considered in this dynamic way, have a very different cost basis from the usual cost basis for inventories. So-called economic order quantities are irrelevant for getting the highest performance for the lowest cost in logistics networks. They do not account for transport rate dynamics, especially for production and shipping buffers.

Dwell (time) buffers - These are the time intervals at the end or middle of a haul cycle before the return trip or next cycle begins, and applies to the trips which trains, trucks, etc. take. Reducing the dwell time, shortens the overall haul cycle time for transport resources, and increases utilisation.

Capacity assessment and planning

There can be significant misunderstandings and confusions around the capacity of a transport network, from various viewpoints within the organisation. There is a contractual capacity for example, implied by contract commitments and often with a number of significant assumptions not articulated. There is also an "optimal capacity" which is used for planning purposes, which is often idealistic with assumptions of no delays, linear behaviours and perfectly clear running. This optimal capacity can be contrasted with the nominal capacity which acknowledges actual and, currently necessary,
interference, congestion and breakdown, and is normally only available through a dynamic reference capacity model (RCM).

In contrast to train scheduling requirements, capacity planning endeavours to maximise commodity flows recognising the impact of intra-traffic conflict for the primary transport resource (network). Various current operational policies (e.g. daylight loading only) are also acknowledged, further reducing nominal capacity. Business planning can proceed by exploring various measures and policy changes, such as those involved in manipulating the haul cycle parameters. Pooling diagrams (which show a week’s availability for the rail network) and train/transport graphs, available with the animated management tools used for capacity reference, are also useful for supporting such explorations.

These management tools are making a significant contribution to assisting mining and rail to improve their competitiveness. Through them, effective methods for improving the utilisation of mine and rail assets, therefore enhancing network capacity can be explored, especially in coordination with other stakeholders. Furthermore, exploring the feasibility and future returns of joint ventures to establish economic transport networks for the opening of new mines, can also be carried out in a similar manner.

The determining constraint is that resource which is the current operational bottleneck. A change to the determining constraint tends to financially overwhelm other parameters, hence strategic choices regarding it need to be made first, before other alternatives can be considered.

Robust scheduling

Scheduling the carrier resources in a transport network with many clients, can become a complex and vexing issue. The operational experience is often that all resources are either running flat out, or else that everything is broken. If an approach is taken which searches for some optimum "academic" schedule, it is our position that typically a number of blind and impractical avenues will be taken, which often yield complex schedules which are only relevant for the assumed conditions existing at a single moment in time. When said conditions change, these schedules often become infeasible or very difficult to adapt to on-going change or contingency; that is, these schedules are often "brittle". Further difficulties which can be encountered with the optimum approach is that the resulting schedules may not be easy for operations to understand or work with, and may impose quite tricky and sometimes costly demands upon labor.

The situation which enables the use of resources required to meet a given demand to be driven down the most, is a continuous and uniform demand. However, if planning is carried out under the assumption that such demand exists when in fact the demand is highly variable, natural vagaries in the whole-of-system will ensure that things will become very congested. However, it is possible to respond to bursts of heavy demand by temporarily doubling or tripling capacity, by making adjustments to the haul cycle or work practices for short periods. How can a manager approach this proactively, without having to undergo expensive reactions?

One way that this can be accomplished is by taking an approach which produces a great deal of uniformity and regularity for a base load of demand on the logistics network, and then sets aside capacity for bursts in demand or other contingencies. In this manner, flexibility can be built into the schedule. Further, by rehearsing responses to potential breakdown situations, and accounting in the schedule for their potential, scheduling can become very robust. Schedules which are robust are basically valid no matter what happens, and impose a degree of regularity which makes operational planning easier. This entire approach is called Robust Scheduling, because it adds flexibility and manageability to scheduling in a manner which optimal approaches find difficult.

This approach to overall logistics was proven out in many applications for mines and railways, before applying them to a substantial multi-stakeholder logistics situation for grain export. Moreover, robust scheduling has been shown to be a very effective method for access management (beyond the scope of this paper) to a determining transport resource such as rail. For now, it is enough to observe that there exist many rewarding opportunities for rail and port providers across the globe, to handle the booking of access slots effectively.

The prospect of integrating logistics considerations across all of the stakeholders in a supply chain has been an opportunity staring at our applications from the beginning. However, it was out of a recent multi-stakeholder application in the grain supply chain for Western Australia, that the joint industry planning platform, now being applied to coal supply chains, has emerged.
JOINT INDUSTRY PLANNING PLATFORMS

Since September 1992, InterDynamics has been working with various components of export chains involving coal, sugar and nickel, largely with Queensland Rail and Westrail. These often involved either a mine or refinery and the railroad, or the railroad and the port loading facility. InterDynamics' whole-of-business approach indicated that modelling the overall export chain made a lot of sense. However, it was the WA Grain supply chain experience which crystallised many of the ideas now being applied to coal supply.

The WA grain export chain experience

"There is a need for us to be able to respond quickly to the dynamics of the marketplace which requires up to date information on shipping requirements and stock inventory levels."  
— Michael Bell, Westrail

Beginning in early 1996, an initiative was undertaken in Western Australia, which took about 18 months to mature, and which has seen emerge a "joint industry planning platform". Spearheaded by John Goodall of Westrail, Bob Bridges and Klaus Fahrner of InterDynamics, this project gradually marshalled the interests and information resources of the stakeholders in the export of grain from that state. The stakeholders included marketing authorities, bulk handlers and assemblers, transporters and port loading roles in the export chain.

The construction of the WA Grain planning platform was undertaken to optimise financial returns from export grain by delivering a high level of customer satisfaction through available product quality and quantities, and by effective timing of grain delivery to international markets. Delivering such returns also means minimising overall logistics costs through performance enhancement and effective use of logistics resources and assets.

Various animated management tools were constructed to assist the various parties to effectively anticipate the on-going and highly volatile, shipping demands at each port. These included a seasonal planner for how the harvest would be handled, a clearance planner to schedule transportation movements, shipping planners which monitor the ship demand, rail capacity planners, and cargo assembly planners.

Using the individual planning tools and the Navigator, which provides an overall picture of the state of play in the grain supply chain, enables planners to achieve the best balance between ordering and administration, warehousing, inventory and transportation in order to minimise overall logistics costs.

The development and use of a joint industry planning platform for the export supply chains of the Western Australian Grain Industry has demonstrated that an industry facing significant restructuring and increased competition can achieve major throughput and cost reduction gains when stakeholders in export supply chains share key planning information using the Internet and state-of-the-art planning tools. There has been agreement between the stakeholders to share more key information on an open basis using the shared planning tools of the supply chain.

Coal export supply chains

Joint industry planning platforms for export supply chains have been developed, are being considered, or are at initial stages of development for a number of Australasian coal export supply chains (Fig. 5). These include: the Illawarra, Hunter Valley, SE Queensland, Blackwater / Moura, Goonyella, Mt Isa / Townsville and New Zealand export coal regions.
A typical example of a coal export chain is in the Blackwater / Moura system where transport is provided by Queensland Rail (QR). Initial tripartite meetings amongst the mines, the Gladstone Port Authority and QR started in early 1996, at QR's initiative. These meetings are now held monthly where discussions centre on possible coordinated activities that the stakeholders could pursue and commit to. Relevant statistics on the system monthly performance are tabled and the degree of adherence to any previous recommendations made is examined.

All three groups in this chain had their own vested interests in making their initial commitment. While there was a prudent and cautious uptake of the process in the initial stages, the spread of chain relationships have gradually developed to the level where the idea of group coordination is central in everyone's everyday thinking.

Some of the coordinated activity which has resulted from this process includes:

- A representative from the Gladstone Port Authority (GPA) meets each week with Queensland Rail operational staff to establish the train schedule for the week. Matters for discussion include the shipping stem, belt-line and conveyor availability, and stockpiles. While these meetings are now well established, this coordination was a watershed for both the QR and GPA organisations. Some IT developments designed to facilitate scheduling initiatives and implementation have also been developed and instituted as a direct consequence of this approach.

- When particular mines may have immediate demand to service (i.e. ships berthed at port) and low stockpile levels, QR has [on request] negotiated to defer some services to mines where demand is at a lower priority.

- Currently QR is arranging to trial the removal of one consist from the present complement of 15 which services ten mines in the Blackwater system, with overall about seven major mine owners. The system is also required at times, to service railings from five mines in the northern Goonyella system, sent "cross-system".

- "Even" railing demand has been identified as critical for system efficiency. In response, QR has offered the mines the opportunity to trial, over a period of time, a schedule that is negotiated by all the system stake-holders. If this schedule is adhered to, QR will provide the mines collectively, a significant financial incentive. This approach is designed to encourage an even railing pattern and significantly decrease the need to hold surplus rollingstock assets to cater for variability in production and shipping demand.

The continued work of the tripartite coordinating meetings has gradually altered attitudes in a way that has promoted good-will and system efficiencies. This has allowed the evolution of a culture of evenness in production and transport flows where flexibility is still retained. The entrenchment of such a culture will serve the future well, when very significant increases in coal demand are anticipated.
Testimonials from the coal supply chain

According to Robert Elliott of the Gladstone Port Authority,

"Once the initial contacts were made, the spirit of co-operation and good-will snow-balled. Between the Port Authority and Queensland Rail at least, there still exists no firm contractual arrangements and I see the over-riding benefit as enabling what is after all a very complex system the scope to continue to operate even more efficiently without being tied to inflexible contracts. The meetings have set a scene where ideas may be interchanged and a better understanding of each other's business be gained. People on the floor have also begun to develop a better appreciation of the coal system."

The following are comments provided by mining representatives,

"I have gained a better understanding of QR operations and the complexities involved."
"We are more prepared to sacrifice trains for someone else, in an emergency."
"We are prepared to work together even to the extent of sharing stockpile space. A recognition that capacity provision rather than on-time performance by QR is the vital factor."

Components of a joint planning platform

In approaching this project, there are a number of important considerations regarding the business realities in export chains. First of all, any synergistic collaboration must come out of the context of a group of highly independent and autonomous organisations, each pursuing their own self-interests. This context is called the Joint Industry Planning Group.

In the planning group and subsequent contractual relationships, one exploits the individual business styles and scales of commitment to the industry of each stakeholder, assuming that individual organisations have explored the whole-of-industry perspective for themselves. This avoids both the dependent attitude of an organisation which wants the supply chain to look only for it, and any centralist attitudes which could force ineffective policy in the name of equity. Instead no one has to be led, the supply chain organises itself for overall benefits through opportunistic contracts and individual respect.

What typically transpires is that individual models are undertaken for each of the stakeholders which reflect their own viewpoint of their own operations, their own self-interest and their perception of the rest of the supply chain. In effect, the multiple viewpoints of each of the stakeholders is acknowledged via these individual models (Fig. 6). This then provides an important basis when one returns to an overall supply chain model. This overall model, termed a joint industry planning platform, is able to be constructed in a manner which respects these multiple viewpoints, by ensuring that data sources and business rules are also respected, and that their individual key, but not sensitive, performance indicators (KPIs) are readily accessible.

In all of its dynamic modelling, InterDynamics attempts to provide what can be termed "congruent systems images" of the organisation involved as described in Seeley (1996). Weinberg (1994) has described some of the utility of congruence in his book, and Taylor (1995) has utilised a similar kind of notion in his term "convergent". Briefly, "congruent" in this context means that for every component and business process in the actual organisation, there is a corresponding component in the software model, kind of like the software also "walks the talk". This is in distinction to the distorted images which other information systems may convey. What congruent means in this situation is that the way in which the organisation actually behaves over time, with all of its variability and non-linearity, is accurately reflected by the model's execution. This means that a precise application of the corporate business processes, embodying all of their business rules is carried out. InterDynamics has learned from long experience in object-oriented software, how effective this degree of congruency is, in constructing robust and flexible models very quickly.
However, in order for a joint industry planning platform to become a practical reality, these individual organisational images must be blended into an effective and congruent systems image of the entire export chain. Often, the individual stakeholders have only a limited view of the operation of the overall chain. It then becomes possible to achieve a consensus on the overall model by getting each to participate in the process of building a joint industry platform, and to view the result as an animation of how the entire chain works. In this fashion, the dynamic images appearing on the screen become a very effective mediator between the perceptions of the various stakeholders. Since the individual stakeholder perspectives have been respected in the software, then a common reference image (CRI) can emerge for joint industry planning.

With a CRI available, the issues for a joint industry planning platform focus around obtaining source information from the stakeholders, and in distributing the current status of the CRI and its behaviour to the interested parties. This brings us into the area of shared information, and it is a delicate process. It is only the factual, what-is-happening kind of information which is needed by the CRI, and effectively by the other stakeholders. This is in contrast to financial and performance measurement data, information around which each of the stakeholders naturally feels sensitive and should not be divulged to others. This data requirement may well involve accessing and filtering data from a stakeholder’s databases and perhaps mainframes. InterDynamics has focused on constructing such filters in coordination with corporate IT departments. Once it is seen that this distinction can be made, then the planning platform can be progressed to completion. However, it may well need to be supported by contractual relationships.

Once a joint planning platform is completed and validated with a congruent systems image, the next step is to attend to the effective distribution of its data and behaviour to interested parties. This could be done within the IT department of one of the stakeholders. However, competitive energies being what they are, it is difficult to assure all of the stakeholders, that an unfair advantage will not be gleaned by the host. Another approach is to entrust the planning platform to a respected custodian who can then securely distribute the information via the Internet from their host system which can either be a website or an internal bulletin board capability. In effect, an Intranet for the industry is created. As described in the next section however, an effective coordinating caucus between the stakeholders is also necessary.
Hence, the following components are necessary for the construction of a joint industry planning platform:

- Multiple viewpoints
- Congruent Systems Images
- Common Reference Image
- Shared Information
- Contractual Structures
- Information Servers for a Regional Industry Intranet
- Individual Board Approval for the Industry Coordinated Planning group.

**STAKEHOLDER RELATIONSHIPS**

“It is refreshing to note that all of the relevant parties are committed to seeing InterDynamics’ [application] system being developed to its optimum capacity. This will in turn allow a more transparent supply chain to be finessed for the benefit of WA grain growers.” — Craig Thompson, the Grain Pool of WA

Regional export supply chains often had a heritage of little communication between the links in the chain and a tribalistic culture. With the advent of the commercialisation of government owned trading corporations and the development of a team culture in the various logistics supply chains, this situation has changed significantly. A key question for regional export industries is whether this new freedom will be frittered away by acting only from local viewpoints, or will the parties involved take advantage of the opportunities afforded from a whole-of-industry view?

**Contribution relativities**

In entering into contractual agreements, organisations take care to see that their self-interest in the matter is safeguarded. Once the contract is settled, then actions and expenditures are guided by the contract’s requirements and performance measurements, while the individual self-interest of the parties is pursued. In effect, the overall behaviour emerges from the rules of relationship and measurements. This is because the rules shape and constrain the expression of self-interest in the shared context, while the measurement process directs the individual development process of this interest. When considering joint industry planning, how can the mutual self-interest of the industry be pursued given the behaviour of the individual stakeholders?

For example, if one player considers only their own costs relative to the system’s overall level of activity, then it can easily happen that isolated cost containment activities can strangle the overall system. Moreover, a small expenditure in resources by one party may earn many times that amount for the entire system, but their accounting measurements and contract agreements may prevent that party from realising any benefit in a measurable manner. The system-wide benefit may not acknowledge the source of the original investment, or the place where the benefit is realised does not appear to be directly connected to this source. In such circumstances, management would be laughed out of any boardroom.

However, if an investment or cost reduction by a stakeholder creates significant losses for the other stakeholders, yet earns a significant return for the originator, then local management will traditionally be judged to be very successful. Hence, players in the system need to recognise that there exists contribution relativities to system performance of additional investment in resources, depending upon the location of the investment. Our conventional approaches to good financial
decision-making deter the profitable synergy which can occur for an entire export industry. How can this be turned around so that the entire supply chain can benefit?

One point is to observe that enforced equity and cooperation does not recognise the structure and business styles of individual stakeholders. Such an approach implicitly makes assumptions about marginal contributions and opportunities which are simply not valid, and prevent significant opportunities from being taken up for the performance of the entire supply chain. One need not hold up a bulk carrier at a port while waiting for the cargo of a small ship to be assembled from the hinterland. Instead, there are better effects across the board when a robust scheduling and proactive planning approach is applied. Predictability is encouraged and there are rehearsed responses to bursts of demand, and maintenance opportunities when there are gaps in demand (planned cancellations).

Moreover, in comparing the performance of the supply chain or any of its components to its global competitors, it should be kept in mind that only wholistic performance measures, as described in Seeley and Griffith (1994b), should be used. This will avoid suboptimisations and counter-productive measurements, such as using the number of employees when resource investment is far more appropriate.

**Behaviour derives from the rules of relationship**

Further to our response to the question of how the entire chain can benefit, is to encourage contractual relationships between parties which encourage predictable (planned) flows of production or service. Compliance to the plans can be handled by utilising rewards and penalties, and by tying contractual satisfaction to global measurements of system-wide performance.

*In effect, contracts in regional industry planning should establish rules of relationship which encourage the individual self-interest of each stakeholder to be tied to the overall mutual self-interest of the entire supply chain.*

When such relationships are set up, the desirable and profitable synergy (win-win) will naturally emerge without any imposition from outside parties. Hence, the elements seen as necessary for effective stakeholder relationships in joint industry planning are:

- management and board support for synergy, backed by information sharing
- tying self-interest to the mutual interest
- key performance indicators which measure overall system performance
- contract performance tied to global measurements
- contracts which support predictable flows between stakeholders
- recognition that behaviour emerges from relationship rules and measurements

As a result of the information sharing required by the joint planning platform, and suitable contractual relationships like those described above, the participants are committed to continuously improving the throughput performance. The supply chain planning platform enables multiple stakeholders in the chain to work effectively together to improve the performance of their own respective companies as well as the chain itself. Of course, this cannot work in a strictly automated fashion without the face to face discussion and negotiation of an associated joint industry planning group.

**Stewardship of the joint planning platform**

One can certainly conceive of a central authority which would model, assess, and impose decisions based upon a joint industry planning platform. However, in modern economic climates, both the expense of an additional bureaucracy and a
natural resistance to centralised control mitigate against such a direction. One alternative approach which InterDynamics believes is viable, is a stewardship role which would look after and sustain the joint industry platform. The steward could be an agreed upon stakeholder, or rotating participating stakeholder, or an external but trusted agent such as the platform supplier. The latter approach can be very effective because it avoids the pitfalls of the internal politics of a steward stakeholder diverting energy and attention from this very important global function.

The stewardship role would include the following attitudes and functions:

- Supports stakeholder relationships, rather than taking a top-down role - The steward facilitates the functioning of the joint industry planning group, and both technical and political aspects of relationships between stakeholders. This is not seen from a command and control perspective.

- Provides capabilities, not solutions - The steward ensures that joint planning needs can be responded to effectively and that the capabilities of the planning platform are evolving to meet the growing needs of the industry. The steward avoids responding to any expectations that he should provide and promote joint industry issues.

- Maintains a global perspective, not a global management - The steward ensures that an effective whole-of-industry view of the functioning of the export chain is maintained and its data relevant and current, and attempts to enlarge the platform’s scope to include relationships with international clients. The steward avoids any top-down managing.

- Ensures the Congruency of the Common Reference Image - The steward is vigilant about making sure that the CRI accurately reflects the actual components and behaviour of the supply chain, and also maintains an on-going verification of the platform’s operation.

- Monitors and safeguards information sharing - The steward ensures that only information which is agreed upon, avoiding any compromise to the positioning of individual stakeholders, and securing its transmission and exchange.

The regional industry steward is a possible new component in supply chains, and it signals a potentially new kind of organisation which could deliver and steward joint planning platforms, now and in the future.

CONCLUSIONS AND FUTURE DEVELOPMENTS

The major benefit from the use of a Joint Industry Planning Platform is in the establishment of an analytical framework from which alternatives can be explored with which to best meet the overall demand upon the chain. This advantage is supplemented by the road/rail response capability that can be activated through the coordination of stakeholders which have often seen themselves as competitors. For example in early 1997, this was dramatically demonstrated by the WA Grain industry’s coordinated response to a windfall opportunity. This opportunity occurred while a major port was shut down, and while there was severe pressure on a supplying region. During the second quarter, a coordinated response came out of this shared analytical framework.

The use of the Joint Industry Planning Platform enables both opportunities and difficulties to be anticipated in a manner far more effectively than what can be done via a single stakeholder. Moreover, the establishment of a coordinating or liaison planning group to oversee the construction and maintenance of the JIPP, ensures that a political vehicle is in place, which can when prompted obtain the appropriate internal actions from each of the parties.

"Our ability to achieve greater throughput from the supply chain at a time of higher world grain prices has resulted in a major revenue benefit in excess of two million dollars." — Rob Allen, Australian Wheat Board

There are a variety of other improvements to the operation of the supply chain which may also come out, once the parties can visualise the impact of their actions upon the rest of chain, and how all of this comes around to effect them. For example, consider the filling of stockpiles with different grades of export coal which are adjacent to the ports. If this is
done without regard to the upcoming shipping stem's requirements, a situation can be created where a ship has to wait, incurring demurrage, for the correct grades of coal. Limited storage capacity at the port could make this situation sticky. When the big picture of this pending situation is provided in a context of coordinated decision-making among stakeholders, then appropriate planning can be done to minimise expenses. Strictly local considerations by a mine give way to a global perspective which benefits everyone.

Another acknowledged benefit is that the business rules and business processes by which the joint industry works, become very clear and negotiable through the animated visualisation process. Constructive changes then become readily evident, and the role of these business rules in contractual agreements becomes highlighted.

Overall, the joint industry coordinating committee can become an effective leader for the individual business planning and decision support units within each stakeholder's organisation. Not only does its strategic perspective open up many opportunities for the individual stakeholder to make overall savings, but it also opens up and makes visual, prospects for coordinated business development. Hence, the establishment of a joint industry coordinating group which is given the stamp of approval from the board level of each stakeholder, becomes the final necessary requirement to make a joint industry planning platform work.

As experience with joint industry planning platforms grows and their capabilities mature, especially in key commodity export chains such as coal, grain, sugar and other minerals, a number of further capabilities could be explored. These include:

- Responding with additional capacity - Developing more sophisticated end-user routines in the platform which would find increased capacity alternatives, both for temporary responses or for long-term requirements.
- Live Imaging and Live Scheduling - Using data from currently available databases, live imaging provides end-users throughout an organisation, with a dynamic whole-of-business visualisation. Live scheduling enables re-scheduling which accounts for changing requirements and resources, to proceed from any point in time.
- Refining the construction of collaborative contract relationships - A critical component of an effective joint planning platform is the completion of contract relationships which enhance the synergy of the supply chain. Refining the formulation of such contracts as experience with them matures, will be a necessity.
- Global Models & Positioning - Expanding the scope of the dynamic modelling within the platform to include the state of play within the global commodity marketplace. Developing global commodity models which include the export client relationships, is a natural extension.

Recent business developments in adopting effective organisational structures and utilisation of the Internet, have made it possible for new forms of business organisations to emerge, which carry out functions which were only dreamed about in previous business climates, or functions which were hopelessly left to government regulations to solve. The pressure from the economic competition of globalisation has challenged the smaller national export industries to play smarter, in order to reap the benefits of locally generated wealth. The innovation of the joint industry planning platform and its application to coal and other commodity export chains, based on our experience so far, should make a significant contribution to the health of economic regions in this age of the borderless economy.

REFERENCES

New Environmental Legislation: Implications and Issued for Coal Mining in NSW

M Astill

INTRODUCTION

In December 1997, a series of new environmental legislation was enacted including:

- Environmental Planning and Assessment Amendment Act 1997;
- Contaminated Land Management Act 1997;
- Protection of the Environment (Operations) Act 1997;
- Native Vegetation Conservation Act 1997; and
- Pollution Control (Load Based Licensing) Act 1997.

These new Acts will have varying degrees of impact on the coal mining industry in New South Wales when they commence later this year.

Some of the changes will have minimal or no impact on the coal mining industry. For example, although the Native Vegetation Conservation Act has extensive provisions to provide tight protection of the State's native vegetation, the Act does not apply to clearing that is authorised under the Mining Act 1992.

However, other changes and initiatives are likely to have significant consequences for the coal mining industry. This paper will focus on the pivotal new laws set down in the Contaminated Land Management Act, and in the Protection of the Environment (Operations) Act. It will also briefly address the recent Parliamentary Review of the Threatened Species Conservation Act.

CONTAMINATED LAND MANAGEMENT ACT 1997

Introduction

This Act is designed to promote better management of contaminated land and to harmonise the current piecemeal legislation on contaminated land. It amends the Environmentally Hazardous Chemicals Act 1985.

The Act identifies the Environmental Protection Authority (the "EPA") as the regulatory body with jurisdiction over the management of contaminated land. In carrying out its functions, the EPA must have regard to the principles of ecologically sustainable development, defined in section 10(2) to include, among other things, the prevention of serious or irreversible environmental damage and the conservation of biological diversity.

The Act links principles of ecologically sustainable development with a "polluter pays" system, in which, in theory, those who generate pollution and waste bear the cost of containment, avoidance, or abatement of that pollution or waste (section 10(2)(d)(i)).

The Act is of particular interest to the coal mining sector because it operates retrospectively. In other words, the Act covers any contamination or risk from contamination that was present prior to the commencement of the Act. A person or company will be responsible for land contamination even if the contamination occurred indirectly, or the activity was lawful at the time of contamination, or if the risk arose from a change of use of the land.

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This section will address the following issues:

- The EPA's new powers;
- What land is affected?
- Who can receive an order? and
- Evidence: who contaminated the land?

**The EPA's new powers**

Section 6 identifies the general duties of the EPA which include:

- to examine and respond to information and reports it receives of land contamination;
- to address any significant risk of harm that the contamination presents; and
- to record its actions and the reasons for those actions.

Where the contamination presents a significant risk of harm, the EPA has the power to:

**Keep records**

The EPA may:

- make records (or ensure that records are made) of the evidence of the contamination, risk and harm; and
- maintain records of any current declaration and investigation or remediation order.

Full details of the contamination must be available for public inspection at the EPA's principal office and any other place the EPA thinks fit. In addition, if any person applies for a copy of the record of contamination, the EPA must provide that information (section 58(2)).

**Employ community awareness strategies**

The EPA may employ community-based strategies to minimise the contamination, risk or harm through education and public awareness (section 7(c)). It may also generate any kind of publicity about a particular type or source of contamination that it feels would improve public awareness or understanding (section 104). The possibility (or threat) of a company being identified publicly as a perpetrator of a pollution incident may encourage companies to further develop and refine their environmental risk management policies and procedures.

**Order investigation**

The EPA may declare land to be an "investigation area" if it has reasonable grounds to believe that the land is contaminated (section 15). Once the land has been declared an investigation area, the EPA must "with reasonable expedition" either investigate the land or order persons to investigate it (see Part 3). Orders to investigate must be made by written notice (section 17).

**Order remediation**

The EPA has broad powers to declare land to be a remediation site, and order persons to remediate it (in accordance with Part 3).

"Ordering remediation" may include the following actions:

(a) erecting a fence,
(b) erecting a wall or bund or other barrier,
(c) either treating, storing or containing soil (or other solid or liquid) on the land, or removing that substance from the land and treating it elsewhere,
(d) vacating the land or ceasing all and any activities upon the land,

(d) erecting appropriate warning signs,

(f) refraining from disturbance or further disturbance of the land or at least below a specified depth,

(g) monitoring the effectiveness of remediation or the risk of harm presented by the contamination of the land,

(h) informing the EPA of any change in the ownership or occupancy of the land, to the extent that the person subject to the requirement is aware of the change.

This is not an exhaustive list. The EPA's broad powers to contend with contamination are coupled with a strict onus placed on landowners and occupiers to report contamination to the EPA as soon as it is discovered (discussed below).

What land is affected?

_Land that comes within the ambit of the Act is land that is:_

1. contaminated; and
2. presents a significant risk of harm (section 9(1)).

Relevantly, "contamination" is defined as the presence in, on or under the land of a substance at a concentration above that normally present in the same environment that presents a risk to human health or the environment.

"Harm" means harm to human health or any aspect of the environment. It includes direct or indirect alteration of the environment that degrades it. It is a broad definition. However, as "degrades" is not defined, it may be that the mere existence of a substance may not be enough if it does not actually degrade the land.

To assess the land, the EPA must consider a number of factors:

(a) whether the contamination has already caused harm;

(b) the toxicity and quantity of the substances present;

(c) whether there are exposure pathways between the substances and human beings and other aspects of the environment;

(d) whether the current or approved use of the land and adjoining land is likely to increase the risk of harm being done (eg. where the land is being used for child care);

(f) whether the substances have migrated or are likely to migrate from the land; and

(g) any guidelines made or approved by the EPA on contamination and remediation.

Who can receive an order?

As discussed, the EPA has wide powers to declare investigation areas and remediation sites and to order appropriate persons to investigate or remediate land.

Persons who may be issued with an order include (section 12):

- the person who caused the contamination, or, if that is not practical;
- the current owner of the land, whether or not that person had any responsibility for the contamination, or, if that is not practical;
- a notional owner of the land (whether or not the person had any responsibility for the contamination). A "notional owner" is defined as a person who has some kind of freehold interest in the land, and includes a mortgagee in possession (Section 14).

Under section 68, it is up to the recipient of an order to prove that they are not the owner of the relevant land. Furthermore, the burden placed on land owners is considerable as land owners must give the EPA written notice of
contamination as soon as the contamination is discovered. The financial penalties for failure to report are steep (see section 68: 1,250 penalty units [one penalty unit is $110] for a corporation and 600 penalty units for an individual).

In addition, orders may be made against directors and companies to investigate or remediate contaminated land at their own expense (Part 7). The Land and Environment Court may order a director personally, or a holding company, to comply with orders made if:

- a company was wound up without having complied with an investigation or remediation order, or
- if the company disposed of the land within 2 years before the Court made the order.

Evidence: who contaminated the land?

The Act addresses the difficult issue of proving who actually contaminated land. While an innocent landowner may be ordered by the EPA to investigate or remediate land, the Act gives the landowner rights of recovery in court of the costs of compliance with the orders. The onus of proof of innocence lies on the person who circumstantially was most likely to have caused the contamination. Section 67 says:

In any proceedings under this Act to recover from a person the cost of carrying out an investigation or remediation order in relation to any land, the person is taken to have responsibility for contamination on that land if:

(a) the person carried on activities on the land, and
(b) activities of that sort generate or consume the same substances as those that caused the contamination.

Protection of the Environment Operations Act 1997 ("PEO Act")

Introduction

The PEO Act represents a fundamental re-writing of pollution regulation in New South Wales. It repeals the following pieces of pollution legislation:

(a) Pollution Control Act 1970;
(b) Clean Air Act 1961;
(c) Clean Water 1975; and
(d) Noise Control Act 1975;
(e) Environmental Offences and Penalties Act 1989;
(f) parts of the Waste Minimisation and Management Act 1995.

The PEO Act contains many innovations of potential significance for the coal mining industry. It replaces the numerous different licences and approvals under separate legislation with a single licensing arrangement that seeks to cover:

- all forms of pollution; and
- the development and operational stages of controlled activities.

The Act represents welcome, but long overdue, reform. This "Stage 2" legislation has been heralded for a number of years, yet the lack of real consultation with other States and Territories during its preparation means that businesses whose enterprises straddle borders will still spend time needlessly checking compliance under a variety of systems. The EPA has also been very quick to trumpet the Act as a "streamlining" of regulation, but the added burden of the wide-ranging conditions which can be applied to the new integrated licences, together with the new EPA investigative powers, mean that any streamlining will be enjoyed more by the EPA than by business. There are still more sticks than carrots for business. The question remaining is "Can't business be trusted yet?".
This section will address:

- The EPA’s new investigative powers;
- Environmental audits;
- Integrated planning approvals;
- Protection of the Environment Policies;
- Integrated licensing; and
- Penalties.

Investigative powers of the EPA

The PEO Act provides the EPA with broad powers to:

- require, by written notice, any person to provide the EPA with any information and/or records in connection with any matter that comes within the ambit of the Act (section 191);
- enter premises (see section 196) and "do anything that in the opinion of the authorised officer is necessary to be done for the purposes of this Part..." (section 198(1)); and
- search premises, where a search warrant has been issued under the Search Warrants Act 1985.

Upon entering any premises in accordance with this Part, an authorised officer may take any number of actions, which may include:

- examine and inspect any works, plant, vehicle, or article;
- take and remove samples;
- make any necessary examinations, inquiries, and tests;
- take any necessary photographs, films, audio, or other recording;
- require production of any records for inspection;
- copy any records; and
- seize anything an officer believes is connected with an offence.

The PEO Act makes clear that this list is not exhaustive. One slight limitation, however, on the EPA’s powers under Part 7 appears to be in section 201:

"Care to be taken
In the exercise of a power of entering or searching premises under this Part, the authorised officer must do as little damage as possible."

Authorised officers of the EPA are also empowered to require a person to give any information in relation to a matter that comes under the Act (section 203(1)). The EPA or any other regulatory authority can also issue a notice in writing to a corporation and require that a director or officer be nominated to answer questions (section 203(2)). Those answers will be binding on the corporation (section 203(3)).

Environmental audits

The PEO Act introduces voluntary and mandatory environmental audits. The explanatory note to the Bill explains: "Information obtained by licensees in the course of voluntary audits are protected from official inspection or use in proceedings, but those obtained in the course of mandatory audits directed by the EPA are not so protected."

A mandatory audit may be required by a licence condition if the appropriate regulatory authority reasonably suspects that:

(a) the holder of a licence has on one or more occasions contravened the Act, regulations, or licence conditions; and
(b) the contravention or contraventions have caused, are causing, or are likely to cause, harm to the environment.

Chapter 6.3 provides that documents prepared for the sole purpose of a voluntary audit are "protected documents" for the purposes of the Act, that is, the documents:

(a) are not admissible in evidence against any person in any proceedings connected with the administration or enforcement of the Act; and

(b) may not be seized or obtained by the EPA or any other person for any purpose connected with the administration of the Act.

Integrated planning approvals

The Environmental Planning and Assessment Amendment Act 1997 was enacted in December 1997. It introduced a system of integrated development assessment intended to avoid the duplication of approvals between the EPA Act and approvals granted by other regulatory authorities.

"Integrated development" is development which requires a development consent and one or more of the approvals under ten "primary" Acts listed in the Act. For example, development which requires development consent as well as a licence under the PEO Act will be integrated development.

Integrated development is intended to encourage a coordinated approach by consent authorities and approval bodies to ensure that approval bodies are involved in the development application assessment process, thus avoiding the prospect that a project will receive development consent but that subsequent approvals will either be refused or granted in terms inconsistent with the development consent.

For example, the consent authority for a mining operation must liaise with the EPA before granting development consent, and obtain from the EPA the "general terms of any approval proposed to be granted" by the EPA in relation to the development. Any development consent granted by the consent authority must be consistent with those general terms, and if the EPA indicates that it will not grant the relevant approval then the consent authority must refuse the development application.

Once a development consent is granted, the discretion of approval bodies will be restricted in two important respects:

(a) the approval body cannot refuse to grant an approval in respect of the development; and

(b) the initial approval must not be inconsistent with the development consent.

These restrictions only apply to an approval sought within three years after the development consent is granted.

If a dispute arises between the consent authority and an approval body, that dispute may be referred to the Premier under an existing dispute resolution mechanism in the EP&A Act.

For State significant development which is also integrated development, similar coordination must occur between the Minister (as consent authority) and other approval bodies, although the Minister has an additional power to refer to the Premier a dispute where the Minister considers that any proposed "general terms" of an approval are inappropriate.

PEPs (Protection of Environment Policies)

The Act introduces PEPs as a means of setting environmental goals, standards, guidelines, and protocols that must be taken into account (section 30) by government decision-makers (including the EPA). A PEP may be prepared by the EPA in respect of the following (section 11(4)):

- the whole or any part of the State;
- the environment generally or any part of it;
- any activity that may impact, or has impacted, on the environment;
• any form of pollution;
• any aspect of waste;
• any kind of technology or process;
• any kind of chemical or other substance that may impact, or has impacted, on the environment; and
• any matter in respect of which national environment protection measures may be made.

A PEP is made by the Governor on the recommendation of the Minister. The Minister for the Environment may also direct the EPA to prepare a draft PEP in certain circumstances, for example, to implement a national environment protection measure.

The PEO Act provides for public consultation on draft PEPs and the accompanying impact statement, which is to include a full assessment of the economic and social impact on the community (including industry) that the policy will have (section 16(2)(d)).

The creation of PEPs is of importance to the coal mining industry and industry in general because:

- the EPA must take PEPs into account before issuing an environment protection licence or notice under this Act;
- similarly, a consent authority must consider PEPs when determining a development application, and
- so must a determining authority when considering whether an activity will have a significant impact on the environment under Part 5 of the EPA Act.

Integrated licensing

Introduction

One integrated schedule

The PEO Act introduces one integrated schedule of EPA licensed activities which will replace the various schedules under the former air, noise, and water legislation. All activities to be regulated by licence are listed in Schedule 1 to the PEO Act.

Schedule 1 relevantly includes:

"Coal mines that mine, process, or handle coal and are:

(1) underground mines, or
(2) open cut mines that:

(a) have an intended production or processing capacity of more than 500 tonnes per day of coal or carbonaceous material, or
(b) disturb or will disturb a total surface area of more than 4 hectares of land by:
   (i) clearing or excavating, or
   (ii) constructing dams, ponds, drains, roads, railways or conveyors, or
   (iii) storing or depositing overburden, coal or carbonaceous material or tailings."

and:

"Coal works that store or handle coal or carbonaceous material (including any coke works, coal loader, conveyor, washery or reject dump) at an existing coal mine or on a separate coal industry site, and that:

(1) have an intended handling capacity of more than 500 tonnes per day of coal or carbonaceous material, or
(2) store more than 5,000 tonnes of coal or carbonaceous reject material except where the storage is within a closed container or building."

The inclusion of coal mining activities as scheduled activities in the PEO Act means that those activities will be subject to EPA regulation, primarily through the issue of an Environment Protection Licence to authorise and to control those activities.
One Environment Protection Licence ("EPL")

Chapter 3 of the Act provides for the issue of "environment protection licences". This is the equivalent of the former "pollution control licence" under the soon-to-be-repealed Pollution Control Act 1970. Under section 44, EPLs are required for:

- the authorisation of either or both scheduled development work and scheduled activities;
- the regulation of all forms of pollution emanating from that work or activities; and
- any water pollution.

As discussed above under 2.1, coal mining is a "scheduled activity" and will require an EPL.

It is likely that "schedule development work" will also apply to coal mining. "Scheduled development work" is defined as follows:

"Scheduled development work means work in or on any non-scheduled premises that is designed to enable scheduled activities to be carried out at the premises."

A coal mining company should ensure that any proposed scheduled development work will be authorised by the operation's EPL.

The maximum penalty for conducting scheduled coal mining activities without a licence is, in the case of a corporation, $125,000. Where the offence continues, there is a further penalty of $60,000 for each day the offence continues.

What's new about an EPL?:

Wide ranging conditions

An EPL can include wide ranging conditions which will be discussed in some depth below. By way of example, however, the conditions can require a company to undertake mandatory environmental audits or to develop a pollution reduction program.

No expiry date

A single EPL remains in force until it is suspended, revoked, or surrendered (section 77). By contrast, under the current Pollution Control Act 1970, a pollution control licence has a maximum duration of 12 months. Although an EPL would be subject to EPA variation and review, it is possible for one licence to apply over the life of a particular mine. This is a significant change in NSW's pollution regulatory regime.

Can impose a load-based licensing fee

The new Pollution Control Amendment (Load-Based Licensing) Act provides for a new load-based fee structure for EPLs. Fees will be calculated based largely on the quantity or harm caused by emissions from the activity. This is an example of the "polluter pays" principle underlying much of the new environmental legislation.
Before the EPA grants an EPL, it must consider whether the person concerned is a "fit and proper person" (section 45(f)). This is defined as meaning, primarily, whether the applicant has breached any environment protection legislation previously, and if the person who is to manage the scheduled activities or works is technically competent (see section 83).

A close look at EPL conditions:

As mentioned briefly above, EPL conditions may require the holder of the licence to undertake:

(a) the "usual" things, like monitoring and certification of activities and works regulated by the licence;
(b) mandatory environmental audits;
(c) studies into any aspect of the environmental impact of the activity or work authorised or controlled by the licence, and the development of pollution reduction programs, which may require the acquisition of land as a buffer zone around a coal mine;
(d) tradeable emission schemes. This is a new initiative, the regulation and workings of which may require some fine tuning as companies attempt to put such a scheme into practice. The PEO Act only loosely explains the scheme as including the following elements, though this list is not exhaustive:
   i) the determination of aggregate limits on any form of pollution;
   ii) monitoring and reporting levels of pollution;
   iii) the creation and cancellation of tradeable emission permits or credits;
   iv) the rights and duties of holders of tradeable emission permits or credits; and
   v) the initial sale or allocation and further sale or allocation of tradeable emission permits or credits.
(e) financial assurances in the form of a bank guarantee or a bond, for example;
(f) remediation work;
(g) maintenance of an insurance policy for the payment of costs of any clean-up work, and for claims for compensation for damages resulting from pollution related to the activity authorised by the EPL;
(h) arrangements for the registration of a positive covenant, which will ensure that specified requirements of an EPL condition will run with the land, binding subsequent landowners;
(i) any condition requiring that waste generated by the activity be dealt with in a particular way. This might include the preparation of an environmental waste management plan, for example. Such a condition might also address the issue of transport of waste.

The list of possible conditions is non-exhaustive.

The aim of the new environment protection licensing system is to control more effectively the whole activity (coal mining, in our circumstances), and not simply to regulate some final discharge point, and allowable concentrations and volumes of pollutants.

EPLs – comments

While EPLs do not expire, companies should be aware that the EPA maintains some powers over the provisions of an EPL.

Firstly, the EPA is required by the Act to review each licence every 3 years (section 78(1)).

Secondly, the EPA also may suspend or revoke a licence, but only if the EPA has grounds for doing so. The EPA must give the licence holder prior notice and a reasonable opportunity to make submissions in relation to the licence. Grounds for suspension or revocation of a licence may include:

- the licence was obtained improperly;
- a condition of the licence was breached; or
the licence holder failed to pay the annual fee on time.

Finally, the EPA has broad powers to vary a licence and its conditions at any time (section 58(1)). A licence is varied by notice in writing to the licence holder. However, an aggrieved licence holder may appeal to the Land and Environment Court within 21 days after being given notice of the variation.

Penalties for offences

Like the Contaminated Land Management Act, the thrust of the PEO Act is also that "polluters should pay." The PEO Act creates a tiered system of offences with particularly stiff penalties for corporations. For example, a tier 1 offence is the wilful or negligent disposal of waste in a manner that harms or is likely to harm the environment. It carries a maximum penalty of a $1,000,000 fine and/or 7 years' imprisonment. However, it is a defence if the person establishes that the offence was due to causes over which the person had no control, and the person took reasonable precautions and exercised due diligence to prevent the commission of the offence. Tier 2 offences involve water, air, noise, and land pollution, and carry a maximum $125,000 penalty for corporations.

Significantly, the PEO Act provides the Court with broad powers to order more than the payment of a fine and any costs associated with investigating and remediating the contamination, and compensating someone for damage related to the contamination. A Court may also order the offender to pay, as part of the penalty, the amount of the economic benefit that the polluter received from the commission of the offence (section 249(1)), as estimated by the Court.

Furthermore, the Court may order the offender to take specified action to publicise the offence and its environmental and other consequences (section 250(1)). This publicity could be in a newspaper, in an annual report, or any other notice to shareholders of a company.

REVIEW OF THREATENED SPECIES CONSERVATION ACT


This review made the following recommendations which should be welcomed by the coal industry:

- National Parks and Wildlife Service to review the operation of the Eight Part Test. The current Test was deemed too difficult to operate. The review should be available for comment by mid-1998.
- A review of requirements relating to Species Impact Statements, in particular the accreditation of persons who prepare the statements and the development of guidelines for the preparation of statements.
- National Parks and Wildlife Service to place the emphasis of its efforts on community education, consultation and co-operation rather than on the prosecution of offenders.

Under the new environmental regime in New South Wales, coal mining operations will need to be careful to comply with the requirements of their EPLs, as well as with general legislative requirements, such as mandatory disclosure of pollution incidents. The legislation's focus on public accountability also calls for an ongoing commitment to transparency in a company's environmental management, and most likely, to ongoing public relations work.
Study of Waste Water Quality Management in Illawarra Coal Mines

R N Singh¹, H B Dharmappa¹ and M Sivakumar¹

ABSTRACT

This paper is concerned with two case histories of wastewater quality management in underground coal mines in the Illawarra region. The first investigation briefly presents an analysis of mine water discharge having an extremely high concentration of suspended solids and consistently high barium concentrations, averaging 14.4 mg/l Barium, over the sampling period. A laboratory study of chemical precipitation processes has indicated that about 91% of barium could be removed by using ferric sulphate and lime. On the basis of the information obtained from the environmental audit process an alternative water treatment and reuse system incorporating 51% reduction in the water consumption with 32% less off-site discharge has been suggested (Thomas, 1995).

The second case history is concerned with the storm water management at a mine situated in the Illawarra escarpment where only 20% of the wastewater generated in the colliery is discharged off-site. Computer modelling of the storm water system showed that 75% of the clean runoff becomes contaminated through poor management practices and causes the process wastewater treatment system to fail in wet weather. Suggested improvements include relatively simple alteration to the coal wash filtration dams which are expected to reduce the periods of inefficient operation of these dams by 95%. The use of storm water diversion channels and detention basins can reduce the overflow volumes by 70 - 100 % for a ten year ARI (Average Recurrence Interval) storm event (Wingrove 1996).

INTRODUCTION

Coal mining activities invariably cause environmental problems when contaminated mine water is discharged to environmentally sensitive receiving waters in the Illawarra Region, NSW, Australia. There are 12 coal mines currently in operation in the Southern Coal fields producing approximately 13.35 million tonnes of saleable coal per year. The coal field is the major producer of hard coking coal, which is utilised in the coke ovens in Port Kembla and Whyalla Steelworks and exported to Japan and Europe. Most coal mines in the region are located in the catchment area of the water authority and discharge their effluent to creeks and water courses under licensing conditions imposed by the Environmental Protection Authority (EPA) of New South Wales. In order to meet increasingly stringent water quality guidelines of the EPA and high environmental standards expected by the local community, the mining industry has established a regular program of monitoring and testing mine water effluent. In addition, occasional mine water audits are carried out for characterising the sources of waste water in the colliery and assessing the efficacy of current wastewater treatment processes. Mass balance of water input and discharge from various mining operations and industrial processes are carried out to identify areas of unexplained losses and sources of wastes. The treatment technologies, in plant controls, and wastewater reduction and reuse methods are assessed.

This paper describes research studies concerned with mine water quality management in two mines, one is located in the tablelands about 40 km from the coast and the other located in the escarpment within the Illawarra region.

GENERAL QUALITY OF MINE WATER DISCHARGE IN THE ILLAWARRA COAL MINES

It is known that the mine effluent quality varies significantly from mine to mine in the Illawarra region (Singh 1994, ¹Department of Civil, Mining and Environmental Engineering, University of Wollongong)
Sivakumar et al, 1992, Singh, et al, 1995). The discharge licence conditions also vary from mine to mine depending on the source and receiving waters. The colliery water discharge licence conditions typically require that the selected water quality parameters should be monitored at a minimum of monthly intervals to meet the following conditions:

- **Five day Biochemical Oxygen Demand**: < 20 mg/L
- **Chemical Oxygen Demand**: Not Specified but a target is set at (50 mg/L)
- **Non-filterable Residue**: < 30 mg/L
- **Grease and Oil**: < 10 mg/L
- **pH**: 6.5 - 8.5

**MINE WATER QUALITY AUDIT - A CASE HISTORY OF MINE A**

**Site description**

The Colliery concerned is situated about 60 km north west of Wollongong where underground mining operations started in 1970. The average coal production from this mine is about 2 million tonnes per annum. The surface facilities at the mine occupy three separate areas as follows:

1. The main site contains the access shaft (No. 3 Shaft), the administration buildings, pit head bath, workshop, washery and coal stockpiles and coal loading and handling facilities. All are situated within a rail loop just west of Sydney-Melbourne main railway line.
2. The reject tips are located east of the rail loop and occupy a large coal refuse disposal area. Because of their size and exposure to weather, the waste stockpiles are prone to water and wind erosion. In the waste tip area, the soil overburden is removed and replaced with the coal refuse from the washery. The waste is then compacted, progressively rehabilitated and revegetated.
3. The No. 2 shaft site is located about 3 km north east of the railway loop.

The water discharged from the mining complex comes from seven major sources as shown in Fig. these being:

1. mine water from three pumps,
2. water from surface amenities and storm water runoff near the office block,
3. surface run-off and storm water runoff from coal stock piles, conveyor belt spray and waste dump area,
4. air compressor,
5. plant wash down bay,
6. gas drainage plant, and
7. water from washery plant and tailings dam.

The site concerned has three EPA (NSW) licenced discharge points. Licence No. 1 is located on the property boundary down stream from the final settlement dam 4. Licence No. 2 is located down stream of the final treatment dam near Shaft No. 2. Licence No. 3 is located at the reject disposal area, adjacent to reject loading bin. In addition to these three licenced discharge points, a non- licenced discharge point is located near the coal stockpile and silt drying area towards the southern side of the railway loop (Singh et al, 1996).
Fig. 1 - Schematic diagram of mine wastewater treatment system at Mine A

Wastewater quality audit

There are 12 water sampling and monitoring points where the water quality is monitored at 3-monthly intervals. The parameters measured are pH, electrical conductivity, non-filterable residue, total dissolved solids and barium. Water quality monitoring at 6-monthly intervals is also carried out at two selected sites (points 3 and 4 in Table 1) where, in addition to the above parameters, BOD concentration and Faecal Coliform counts are monitored. Table 1 shows chemical characteristics of the water from mine A.
Table 1 - Water quality analysis results of the mine site (Thomas, 1995)

<table>
<thead>
<tr>
<th>Sampling Point</th>
<th>PH</th>
<th>EC</th>
<th>NFR (mg/l)</th>
<th>TDS (mg/l)</th>
<th>BOD (mg/l)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Town water supply</td>
<td>7.3</td>
<td>156</td>
<td>&lt; 1.0</td>
<td>690-1190</td>
<td>-</td>
</tr>
<tr>
<td>3. Discharge from maturation Dam 1 to maturation Dam 2</td>
<td>8.9-9.9</td>
<td>370-1220</td>
<td>27-132</td>
<td>228-284</td>
<td>18-85</td>
</tr>
<tr>
<td>4. Discharge from conveyor belt spray &amp; central coal Stockpile silt trap</td>
<td>7.5-8.2</td>
<td>254-372</td>
<td>550-1500</td>
<td>137-254</td>
<td>-</td>
</tr>
<tr>
<td>5. Discharge from air compressor</td>
<td>6.2-7.3</td>
<td>797-2180</td>
<td>2-12</td>
<td>104-1403</td>
<td>-</td>
</tr>
<tr>
<td>6. Washdown effluent</td>
<td>8.8-12</td>
<td>403-1030</td>
<td>702-850</td>
<td>254-820</td>
<td>-</td>
</tr>
<tr>
<td>Effluent from Dam 1</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Effluent from Dam 2</td>
<td>7.3-8.1</td>
<td>878-1427</td>
<td>31-140</td>
<td>311-740</td>
<td>-</td>
</tr>
<tr>
<td>7. Effluent from Dam 3</td>
<td>7.6-8.4</td>
<td>638-1452</td>
<td>9-55</td>
<td>397-760</td>
<td>-</td>
</tr>
<tr>
<td>8. Washery effluent</td>
<td>8.5-8.7</td>
<td>957-1424</td>
<td>54-196</td>
<td>608-968</td>
<td>-</td>
</tr>
<tr>
<td>9. Dam 4 discharge (Licence 1)</td>
<td>7.9-8.5</td>
<td>1015-1441</td>
<td>2-23</td>
<td>550-980</td>
<td>-</td>
</tr>
</tbody>
</table>

Further, a two yearly testing programme is carried out at six selected stations where complete water analysis is conducted including the determination of 32 physical and chemical parameters. The sampling locations are designated as follows:

- Mine water
- Licence discharge 1
- Creek upstream of licence 1 discharge
- Creek downstream of licence 1 discharge
- River upstream of Discharge point
- River downstream of Discharge point

A complete water analysis was necessary to assess the performance of wastewater treatment and general water quality management at the site. These parameters are also required to ensure compliance with discharge requirements under the Clean Waters Act (1970).

A typical result for 1994 is given in Table 2 where the chemical constituents of water are given milli-equivalents per litre and in terms of their cation ratio for different water sources. The cation concentrations of water samples are calculated in milli-equivalents by dividing the concentration in milligram/litre by equivalent weight of the ion under considerations.

The results of these 6 discharge points as shown in Table 2 indicate that B, D and F belong to one group of water, while samples A, C and E to another group with similar chemical characteristics. This indicates that the characteristics of water in the creek and the river are influenced by the Licence 1 discharge. Although mine water in terms of quantity forms a major part of Licence 1 discharge, it shows no resemblance because:

1. Process water has a disproportionate effect on the cation component of the Licence 1 discharge.
2. Cation component of wastewater undergoes changes during retention in the settlement dams for a period of 7 days.
3. Cation component of the mine water is variable.
Table 2 - Wastewater classification at the mine site

<table>
<thead>
<tr>
<th>Parameters</th>
<th>Mine water meq/l</th>
<th>Licensed discharge meq/l</th>
<th>Creek upstream meq/l</th>
<th>Creek downstream meq/l</th>
<th>River upstream meq/l</th>
<th>River downstream meq/l</th>
</tr>
</thead>
<tbody>
<tr>
<td>Aluminium</td>
<td>0.004</td>
<td>0.015</td>
<td>0.137</td>
<td>0.101</td>
<td>0.036</td>
<td>0.051</td>
</tr>
<tr>
<td>Calcium</td>
<td>0.659</td>
<td>1.148</td>
<td>0.085</td>
<td>1.262</td>
<td>0.130</td>
<td>0.379</td>
</tr>
<tr>
<td>Copper</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Magnesium</td>
<td>1.517</td>
<td>0.88</td>
<td>0.241</td>
<td>0.971</td>
<td>0.296</td>
<td>0.485</td>
</tr>
<tr>
<td>Sodium</td>
<td>3.349</td>
<td>11.397</td>
<td>0.783</td>
<td>6.873</td>
<td>0.739</td>
<td>3.393</td>
</tr>
<tr>
<td>Iron</td>
<td>0.390</td>
<td>0.057</td>
<td>0.036</td>
<td>0.025</td>
<td>0.050</td>
<td>0.043</td>
</tr>
<tr>
<td>Manganese</td>
<td>0.050</td>
<td>0.009</td>
<td>&lt;0.001</td>
<td>0.004</td>
<td>0.003</td>
<td>0.003</td>
</tr>
<tr>
<td>Nickel</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Potassium</td>
<td>0.072</td>
<td>0.317</td>
<td>0.087</td>
<td>0.315</td>
<td>0.054</td>
<td>0.118</td>
</tr>
<tr>
<td>Zinc</td>
<td>0.002</td>
<td>0.003</td>
<td>0.002</td>
<td>0.002</td>
<td>0.002</td>
<td>0.002</td>
</tr>
<tr>
<td>Chloride</td>
<td>4.823</td>
<td>2.426</td>
<td>0.790</td>
<td>0.903</td>
<td>2.200</td>
<td>1.213</td>
</tr>
<tr>
<td>Sulphate</td>
<td>0.333</td>
<td>0.25</td>
<td>0.104</td>
<td>0.562</td>
<td>0.042</td>
<td>0.167</td>
</tr>
<tr>
<td>Total Anion</td>
<td>8.156</td>
<td>2.676</td>
<td>0.894</td>
<td>1.456</td>
<td>2.242</td>
<td>1.380</td>
</tr>
<tr>
<td>Cl/SO₄</td>
<td>14.483</td>
<td>9.704</td>
<td>7.596</td>
<td>1.607</td>
<td>52.38</td>
<td>7.263</td>
</tr>
<tr>
<td>Mg/(Mg+Ca)</td>
<td>0.697</td>
<td>0.434</td>
<td>0.716</td>
<td>0.435</td>
<td>0.695</td>
<td>0.510</td>
</tr>
<tr>
<td>Sodium/Cl Cation</td>
<td>0.525</td>
<td>0.825</td>
<td>0.582</td>
<td>0.719</td>
<td>0.564</td>
<td>0.758</td>
</tr>
</tbody>
</table>

Note: meq/l = milli equivalents per litre

Characteristics of wastewater

Interpretation of the wastewater sampling results in Table 1, and examination of the mine water discharge shows that the mine water exhibits a near neutral pH averaging 6.87 over the sampling period and relatively high conductivity and total dissolved solids (TDS). The conductivity and the TDS levels enable the water to be classified in Class 3, that is characterised the water as highly saline, which can not be used for irrigation on soils that are not freely draining. The suspended solids content (NFR) of the mine water was variable ranging from 39 to 390 mg/l and the suspended solids were usually reddish brown in colour at low concentration and blackish at high concentrations.

The treated discharge from the sewage plant showed near neutral pH averaging 7.5 and low suspended solids content ranging from 25 to 45 mg/l. The discharge had low to medium conductivity and medium total dissolved solids, thus placing it as Class 2, Medium Saline Water. This water is suitable for irrigating soils of moderate draining characteristics. The BOD₅ of the domestic wastewater was slightly higher, ranging from 28 to 54 mg/l, than levels expected for sewage that has undergone secondary treatment.

The discharge from the first maturation pond exhibited a very high mean pH value of 9.4 over the sampling period and low to high suspended solids ranging from 27 to 132 mg/l. The increase in NFR compared to the discharge from the sewage treatment plant can be attributed to the heavy growth of algae in maturation pond 1. Conductivity and TDS levels enabled this discharge to be classified as the Sewage Treatment Plant effluent. The BOD₅ of the effluent is variable ranging from 18 to 85 mg/l.

The pH of wastewater discharged from the conveyor belt and central stockpile was near neutral, ranging from 7.5 to 8.2. The suspended solid content of the wastewater discharge before entering the silt traps was very high, ranging from 55 to 1500 mg/l and consisting of very fine coal particles. The water also had a visible oil slick on the surface and low TDS content, placing it in the Low Salinity category, suitable for irrigation over a range of soils. The salinity of this discharge indicated that the coal fines are not a major contributing factor to the salinity of the wastewater in the colliery.

The wastewater from the machinery wash down bay displayed high pH ranging from 9 to 12. Suspended solids content
were also extremely high (850 mg/l) for discharge exiting from a washdown silt trap. High conductivity and TDS levels characterise this effluent in the class 3 high saline water, which can be used for irrigating only on freely draining soils.

Gas plant discharge was of near neutral pH averaging 6.7 for the sampling period and had very low suspended solid (6 mg/l). Conductivity and TDS contents were moderate to high, placing the wastewater in Class 3, high saline water.

Washery discharge was characterised by a high pH (average 8.6) water, containing very high suspended solids (54-196 mg/l) comprising very fine coal particles. Conductivity and TDS levels were high placing the wastewater in Class 3. The discharge exhibited visible frothing indicating the presence of surfactants (Thomas 1995).

The licence 1 discharge was measured as having a relatively high pH for the sampling period, averaging 8.2 which is within the stipulated colliery's discharge limit of 8.5. Suspended solid levels were low, ranging from 2 to 23 mg/l. Conductivity and TDS levels place the discharge in Class 3 (high salinity water) which is suitable for irrigation of soils with freely draining properties.

Barium investigations

Wastewater discharged from the mine site under investigation displayed high barium contents which could raise the barium levels of receiving river water. The host river for the mine water discharge is rated as Class P (Protected Water) which limits the barium content in the effluent to 1 mg/l. This limit is regularly exceeded by discharges from dam 4 (licence 1) and dam 6, stockpile area. In the period from January 1994 to February 1995, the barium concentration in Dam 4 and Dam 6 discharges averaged at 2.54 mg/l and discharge averaged at 2.81 mg/l. Options of Barium discharge levels in the receiving water are currently under review by the EPA. Table 3 presents a typical result of barium analysis in the mine wastewater circuit in the colliery with a view to isolate the source of barium in the mine water discharge.

### Table 3 - Barium analysis results in the mine wastewater (Thomas, 1995)

<table>
<thead>
<tr>
<th>Sample points</th>
<th>pH</th>
<th>EC (ms/cm)</th>
<th>NFR (mg/l)</th>
<th>Barium (mg/l)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Town water</td>
<td>7.10</td>
<td>109</td>
<td>&lt;1</td>
<td>0.14</td>
</tr>
<tr>
<td>2. Licence 1</td>
<td>8.20</td>
<td>1472</td>
<td>3</td>
<td>3.92</td>
</tr>
<tr>
<td>4. Maturity pond discharge</td>
<td>9.94</td>
<td>458</td>
<td>41</td>
<td>0.09</td>
</tr>
<tr>
<td>5. Gas plant discharge</td>
<td>6.38</td>
<td>690</td>
<td>2</td>
<td>1.90</td>
</tr>
<tr>
<td>6. Washery discharge</td>
<td>8.54</td>
<td>1424</td>
<td>436</td>
<td>6.00</td>
</tr>
<tr>
<td>7. Coal stockpile runoff</td>
<td>8.32</td>
<td>1678</td>
<td>166</td>
<td>2.9</td>
</tr>
<tr>
<td>8. Central stockpile drainage</td>
<td>7.55</td>
<td>356</td>
<td>2282</td>
<td>11.60</td>
</tr>
<tr>
<td>9. Plant wash down</td>
<td>8.55</td>
<td>506</td>
<td>91</td>
<td>7.91</td>
</tr>
</tbody>
</table>

Source of barium in rock and coal

The amount of barium contamination in the wastewater in the colliery shown in Table 3 is variable which may be derived from a combination of sources. Table 3 also indicates that the largest contributor of barium to the colliery's wastewater is mine water, followed by washery water, plant wash down bay and central stockpile drainage. Pinning down the actual generating point is difficult. If isolation of point source was possible then a strategy of segregation and treatment option could be examined.

It is suggested that the source of barium contamination in mine wastewater might have originated from one of the following sources:

1. Natural rocks surrounding the aquifers;
2. Leachate from coal containing high levels of barium;
3. Oil based drilling fluids containing barytes as a filler; and
4. Lubricants.
A literature review has indicated that barium compounds occur as trace elements in many igneous, sandy and calcareous sedimentary rocks (Bowen, 1979; Swaine, 1990). Most coal contains barium in the form of barytes (\(\text{BaSO}_4\)) and witherite (\(\text{BaCO}_3\)). Those barium compounds found in coal can occur in mineral veins as reported by Forstner and Whittman (1979) in a colliery in Durham, U.K. Table 4 is a compilation of barium levels in selected rocks, naturally occurring water and some Australian coals. Barium content in many soils range from 100 - 1000 mg/kg, however in some geological formation such as fossil fuels much higher levels in excess of 1000 mg/kg have been reported (Bowen, 1979).

Table 4 - Barium contents of various geological materials (Adopted from Swaine, 1990; Bowen, 1979; Forstner and Whittman, 1979; and Thomas 1995)

<table>
<thead>
<tr>
<th>Minerals</th>
<th>Barium (mg/kg)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Rocks</strong></td>
<td></td>
</tr>
<tr>
<td>Granite Rock</td>
<td>420</td>
</tr>
<tr>
<td>Shales</td>
<td>850</td>
</tr>
<tr>
<td>Marine clays</td>
<td>2300</td>
</tr>
<tr>
<td>Sandstone</td>
<td>320</td>
</tr>
<tr>
<td>Limestone</td>
<td>90</td>
</tr>
<tr>
<td>Carbonates</td>
<td>10</td>
</tr>
<tr>
<td>Basalt</td>
<td>250</td>
</tr>
<tr>
<td><strong>Coal</strong></td>
<td></td>
</tr>
<tr>
<td>Latrobe valley, Victoria</td>
<td>60-800</td>
</tr>
<tr>
<td>St Vincent Basin, South Australia</td>
<td>220-440</td>
</tr>
<tr>
<td>Leigh Creek, South Australia</td>
<td>100-2000</td>
</tr>
<tr>
<td>Collie, Western Australia</td>
<td>43-519</td>
</tr>
<tr>
<td>Hunter Valley New South Wales</td>
<td>20-1500</td>
</tr>
<tr>
<td>Western area, NSW</td>
<td>20-300</td>
</tr>
<tr>
<td>Southern Coalfields, NSW</td>
<td>40-100</td>
</tr>
<tr>
<td>Site of Investigation</td>
<td>270-630</td>
</tr>
<tr>
<td><strong>Sea water</strong></td>
<td>0.013</td>
</tr>
<tr>
<td><strong>Fresh Water</strong></td>
<td>0.01</td>
</tr>
</tbody>
</table>

Chemical analysis of coal

An analysis of coal from 3 different locations within the central stockpile on two different dates using Atomic Absorption Spectroscopy has indicated that the coal from this site contains barium between 270-630 mg/kg of coal (Thomas, 1995). Tests carried out by the mine operator on the lubricants used at the site have indicated that the barium level in the oil and lubricants used are not high enough to form a major source of contamination, since the oil spillages are small in comparison to various other sources. However, the moderate to high barium content of the coal and the high barium content in the leachate from the central stockpiles indicate that coal itself may be a major contributor to barium in the colliery's wastewater. It may be observed that ground water travelling in coal aquifers would have the capacity to dissolve barium by ion exchange between ground water and coal stratum over a geological time span.

Physiological effects of barium

The physiological effects of barium on the human body have been studied by the various medical workers including Brenniman and Levy (1985). Australian Drinking Water Guidelines (1994) suggests a limit of 0.7 mg/l of barium in the drinking water. In the majority of Australian water supplies the barium concentration ranges from 0.0005 to 0.3 mg/l. In high concentrations, barium causes constriction of blood vessels, contraction of alimentary canal, convulsion and paralysis. A number of long term studies on the effects of barium on heart disease have shown that no adverse effects were found with barium concentrations in water up to 7 mg/l. In a study using a small number of volunteers, no adverse effects were observed after 12 weeks exposure to drinking water with up to 10 mg/l barium (Brenniman and Levy, 1985).

Barium removal process

Barium can be removed from the wastewater by using one the following processes:
1. Chemical precipitation;
2. Physical adsorption; and
3. Ion exchange.

Thomas (1995) carried out laboratory experiments for removing barium using chemical precipitation method. The results obtained were discussed in relation to other two methods. It was concluded that the most feasible method of reducing barium to below 1 mg/l level in the mine wastewater was the chemical precipitation method, shown in Fig. 2. Chemical precipitation process creates a sludge, which mine operators feel more comfortable in disposing of than dealing with the liquid waste. Other treatment processes, namely ion exchange and reverse osmosis methods have limitations that would require tighter process control during their operations.

![Fig. 2 - Barium removal process using chemical precipitation (Maruyama 1985)](image)

**CASE STUDY 2- STORMWATER MANAGEMENT AT MINE B**

The second underground coal mine selected for investigation was located in the escarpment area in the Illawarra region and produces some 0.4 Million tonnes of raw coal per year from continuous mining operations in the Wongawilli seam. An on-site washery produces 0.3 Million tonnes of clean coal.

**Quantity and quality management of wastewater**

The schematic layout of the current wastewater treatment system for Mine B is given in Fig. 3.

**System input**

The main sources of waste water in the colliery are from (i) mine water discharge, (ii) Washery discharge, (iii) domestic effluent from offices, bath house, loading bays and workshops, and (iv) storm water runoffs. The water requirements for various operations in the mine are given in Table 5.

**System treatment components**

The main components of the wastewater treatment system comprise a tailings dam, filter dam, an intermediate dam, a settlement dam and the main dam. Wastewater from the surface amenities first goes to a stabilisation pond before discharged into the main dam. A number of sediment traps are built in the wash down bays and the storm water systems before they enter the settlement dam.
The main sources of wastewater in the colliery are as follows:

(i) Mine water discharge - The total quantity of water discharged from underground mining operations is 3000 m$^3$/d, which includes 200 m$^3$/d of service water and 2800 m$^3$/d of aquifer water. The main pollutants of the aquifer inflow are dissolved minerals from the aquifers rock strata and non filterable residue (NFR) of 0.4 to 7 mg/l. It is not practicable to prevent the contamination of this water.

(ii) Bathhouse wastewater - The bathhouse effluent of 3 m$^3$/d is predominantly contaminated by coal fines sticking to the body of the workers and soaps used in their showers. Detergents and disinfectants are also used to clean the bathhouse. This wastewater contains NFR levels ranging from 4 to 157 mg/l.

(iii) Process (Washery) wastewater - Wastewater from the washery includes 300 m$^3$/d of liquid effluent and the slurry tailings. The liquid effluent is a result of truck washing, machinery and work area wash down and pipe leakages. As such, the wastewater generated, generally consists of a large amounts of NFR in the range of 4000 -13,659 mg/l.

Table 5 - Water requirement by the Mine B

<table>
<thead>
<tr>
<th>Activity</th>
<th>Quantity, 3 m$^3$/d</th>
<th>Quality requirements</th>
</tr>
</thead>
<tbody>
<tr>
<td>Office (drinking &amp; kitchen)</td>
<td>0.05</td>
<td>Fresh water</td>
</tr>
<tr>
<td>Workshop</td>
<td>1.0</td>
<td>Low NFR, low salts, near neutral</td>
</tr>
<tr>
<td>Bathhouse</td>
<td>3.0</td>
<td>Low NFR, low salts, near neutral</td>
</tr>
<tr>
<td>Underground operations</td>
<td>200</td>
<td>Low NFR, low salts, near neutral</td>
</tr>
<tr>
<td>Washery</td>
<td>1590</td>
<td>Low NFR, near neutral</td>
</tr>
<tr>
<td>Stockpile sprays</td>
<td>4.2</td>
<td>Low NFR, near neutral</td>
</tr>
<tr>
<td>Truck washing</td>
<td>0.17</td>
<td>Low NFR, near neutral</td>
</tr>
<tr>
<td>Road dust suppression</td>
<td>2.5</td>
<td>Low NFR, near neutral</td>
</tr>
<tr>
<td>Total</td>
<td>1800.92</td>
<td></td>
</tr>
</tbody>
</table>
(iv) Tailings dam - The slurry tailings effluent is a waste product from the coal washing process. The colliery currently sells some of these fine "rejects" as a lawn treatment material.

(v) Pit-top operations wastewater - The majority of the pit-top operational water is used to control dust. Methods to reduce the need for using water spraying to control dust include: improving the truck loading system to minimize spillage of coal products and providing windbreaks for large material stockpiles.

vi) Storm water - Storm water runoff from the area surrounding the pit head is responsible for loading the wastewater with NFR which effective makes water treatment ineffective during storm period.

System output
The colliery, currently, discharges approximately $600 \text{ m}^3/\text{d}$ of treated wastewater from the main dam. This quantity represents 20% of the volume of water removed from the underground.

Process wastewater reuse and disposal
A significant amount of colliery wastewater is already being reused for colliery operations. The aquifer inflow water meets all of the colliery's water needs with the exception of drinking and kitchen (potable) water requirements. For health reasons, it is not appropriate to use the aquifer inflow water for either of these purposes. Thus, the only option for increasing reuse levels at site is for additional non-potable purposes. The colliery rehabilitation program involves extensive revegetation of large areas of land and the aquifer inflow water would be suitable for this program. However, the volumes of water involved would not significantly reduce the quantity of off-site discharge.

Currently the colliery does not specifically make its surplus water available to external industries. The water would be suitable for use by many local industries which do not require water of potable quality for their operations such as:

- irrigation water for local farms, parks, golf courses, green belts or lawns;
- industrial cooling water;
- industrial wash down water;
- industrial boiler feed water;
- vehicle washing water;
- dust suppression water; and
- industrial and public fire fighting supplies.

The water could be conveyed on-site by pipeline or tanker trucks. Depending on the use, it may or may not be necessary for the water to be neutralised. This option of increasing off-site utilisation of the water is considered to be the most feasible and most significant method of reducing the off-site discharge of wastewater from the colliery. Treatment efficiency achieved at the settling dams is given in Table 6.

Stormwater management
The investigation into the existing stormwater management system at the colliery indicated two main problem areas:

- Hydraulic overloading of the process wastewater treatment dams during storm conditions; and
- Allowance of essentially uncontaminated runoff to become contaminated.

<table>
<thead>
<tr>
<th>Treatment dam</th>
<th>Effluent NFR concentration</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tailings dams</td>
<td>Decrease by $&gt;99%$</td>
</tr>
<tr>
<td>Filter dams</td>
<td>Decrease by $&gt;99%$</td>
</tr>
<tr>
<td>Intermediate dams</td>
<td>Slight increase</td>
</tr>
<tr>
<td>Settlement dams</td>
<td>Slight increase</td>
</tr>
<tr>
<td>Main dams</td>
<td>Decrease by $&gt;45%$</td>
</tr>
<tr>
<td>Stabilisation pond</td>
<td>Increase by $&gt;50%$</td>
</tr>
</tbody>
</table>

Table 6 - Treatment efficiency achieved in the settling dams
An improved system of stormwater management was, therefore, necessary, with the aim of reducing, or ideally, eliminating these problems. The goals for the improved system are thus to:

- Reduce the pollutant levels in contaminated runoff;
- Reduce the quantity of contaminated runoff;
- Ensure that the quality of colliery discharges is maintained; and
- Ensure that the process water treatment system efficiency is not compromised in storm conditions.

Based on the topography and land uses (Fig. 4) the land use of the colliery is classified into several sub-catchments as shown in Fig. 5. These sub-catchments are grouped together into clean and dirty regions as shown in Table 7. It should be noted that regions C1 and C2 are separated by a cliff line and C2 and C3 are separated by a ridge line. Similarly, D1 and D2 are separated by a ridge line. The grouping allows management options to be applied as it is considered more feasible to manage the runoff in regions as opposed to individual sub-catchments.

<table>
<thead>
<tr>
<th>Region</th>
<th>Runoff quality</th>
<th>Contributing sub-catchments</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>C1</td>
<td>Clean</td>
<td>1A, 2A, 4A</td>
<td>Runoff easily diverted</td>
</tr>
<tr>
<td>C2</td>
<td>Clean</td>
<td>4B</td>
<td>Runoff easily diverted</td>
</tr>
<tr>
<td>C3</td>
<td>Clean</td>
<td>1B, 7B, 9B</td>
<td>Runoff easily diverted</td>
</tr>
<tr>
<td>C4</td>
<td>Clean</td>
<td>6A, 7A, 8A</td>
<td>Runoff not easily diverted (drains by gravity to main dam)</td>
</tr>
<tr>
<td>D1</td>
<td>Dirty</td>
<td>3A, 5A</td>
<td>Runoff easily diverted</td>
</tr>
<tr>
<td>D2</td>
<td>Dirty</td>
<td>2B, 3B</td>
<td>Runoff easily diverted</td>
</tr>
<tr>
<td>D3</td>
<td>Dirty</td>
<td>5B, 6B, 8B</td>
<td>Runoff not easily diverted (contains process water treatment dams)</td>
</tr>
</tbody>
</table>

Pollution prevention of stormwater

Many management practices are available to reduce the pollutant levels in runoff. These practices are often inexpensive and relatively simple but can be very effective. Management practices appropriate for the colliery are provided below in the two categories of low and high contamination potential sub-catchments.

![Fig. 4 - Land use and pit-top operations at Mine B](image-url)
Low contamination sub-catchments

To ensure runoff from low contamination potential areas remains uncontaminated, it is imperative that the flow be diverted away from high contamination areas. This has been discussed (USEPA 1993) and can be achieved through the use of (USEPA, 1993):

- catch drains;
- interceptor dykes;
- berms;
- open channels; and
- pipelines.

Presently, runoff from area 1A is the only "clean runoff" which is diverted to prevent its contamination. Runoff from this area represents approximately 12% of the total clean runoff volume and 8% of the total runoff volume. If all of the clean runoff were diverted away from high contamination areas, the total volume of contaminated runoff would be reduced by more than 50%. This is a substantial reduction in the quantity of stormwater contamination.

Although considered "clean", runoff from low contamination sub-catchments contains soil particles. The quantity of soil particles picked up by the runoff can be reduced by:

- Increasing the vegetative ground cover. This has additional benefits of absorbing rainfall energy, roots holding soil in place, increasing absorptive capacity of the soil, reducing the runoff velocity as well as acting as a filter to catch sediments. Areas 4A, 4B, 7B and 9B are largely open grassland. The introduction of shrubs and trees is also appropriate.
- Installing straw bale barriers and check dams in diversion channels to decrease the channel flow velocity and thereby allow sediments to settle out of the flow. A reduction of channel flow velocity would also decrease any erosion caused by the flow downstream.

![Diagram](image_url)

Fig. 5 - Classification of land use and drainage routes for pit top operations
The contamination of runoff in these areas can be greatly reduced by minimising the possibility of runoff coming into contact with pollutants. Methods appropriate for the colliery suggested in (USEPA, 1993) include:

- The containment of drips, overflows, leaks or other material releases from vehicles, workshop areas, the washery, and the conveyor belt. This can be achieved through dykes, drip pans and sumps.
- Enclosing material storage areas with curbing barriers to divert runoff around the polluted areas. This is especially suitable for the washery and workshop areas. This can be supplemented by covering the areas to prevent precipitation falling into the curbed area. This, however, requires greater capital investment.
- Ensuring trucks are well positioned to minimise spillage of materials during loading and unloading operations.
- Cleaning up or recovering a substance after it has been released or spilled to reduce the potential impact of the spill before it reaches the environment.
- Controlling wind dispersion of particles through the use of water spraying, coverings and wind breaks. The colliery only has water sprays in place on its main coal product stockpile. Additional sprays should be placed on three other substantial material stockpiles which are currently unprotected from the wind. Water spraying has the advantage of confining the pollutants within an area, however it does lead to contamination of that water, which thus requires treatment.
- Trucks operating within the site should be covered in windy conditions.
- The site roads are currently water sprayed daily. It is appropriate for those which carry the heaviest traffic.

A major source of contamination for these areas is the coal product and waste material stockpiles. Due to the size of the stockpiles, methods to minimise the runoff contamination from these areas, such as covering, would be very expensive and thus considered impractical. It is, however, suggested to prevent runoff from other areas entering the stockpile areas. Runoff that discharges from the stockpile areas is highly contaminated by coal fines and should be treated. Similar arguments hold for the process water treatment dam areas.

**Stormwater management options**

The main aim of managing the clean water runoff is to ensure it remains uncontaminated. In addition, it is desirable to remove the soil loading and control the release of the runoff off site to prevent downstream siltation and flooding. The main aim of managing the dirty water runoff is to ensure it does not compromise the process water treatment system. It is also desirable to remove the coal fines load and control the release of the runoff off site to prevent downstream siltation and flooding. Management options which would achieve, or partially achieve, these goals are outlined in the following in increasing order of complexity and cost (Wingrove, 1996).
• **Option 1** - involves the use of diversion channels to collect clean and dirty stormwater runoff and convey it directly to the natural creek system. The clean and dirty water diversion channels may or may not be combined.

• **Option 2** - involves the use of diversion channels to collect clean and dirty stormwater runoff and convey it to the existing process water sedimentation dams (i.e., the intermediate, settlement or main dams).

• **Option 3** - involves the use of diversion channels to collect clean and dirty stormwater runoff and convey it to the process water sedimentation dams, where these dams have been modified to increase their maximum capacity and thus increase their freeboard volume (Table 8).

<table>
<thead>
<tr>
<th>Process water treatment dam</th>
<th>Freeboard volume, $m^3$</th>
<th>Existing</th>
<th>Additional feasible</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Intermediate dam (east)</td>
<td>600</td>
<td>2,120</td>
<td>2,720</td>
<td></td>
</tr>
<tr>
<td>Intermediate dam (west)</td>
<td>800</td>
<td>1,800</td>
<td>2,600</td>
<td></td>
</tr>
<tr>
<td>Settlement dam</td>
<td>0</td>
<td>3,000</td>
<td>3,000</td>
<td></td>
</tr>
</tbody>
</table>

**Option 4** - involves the use of separate diversion channels to collect clean and dirty stormwater runoff and convey it to purpose-built clean and dirty stormwater detention basins. Lack of suitable land due to topography and heavy capital expenditure requirements precludes this option.

**Option 5** - involves the use of separate diversion channels to collect clean and dirty stormwater runoff and convey it to purpose-built detention basins. The stormwater is slowly released into holding tanks or dams to store the clarified water for future use.

All of the above options are superior to the existing management method which allows 88% of clean runoff to become contaminated which causes the process water treatment system to fail. The diversion of all runoff away from the process water treatment dams, and in during wet weather particular the filter dams (filter dam walls can collapse and be washed downstream due to overloading) should reduce or eliminate this problem. Of these the most appropriate and cost effective option depends on the volume of runoff that is involved.

**Clean stormwater runoff management**

This section quantifies the volume of clean stormwater runoff, which is considered capturable and determines the detention times required for the soil particles to be removed from this runoff.

**Volume of Diverted Runoff** - Ideally, all clean runoff should be captured or diverted. This is somewhat unrealistic due to the topography of the colliery site and the practical locations of diversion channels. The total capturable volume represents approximately 75% of the total volume of runoff from low contamination areas. The total volume of runoff discharged from the four clean regions is summarised in Table 9. Detailed calculations are provided in Wingrove (1996).

**Solids Removal** - To remove soil particles from stormwater a detention time of 2 hours is typically used (Field et al., 1993). Considering the storm duration modelled and the peak flow rates (Wingrove, 1996) the detention volumes which are estimated to be required for each of the clean regions are summarised in Table 9.

**Dirty stormwater management**

This section quantifies the volume of dirty stormwater runoff that is considered capturable and determines the detention times required for the coal fines to be removed from this runoff. The quantification is based on the runoff volumes (Wingrove, 1996). Similar to that of clean stormwater management discussed earlier, the capturable volume of the dirty stormwater is summarised in Table 10. The detention time required for each region is summarised in Table 11.
Table 9 - Total discharge volume in low contamination regions

<table>
<thead>
<tr>
<th>Regions</th>
<th>Contributing Sub-Catchment</th>
<th>Total inflow m³</th>
<th>Detention volume, m³</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Average Recurrence interval</td>
<td>20yr</td>
</tr>
<tr>
<td>C1</td>
<td>1A, 2A, 4A</td>
<td>17,722</td>
<td>9371</td>
</tr>
<tr>
<td>C2</td>
<td>4B</td>
<td>6,342</td>
<td>3355</td>
</tr>
<tr>
<td>C3</td>
<td>1B, 7B, 9B</td>
<td>31,634</td>
<td>16,724</td>
</tr>
<tr>
<td>C4</td>
<td>6A, 7A, 9A</td>
<td>10,933</td>
<td>5812</td>
</tr>
</tbody>
</table>

Table 10 - Total discharge volumes in high contamination regions

<table>
<thead>
<tr>
<th>Regions</th>
<th>Contributing Sub-Catchment</th>
<th>Total inflow m³</th>
<th>Detention volume, m³</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Average Recurrence interval</td>
<td>20yr</td>
</tr>
<tr>
<td>D1</td>
<td>3A, 5A</td>
<td>17,161</td>
<td>9162</td>
</tr>
<tr>
<td>D2</td>
<td>2B, 3B</td>
<td>9003</td>
<td>4835</td>
</tr>
<tr>
<td>D3</td>
<td>5B, 6B, 8B</td>
<td>21,724</td>
<td>11,096</td>
</tr>
</tbody>
</table>

Improved stormwater management

A preferred method of management of the clean and dirty stormwater runoff is shown in Fig. 6. This method is selected based on the following assumptions;

- a combination of the five options outlined in a previous section.
- runoff volumes for minimum 10 year ARI period
- the topography permits the location of the diversion channels and the detention basins.
- detention volumes are based on a minimum of 2 hour detention time for 10 year ARI storms.

Table 11 - Detention basin design parameters

<table>
<thead>
<tr>
<th>Influent Region</th>
<th>Basin location</th>
<th>Discharge to</th>
<th>Max. capacity (m³)</th>
<th>Surface area (m²)</th>
<th>Avg. depth (m)</th>
<th>Detent. time (hr) 20 yR ARI</th>
<th>10 yR ARI</th>
<th>5 yR ARI</th>
<th>Overflow rate (m³/hr) 20 yR ARI</th>
<th>10 yR ARI</th>
<th>5 yR ARI</th>
</tr>
</thead>
<tbody>
<tr>
<td>C1</td>
<td>4A</td>
<td>off site</td>
<td>300</td>
<td>200</td>
<td>1.5</td>
<td>1.83</td>
<td>2.29</td>
<td>3.07</td>
<td>0.82</td>
<td>0.65</td>
<td>0.49</td>
</tr>
<tr>
<td>C2</td>
<td>4B</td>
<td>off site</td>
<td>80</td>
<td>80</td>
<td>1.0</td>
<td>2.01</td>
<td>2.51</td>
<td>3.35</td>
<td>0.50</td>
<td>0.40</td>
<td>0.30</td>
</tr>
<tr>
<td>C3</td>
<td>9B</td>
<td>off site</td>
<td>350</td>
<td>233</td>
<td>1.5</td>
<td>1.77</td>
<td>2.21</td>
<td>2.95</td>
<td>0.85</td>
<td>0.68</td>
<td>0.51</td>
</tr>
<tr>
<td>D1</td>
<td>5A</td>
<td>settlement dam</td>
<td>270</td>
<td>180</td>
<td>1.5</td>
<td>1.70</td>
<td>2.11</td>
<td>2.76</td>
<td>0.88</td>
<td>0.71</td>
<td>0.54</td>
</tr>
<tr>
<td>D2</td>
<td>3B</td>
<td>intermedi. dam</td>
<td>100</td>
<td>100</td>
<td>1.0</td>
<td>1.77</td>
<td>2.18</td>
<td>2.82</td>
<td>0.56</td>
<td>0.46</td>
<td>0.36</td>
</tr>
</tbody>
</table>

Detention basin design

The main design considerations for stormwater detention basins are the detention time and overflow rate. The detention volumes established in Tables 9 and 10 are based on a 2 hour detention time. An appropriate basin volume is adopted using the 10 year ARI detention volume as the minimum design volume. The detention time for each basin is thus greater than 2 hours for the 5 and 10 year ARI storms and slightly less than 2 hours for the 20 year ARI storm. To determine the area and depth of the basins the overflow rate design criteria is used. In this criteria, it is desirable to have
the overflow rate \( (v_0) \) of the detention basin to be less than the settling velocity \( (v_s) \) of the particles in the stormwater. The settling velocity of the soil particles has been estimated to be 0.98 m/hr. Detailed calculations are provided in Wingrove (1996). The surface areas of the detention basins have been adopted such as to ensure \( v_0 \) is less than \( v_s \). Table 11 summarises the features of the suggested detention time. It should be noted that although five new detention basins are suggested to be constructed, the relatively small volume of the basins would result in low construction costs. Construction could be carried out by plant equipment already owned by the colliery. The detention basins which collect the clean stormwater runoff could be omitted and the net effect on the natural creek system would be superior to the effect resulting from the existing stormwater management methods. However, the benefits of detention basins are considered to far outweigh the costs, and thus their use is highly recommended.

Effect on process water treatment system

By implementing the measures outlined above, a substantial quantity of stormwater would be diverted away from the process water treatment dams. This would significantly reduce the hydraulic loading of these dams and thus the wet weather efficiency would approach the dry weather efficiency. Table 12 summarises the percentage reductions of the volume of stormwater discharged into the process water dam sub-catchments and the corresponding reductions in overflow volumes from these sub-catchments.

The following points can be noted from Table 12:

- The overflow volumes from all dams would be substantially reduced by the improved stormwater management.
- The existing method of stormwater management is considered to cause the process water treatment dam to fail. Under the improved method, the process water treatment system would maintain acceptable efficiency for even the 20 year ARI storm.
- For the 5 year ARI storm, there would be no overflow from the process water treatment dams.
- For the 10 year ARI storm, there would be no overflow from the intermediate dams and the filter dams. The volume of overflow from the main and settlement dams would be reduced by over 70% compared to the overflow which results from the existing management.
- For the 20 year ARI storm, the overflow volume from the filter dams would be reduced by 95% compared to the overlap which results from the existing management. It is particularly important to maintain the treatment efficiency of the filter dams as they play a very significant role in the removal of NFR from the process wastewaters. The overflow volume from the intermediate dams would be reduced by over 70% and the overflow from the main and settlement dams would be reduced by approximately 50%.

<table>
<thead>
<tr>
<th>Sub-catchment</th>
<th>Process water treatment dams</th>
<th>Increase in process water treatment dams freeboard volume, (%)</th>
<th>Reduction of stormwater discharge into sub-catchment (%)</th>
<th>Reduction of process water treatment dams overflow volumes (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>20 yr ARI</td>
<td>10 yr ARI</td>
<td>5 yr ARI</td>
</tr>
<tr>
<td>8A</td>
<td>Main &amp; settlement</td>
<td>43</td>
<td>28</td>
<td>27</td>
</tr>
<tr>
<td>5B</td>
<td>Filter dams</td>
<td>0</td>
<td>65</td>
<td>65</td>
</tr>
<tr>
<td>6B</td>
<td>Intermediate dams</td>
<td>280</td>
<td>54</td>
<td>-17</td>
</tr>
</tbody>
</table>

The above significant decreases in overflow volumes indicate notably improved wet weather efficiency of the process water treatment system. The corresponding reduced impact on the receiving natural creek environment would also be significant.

CONCLUSIONS

The waste auditing technique provide a powerful tool to assess periodically the efficacy of the mine wastewater treatment
system. This will provide an opportunity to the mine operators to the change the mining and processing conditions so that the environmental and economic goals can be achieved. This technique has been successfully applied to a mine site in the Illawarra region where wastewater of dissimilar chemical characteristics could be segregated into separate streams for further treatment.

The wastewater auditing technique has enabled identification of the presence of barium in the mine wastewater. Based on the wastewater monitoring, and the chemical analyses of coal, it has been concluded that the barium in the wastewater is originated from coal. Laboratory assessment of various barium removal options has indicated that the chemical precipitation method is a suitable option for Mine A. The wastewater quality monitoring method has also indicated that the site needs to upgrade its NFR treatment system in case of heavy storm events. A new flow sheet of mine wastewater treatment strategy is developed by Thomas (1995) which allows considerable reuse of water for dust suppression, thus reducing the freshwater consumption by about 50%.

The second case history at Mine B utilised the concept of 'source reduction' to segregate the stormwater into clean and dirty components. The dirty stormwater is then proposed to be diverted using diversion channels and treated with detention basins. These modifications were found to reduce the overflow volumes of the process wastewater treatment dams in 5 year average recurrence interval (ARI) storms by 100%, with reductions of 70% to 100% achievable for a 10 year ARI storm.

Improved process water management systems are also proposed. Relatively simple alterations to the operation of the coal wash filtration dams are expected to reduce the periods of inefficient operation of these dams by 95%. As highlighted in this paper, often there is significant economic benefit resulting from the application of waste minimisation. In addition, there is always a major benefit to the environment.

ACKNOWLEDGMENTS

The research work reported in this paper forms a part of BE (Honours) thesis of G.L. Thomas, Kate Wingrove and Sally Thompson supervised by the authors. The original contribution of the honours students to this paper is gratefully acknowledged. Thanks are due to the Management of Mining Companies for allowing access to their mining operations and making available their facilities. Thanks are also due to Mr D. Olsen and Mr K. Woodley for providing access to the mine site and participating in discussions throughout the investigations. However, the opinions expressed in this paper are responsibilities of the authors alone and not necessarily that of the management.

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Co-Disposal of Coarse Coal Reject with Sand Mining Reject for the Control of Metal Concentration in Runoff Water

C F Gosling¹, S J Riley¹ and C V McQuade²

ABSTRACT

Acid mine drainage is an unavoidable consequence of some coal mining operations. Typically runoff pH is below 3.5 and at these pH levels heavy metals are mobilised. Leachate from coal reject dumps may require collection and treatment to raise the pH and precipitate the metals before being discharged.

A bi-product of coal mining operations at Clarence Colliery is coarse washery reject. At present the coarse reject is deposited above ground and rehabilitated. Adjacent to Clarence Colliery is the Kable’s Transport Pty Ltd sand mining operation. It has been proposed that co-disposal of the rejects from both operations may produce a product whose leachate has near neutral pH.

The University of Western Sydney undertook laboratory experiments to investigate the chemistry of leachate water from 22 co-disposal options. Reject material was placed in 205L drums. Each drum contained coarse reject and either sand or clay, either mixed or layered. Three control drums were used, being 100% coarse reject, 100% sand and 100% clay.

Deionised water was introduced to the co-disposed material at approximately 4mL/min for one month and the leachate tested for pH and conductivity. On two occasions samples were collected and analysed for metals concentration using Inductively Coupled Plasma Mass Spectrometry. This paper presents the results of the metals analysis, comparing the materials, quantities and modes used for co-disposal.

INTRODUCTION

Acid Mine Drainage (AMD) is produced when sulphide materials are exposed to water in oxygenated conditions. Catalysed by the bacteria Thiobacillus ferrooxidans (Paulin, Patterson and Hadjigeorgiou, 1994) the acid producing reactions cause a drop in pH which increases the presence of metals in leachate.

Clarence Colliery is located on the Newnes Plateau near Lithgow, (NSW) and mines about two million tonnes of thermal coal per annum. Ten percent of the washed coal is rejected, this reject is placed above ground. Tailings are extracted and blended with the washed product and sold. The coarse reject is generating acidic drainage with associated elevated levels of some heavy metals. Water treatment systems are in place at Clarence to treat the leachate to meet Environmental Protection Authority guidelines. However, they are expensive to establish and operate. An alternative long-term passive method for dealing with the reject and its bi-products is needed.

Adjacent to Clarence Colliery is Kable’s Transport sand mine which produces bi-products of silts, clays and fine sand tailings. The poor structural stability of the tailings and the large voids created by the mining process cause rehabilitation difficulties for Kable’s.

Disposal of coal washery reject in a manner having minimal environmental impact depends upon the mine and its location. Neutralisation using lime or fly ash additive is uneconomical for Clarence and is unlikely to provide long term prevention of acidic drainage as lime leaches out of the rejects before neutralising all the sulphides (Phipps, Fletcher and Skousen, 1996).

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² Clarence Colliery, Lithgow, Australia
A wetland treatment system may assist in neutralising some acidic leachate, but ongoing management would be required.

It has been proposed that co-disposal of the rejects from both operations may produce a product whose leachate has near neutral pH, thus limiting mobilisation of metals. Potential benefits of the co-disposal of sand mine tailings and coarse coal reject with regard to the leachate are:

1. Chemical reactions between the coarse washery rejects and sand mining tailings may reduce the acid generation. The tailings may adsorb metals.
2. Lowering infiltration rates through the reject will reduce the oxygen and water input to the dump and hence reduce AMD.

Clay layers are effective barriers for infiltration and adsorb toxic elements such as heavy metals. Al-Hashimi et al (1995) found that solvated elements are retained in clay barriers by adsorption and precipitation processes. With a geobarrier, such as clay, the reactions at the clay surface effectively raise the pH and precipitate the metals. Similar reactions can occur with sand barriers except that the interactions are related to metal complexes on the sand grains (Manahan, 1991).

This paper presents the results of a study of the metal concentration of leachate from co-disposal options in a controlled laboratory experiment. The aim is to determine the option which produces the lowest load and concentration of metals in leachate.

**METHOD**

Sand tailings and coarse reject can be disposed of in one of two ways, either by layering or by mixing. The coal washery reject and sand tailings product quantities and disposal options gave 22 combinations of materials for laboratory experiments, discussed in Gosling et al (1997). The percentage of coarse reject by weight for each of the sand and clay, and for each co-disposal method was selected to be 100%, 90%, 80%, 70%, 50%, 20% and 0%. Consideration of production rates of coal reject to sand and clay tailings lead to the higher proportion of coarse reject in the experiment. Three references, 100% of each material (coarse reject, sand and clay tailings), were used to compare the leachate metals concentration from the co-disposal options. The feasibility experiment was carried out using 205L drums. The drum and water arrangement is shown in Fig 1. Acid generation is minimised by elimination of either water or oxygen from the system. In order to assess the effectiveness of the co-disposal non-saturated conditions were used in the experiment. Details of the experiment are presented in Gosling et al (1997).

The experiment was monitored for one month. Daily measurements of leachate pH and conductivity were taken using portable pH and conductivity meters. Leachate samples for metals analysis were collected two weeks after the experiment commenced (15th November 1996) and at the end of the month (28th November 1996).

Leachate metals analysis was undertaken using Inductively Coupled Plasma Mass Spectrometry (Eaton et al, 1995). The samples were analysed for aluminium, arsenic, cadmium, cobalt, chromium, copper, iron, mercury, manganese, nickel, lead, antimony, selenium and zinc. These elements had been identified as elevated in mine water from a 1996 study by Jones and Eames (1996). The results were compared to Australian and New Zealand Environment and Conservation Council (ANZECC) (1992) guidelines for discharge into fresh waters and to metals concentration in the Wollangambe River upstream of Clarence Colliery (Tables 2 and 3). The purpose of the work was to ascertain whether there was a potential environmental issue with metals in reject leachate by an initial assessment of leachate metal concentration. It was not an extensive study.
The metal load from each experiment was calculated by multiplying leachate volume for each 24 hour period by the metal concentration determined from statistically significant relationships between leachate conductivity and metal concentration. Thirty day total metal loads were calculated for each co-disposal option. Comparison was made between options having low 30 day load and low metal concentration to assess which option would produce leachate having minimal impact upon receiving waters. It is desired that the metal load into receiving waters be minimal. Co-disposal options having elevated leachate metal concentration would have less impact upon receiving waters if the total quantity of pollutants is low.

A means of checking for possible failure in the drum lining and contamination from the drum was undertaken by comparing iron concentration and pH (Adeloju, 1997). It is known that iron is soluble at low pH. As the pH rises the solubility decreases. Thus high concentrations of iron at high pH's would suggest contamination from failure of the liner and the steel drum. Plots of leachate pH against leachate iron concentration (Fig. 2) suggest contamination from the drum has occurred for the trials 50% coal reject mixed and layered with 50% clay. There is possible contamination in the 70% coal reject layered with 30% clay readings and for 90% coal reject layered with 10% clay on 28th November. The 50% coal reject and 50% clay drums had high iron concentration, above 7570 mg/L, and high pH, above 6.0, for all samples. Seventy percent coal reject layered with 30% clay had iron concentration 12600 mg/L and pH 4.9 (Table 1). While the pH is lower than 6.0, the iron concentration is high. Similar values to the 7:3 layered coal reject to clay ratio are recorded for 90% coal reject layered with 10% clay on 28th November.

Table 1: Leachate iron concentration and pH for contaminated drums

<table>
<thead>
<tr>
<th>Co-Disposal Option</th>
<th>Date Sampled</th>
<th>Leachate pH</th>
<th>Iron Concentration (mg/L)</th>
</tr>
</thead>
<tbody>
<tr>
<td>50% CR, 50% Clay (m)</td>
<td>15 November 1996</td>
<td>6.1</td>
<td>7.6</td>
</tr>
<tr>
<td></td>
<td>28 November 1996</td>
<td>6.0</td>
<td>8.3</td>
</tr>
<tr>
<td>50% CR, 50% Clay (L)</td>
<td>15 November 1996</td>
<td>6.2</td>
<td>10.6</td>
</tr>
<tr>
<td></td>
<td>28 November 1996</td>
<td>6.6</td>
<td>12.7</td>
</tr>
<tr>
<td>70% CR, 30% Clay (L)</td>
<td>28 November 1996</td>
<td>4.9</td>
<td>12.6</td>
</tr>
<tr>
<td>90% CR, 10% Clay (L)</td>
<td>28 November 1996</td>
<td>5.2</td>
<td>12.3</td>
</tr>
</tbody>
</table>
The plots of leachate pH against leachate iron concentration for coal reject co-disposed with sand suggest contamination has not occurred. Iron concentration was high for low pH and low for high pH (Fig. 2).

The subsequent data analysis excludes readings from 50% coal reject mixed with 50% clay and 50% coal reject layered with 50% clay.

No co-disposal option provided a leachate that satisfied all the ANZECC (1992) guidelines for direct discharge into fresh waters (Tables 2 and 3).

![pH and Iron Concentration - Coal Reject and Clay](image1)

Fig 2: pH vs Iron concentration: Coal reject and clay leachate.

![pH and Iron Concentration - Coal Reject and Sand](image2)

Fig 3: pH vs Iron concentration: Coal reject and sand leachate.
RESULTS

The following summarizes the results of the leachate analysis in terms of the ANZECC (1992) guidelines.

In the following analysis the metal concentration in the leachate was grouped by date and co-disposal option to establish whether or not a statistically significant relation exists between the quantity of coal reject and the leachate metals concentration, for the mixed and layered situations. This analysis may indicate a maximum percentage of coal reject that may be mixed or layered with sand or clay before ANZECC guidelines are exceeded. Consequently, optimal reject ratios may be determined.

Table 2: Coal reject and clay co-disposal, leachate metals concentration compared to ANZECC guidelines

<table>
<thead>
<tr>
<th>Metal</th>
<th>Coal Reject and Clay</th>
<th>% Coal Reject</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Layered</td>
<td>Mixed</td>
</tr>
<tr>
<td></td>
<td>10  90  80  70  50  0</td>
<td>10  90  76.3  70  50  0</td>
</tr>
<tr>
<td>Al</td>
<td></td>
<td></td>
</tr>
<tr>
<td>As</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cd</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Co</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cr</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Fe</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hg</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mn</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ni</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Pb</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Sb</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Se</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zn</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

** indicates the leachate was within ANZECC guidelines. In the case of Manganese comparison has been made with the Clean Waters Regulation 1972 - Schedule 2 (P class waters) as no value was given in the ANZECC guidelines. For cobalt comparison has been made with measured values in the Wollangambe River upstream of Clarence Colliery as no value was given in the ANZECC guidelines. Note iron contamination from the drums seems to have effected the measurements - see discussion.

Statistically significant relationships (P < 0.05), exist between the leachate metals concentration and the percentage coal reject mixed with clay for aluminium, cadmium, cobalt, copper, manganese, nickel, lead and zinc (Table 4). Prior to the removal of data from 1:1 coal reject to clay a significant relationship for aluminium did not exist. For leachate cadmium and lead concentrations the relationship between leachate metals concentration and percentage coal reject changed from being statistically significant on 15th November to not showing a significant relationship on 28th November. In both cases the leachate cadmium and lead concentration for 100% clay on 28th November was much higher than those concentrations for samples on 15th November, possibly causing the reduction in the level of statistical significance.
Table 3: Coal reject and sand co-disposal, leachate metals concentration compared to ANZECC guidelines

<table>
<thead>
<tr>
<th>Metal</th>
<th>Coal Reject and Sand</th>
<th>% Coal Reject</th>
<th>Layered</th>
<th>Mixed</th>
<th>Layered</th>
<th>Mixed</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>10 90 80 70 50 20 0</td>
<td>100 90 80 70 50 20 0</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Al</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>As</td>
<td></td>
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<td></td>
<td></td>
</tr>
<tr>
<td>Cd</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Co</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cr</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td></td>
<td></td>
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<td></td>
<td></td>
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<tr>
<td>Fe</td>
<td></td>
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<td></td>
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<tr>
<td>Hg</td>
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<td></td>
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<tr>
<td>Mn</td>
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<td></td>
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<td></td>
<td></td>
</tr>
<tr>
<td>Ni</td>
<td></td>
<td></td>
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<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Pb</td>
<td></td>
<td></td>
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<td></td>
<td></td>
<td></td>
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<tr>
<td>Sb</td>
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<td>Se</td>
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<td></td>
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<tr>
<td>Zn</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

--- indicates the leachate was within ANZECC guidelines. In the case of Manganese comparison has been made with the Clean Waters Regulation 1972 - Schedule 2 (P class waters) as no value was given in the ANZECC guidelines. For cobalt comparison has been made with measured values in the Wollangambe River upstream of Clarence Colliery as no value was given in the ANZECC guidelines.

Table 4: Level of significance where a statistically significant relationship between percentage coal reject and leachate metals concentration exists, coal reject and clay (Data from 1:1 coal reject to clay ratio removed)

<table>
<thead>
<tr>
<th>Metal</th>
<th>Level of Significance</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Coal Reject and Clay</td>
</tr>
<tr>
<td></td>
<td>Mixed 28/11/96</td>
</tr>
<tr>
<td></td>
<td>15/11/96 0.01</td>
</tr>
<tr>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Layered 28/11/96</td>
</tr>
<tr>
<td>Al</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Cd</td>
<td>0.001</td>
</tr>
<tr>
<td>Co</td>
<td>0.001</td>
</tr>
<tr>
<td>Cu</td>
<td>0.003</td>
</tr>
<tr>
<td>Fe</td>
<td>0.014</td>
</tr>
<tr>
<td>Mn</td>
<td>0.035</td>
</tr>
<tr>
<td>Ni</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Pb</td>
<td>0.007</td>
</tr>
<tr>
<td>Se</td>
<td>0.021</td>
</tr>
<tr>
<td>Zn</td>
<td>0.024</td>
</tr>
</tbody>
</table>

The cadmium reading for 100% clay (Fig. 4) on 28th November appeared to be incorrect given that the reading was higher than those for 76%, 70% and 50% coal reject, and almost the same as that for 90% coal reject. Supporting the possibility of an erroneous reading is the low cadmium reading for 100% clay on 15th November, this reading being the lowest of all the cadmium readings. Removal of this value from the cadmium data did not improve the statistical significance of the
relationship. Removing the sample for 100% clay increased the level significance of the relationship for selenium and zinc only, the greatest improvement being for selenium, from a level of significance of 0.041 to 0.01.

No significant relationship was found between leachate iron concentration and percentage coal reject for coal reject and clay mixed. Removal of the readings from 50% coal reject mixed with 50% clay did not improve the statistical significance of the relationship.

The leachate metals concentration increased exponentially as the relative percentage of coal reject mixed increased, except for the element aluminium where the metal concentration decreased exponentially as the percentage of coal reject mixed increased. Figs 4 and 5 show the relationship between leachate metals concentration and percentage coal reject where the level of significance was greater than 5% for cadmium (Fig. 4) and manganese (Fig. 5).

Statistically significant relationships (P • 0.05), exist between the leachate metals concentration and the percentage coal reject layered with clay for cadmium, cobalt, iron, manganese, nickel and zinc (Table 4). Removal of the 100% clay sample data for the 28th November increased the level of significance for cadmium (0.299 to <0.001) and zinc only. Note that without removing the 100% clay reading a significant relationship between coal reject and metal concentration for zinc existed.

The concentration of metals in the leachate increased exponentially as the relative percentage of coal reject increased for the coal reject/tailings layered configuration.

\[ Cd = 0.0097e^{0.0584Rc} \]
\[ R^2 = 0.99 \]
\[ P < 0.001 \]

Fig 4: Cadmium concentration: Coal reject and clay leachate (Data from 1:1 coal reject and clay removed)

The leachate aluminium concentrations for coal reject layered with sand in 7:3 and 8:2 ratios are much higher than the leachate aluminium levels for other coal reject and sand layered ratios. There is probably not a drum contamination problem as there is no statistically significant relationship between the quantity of coal reject and the leachate aluminium concentration.

Statistically significant relationships (P • 0.05), exist between the leachate metals concentration and the percentage coal reject layered with sand for cadmium, cobalt, copper, iron, manganese, nickel, lead, selenium and zinc (Table 5).
Leachate from 90% coal reject mixed with 10% sand on the 15th November had much higher concentration of cadmium, cobalt and zinc than the other coal reject and sand mixed ratios. Removal of the 9:1 sample from all data sets on that date showed statistically significant relationships \((P < 0.05)\), exist between leachate metals concentration and the percentage coal reject mixed with sand for cadmium, cobalt, copper, manganese, iron, nickel and zinc (Table 5). Statistically significant relationships, \((P < 0.05)\), exist between leachate metals concentration and the percentage coal reject mixed with sand for cadmium, cobalt, copper, lead, selenium and zinc for samples collected on 28th November (Table 5). The leachate metals concentration increased as the percentage of coal reject mixed or layered increased for all elements.

Table 5: Level of significance where a statistically significant relationship between percentage coal reject and leachate metals concentration exists, coal reject and sand (Data from 9:1 coal reject to sand mixed ratio 15th November removed)

<table>
<thead>
<tr>
<th>Metal</th>
<th>Level of Significance</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Coal Reject and Sand</td>
</tr>
<tr>
<td></td>
<td>Mixed</td>
</tr>
<tr>
<td></td>
<td>15/11/96 28/11/96</td>
</tr>
<tr>
<td>AI</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Cd</td>
<td>0.001</td>
</tr>
<tr>
<td>Co</td>
<td>0.027</td>
</tr>
<tr>
<td>Cu</td>
<td>0.034</td>
</tr>
<tr>
<td>Fe</td>
<td>&lt;0.001</td>
</tr>
<tr>
<td>Mn</td>
<td>0.015</td>
</tr>
<tr>
<td>Pb</td>
<td>0.042</td>
</tr>
<tr>
<td>Se</td>
<td>0.021</td>
</tr>
<tr>
<td>Zn</td>
<td>0.009</td>
</tr>
</tbody>
</table>

Figs 6 and 7 show the relationship between leachate metals concentration and percentage coal reject where the level of significance was greater than 5% for cadmium (Fig. 6) and manganese (Fig. 7).
Statistically significant relationships between the leachate metal concentration and percentage coal reject where the trends changed between the two sampling events are shown in Table 6.

Table 6: Change in leachate metals concentration over time for coal reject and sand tailings co-disposal

<table>
<thead>
<tr>
<th>Metal</th>
<th>Change in leachate metals concentration over time</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Coal Reject and Clay</td>
</tr>
<tr>
<td></td>
<td>Layered</td>
</tr>
<tr>
<td>Cd</td>
<td>Decreased</td>
</tr>
<tr>
<td>Co</td>
<td>Increased</td>
</tr>
<tr>
<td>Cu</td>
<td>Decreased</td>
</tr>
<tr>
<td>Mn</td>
<td>Increased</td>
</tr>
<tr>
<td>Ni</td>
<td>Increased</td>
</tr>
<tr>
<td>Pb</td>
<td>Decreased above 90% coal reject</td>
</tr>
<tr>
<td>Se</td>
<td>Decreased</td>
</tr>
<tr>
<td>Zn</td>
<td>Increased above 85% coal reject</td>
</tr>
</tbody>
</table>

Cadmium Concentration - Coal Reject and Sand

\[
Cd = 0.1424e^{0.0399R_c}
\]

\[
R^2 = 0.67
\]

\[
P < 0.001
\]

Fig 6: Cadmium concentration: Coal reject and sand leachate (Data from 9:1 coal reject to sand mixed ratio 15° November removed)
Leachate metal load calculations for the 30 days of monitoring show coal reject and clay layered options produced the lowest total metal loads of the four co-disposal options (Table 7).

A consequence of very little of the water introduced to the coal reject and clay layered options passing through the material was the total leachate metal load was at most one tenth of the load from the three other co-disposal modes where all introduced water had passed through. Leachate from coal reject and clay layered had zinc, manganese, iron, copper, cobalt and nickel concentrations above those recommended by ANZECC Guidelines (Table 2). Compared to the 30 day total load for leachate from 100% coal reject, the total metal loads in leachate from coal reject and clay layered for these metals was reduced by two orders of magnitude except for the 7:3 coal reject to clay ratio where the load was reduced by one order of magnitude. 30 day loads for leachate from 100% sand for manganese, copper and cobalt was not found as the relationship between leachate conductivity and metal concentration for co-disposal with sand gave negative values when metal load was calculated. Consequently comparison between metal loads for leachate from 100% sand and leachate from coal reject layered with clay cannot be made for these metals. Compared to the 30 day total load for leachate from 100% clay, the total metal loads in leachate from coal reject and clay layered for zinc, manganese, iron and nickel increased by two orders of magnitude except for the 7:3 coal reject to clay ratio where the load increased by three orders of magnitude. The copper and cobalt loads for leachate from coal reject and clay layered was two orders of magnitude greater than for leachate from 100% clay, again except for the 7:3 coal reject to clay ratio where the increase was three orders of magnitude.
Table 7: Total metal loads (mg) after 30 days

<table>
<thead>
<tr>
<th>Co-disposal option</th>
<th>Al</th>
<th>As</th>
<th>Cd</th>
<th>Co</th>
<th>Cu</th>
<th>Fe</th>
<th>Mn</th>
<th>Ni</th>
<th>Pb</th>
<th>Se</th>
<th>Zn</th>
</tr>
</thead>
<tbody>
<tr>
<td>CR &amp; Cl (L) 9:1</td>
<td>6.9</td>
<td>0.025</td>
<td>3.5</td>
<td>0.26</td>
<td>87</td>
<td>37</td>
<td>8.5</td>
<td>-</td>
<td>0.056</td>
<td>8.5</td>
<td></td>
</tr>
<tr>
<td>8:2</td>
<td>5.2</td>
<td>0.016</td>
<td>0.9</td>
<td>0.085</td>
<td>56</td>
<td>24</td>
<td>2.2</td>
<td>-</td>
<td>0.033</td>
<td>3.3</td>
<td></td>
</tr>
<tr>
<td>7:3</td>
<td>25</td>
<td>0.1</td>
<td>110</td>
<td>5.1</td>
<td>460</td>
<td>140</td>
<td>300</td>
<td>-</td>
<td>0.36</td>
<td>110</td>
<td></td>
</tr>
<tr>
<td>5:5</td>
<td>6.4</td>
<td>0.019</td>
<td>0.83</td>
<td>0.085</td>
<td>68</td>
<td>28</td>
<td>2</td>
<td>-</td>
<td>0.039</td>
<td>3.5</td>
<td></td>
</tr>
<tr>
<td>2:8</td>
<td>10</td>
<td>0.013</td>
<td>0.59</td>
<td>0.076</td>
<td>120</td>
<td>24</td>
<td>1.4</td>
<td>-</td>
<td>0.063</td>
<td>3.8</td>
<td></td>
</tr>
<tr>
<td>CR &amp; Cl (M) 9:1</td>
<td>6.9</td>
<td>0.025</td>
<td>3.5</td>
<td>0.26</td>
<td>87</td>
<td>37</td>
<td>8.5</td>
<td>-</td>
<td>0.056</td>
<td>8.5</td>
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Note: Where no load values are presented there was not a statistically significant relationship between leachate conductivity and metal concentration for leachate from the two sampling dates. 30 day loads for leachate from 100% sand for manganese, copper and cobalt was not calculated as the relationship between leachate conductivity and metal concentration for co-disposal with sand gave negative values when metal load was estimated.
DISCUSSION

No co-disposal option satisfied all the ANZECC guidelines for discharge into fresh water. Except for the elements iron and manganese, the leachate from coal reject and sand co-disposal options had higher metals concentration than leachate from coal reject and clay co-disposal. Flow monitoring of coal reject and clay layered options showed that very little of the water introduced to the material passed through to the coal reject as most water became overflow. While leachate from the layered coal reject and clay has some elevated metals concentration the low discharge volume allows for passive downstream supplementary treatment by such means as wetland filters.

Results of this study suggest that co-disposal of coarse coal washery reject from Clarence Colliery with reject from Kable’s Transport sand mine is a suitable option for reducing the effects of Acid Mine Drainage. It is possible to reduce the leachate metals concentration and minimise leachate volume by layering coal reject with clay. Consideration of leachate pH, conductivity, volume and metals concentration, as well as the production rate of reject material suggest that the optimal ratio of coal reject to clay layered would be 9:1. Field trials need to be carried out to assess the long term viability of such a system.

ACKNOWLEDGMENTS

The authors acknowledge Kable’s Transport Pty Ltd., Lithgow, for the supply of sand and clay reject used in the experiments.

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Sampling Surficial Sediments of a River Receiving Minewater Discharges

D J Cohen¹, C V McQuade², S J Riley¹, and S Adeloju¹

ABSTRACT

Metal contamination of sediments can be an issue for minesites discharging water. Water sampling of receiving waters is frequently undertaken. Sampling and analysis of sediment is less common. The variability of receiving environments makes the formulation of a generic sediment sampling strategy virtually impossible. Each situation needs to be carefully assessed before sampling is undertaken. This paper outlines the approach taken for one such survey.

The main factors that need to be considered in a survey of surficial sediments are sample representativeness and sample variability. A simple data set from one site within a regional survey is used to illustrate the importance of these factors. The metal content of sediment is shown to vary significantly over small spatial scales.

The variability associated with the spatial distribution of samples within a site is shown to be the most significant source of variability in the study. This variability could bias the results of a study if not planned for in the sampling methodology. Taking multiple samples from within each site (sub-site samples) and combining the data gives a more representative indicator of overall conditions than a single sample. The optimum number of sub-site samples to be taken from each sampling site was found to be 15.

It is shown that a composite of sub-site samples can give a good indication of the average site metal content, while considerably reducing the sample preparation and analysis effort.

INTRODUCTION

Clarence Colliery is located about 100 km west of Sydney, near Lithgow. Approximately 14ML of water per day, including underground dewaterings and site drainage, is treated to Environmental Protection Agency (EPA) licence requirements and released into the Wollangambe River. The licensing requirements on the water released are comparably stringent to drinking water standards. A study of the metal concentrations in the sediments of the Wollangambe was initiated as part of the development of Clarence Colliery's environmental best practice program.

Six sites were sampled for the Wollangambe River regional sediment study, along a 25 km stretch of the river. Multiple samples were collected from each of the sites and analysed to determine the concentration of key metals in the surface layer of the sediment. Previous data was used to select the metals to be included in the study. The metals selected were cobalt, iron, manganese, nickel and zinc. The mass of the metals per unit area of the riverbed was calculated for each site. A regional picture of the Wollangambe River was developed from this data.

Multiple samples were collected from spatial locations that varied over a small distance (of the order of metres) at each of the sampling sites. This enabled the variability associated with the spatial location of the sediment metal concentration to be assessed for each of the sites. This variability was then compared to the variability associated with the preparation and analysis of the samples to assess its significance. The optimum number of sub-site samples required to achieve a figure representative enough of the true sediment metal concentration was evaluated.

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STUDY METHODOLOGY

Sampling

The Wollangambe River system is a highly variable environment. The river rapidly changes from sections of fast flowing riffle to slow moving pools. A wide variety of depositional environments are encountered, even over a small area of the order of just a few square metres. Taking a sample from just one of these environments would not provide data indicative of the overall conditions for that site on the river. For the data to have meaning in the development of a regional picture of metal contents many samples must be taken from each site and an average metal content for the site calculated. This reduces the risk of small-scale spatial variability of the sediment metal content biasing the data collected.

A research program was conducted on the sampling of sediment in rivers. Previous work conducted indicates that sediment can exhibit considerable variation at all spatial scales (Morrisey et al, 1994). Such variation lead Herr and Gray (1997) to conclude that subsite sampling is essential to obtain reasonably accurate estimates of the metal concentrations of a specific site. For the Wollangambe River regional sediment survey, it was decided that a nominal number of 25 sub-site samples should be collected from each site. Samples were taken from the nodes of a 2m x 2m grid placed over the river bed area. Fig. 1 shows a graphical representation of the sampling grid used at the first site sampled in the study (site W1). The sample data set upon which this paper is based comes from this site.

Samples were taken systematically from every node of the grid. Systematic sampling provides unbiased results, providing the position of the starting node is randomly selected. A random sampling pattern was considered, however the use of such methods is discouraged (NSWEPA, 1995). Random sampling is statistically unbiased, but the chance of sampling points clustering together makes the method unsuitable for providing an overall picture of the spatial distribution of metals within the sediments of a site.

The surface layer of sediment is subject to the most recent accumulation of metal ions. It is in direct interaction with the water column and river biota (Herr and Gray, 1997). The top 25 mm of sediment was sampled from the river bed and all available material was collected when the depth of the sediment layer was under 25 mm.

Sediment samples were collected using a PVC plastic cylinder of 100mm diameter and 25 mm depth. One end of the cylinder was open to allow it to be driven into the sediment. A hole was drilled in the enclosed end of the cylinder to enable water to escape as the cylinder was driven into the sediment. After the cylinder was driven into the sediment, a stiff piece of plastic was inserted under the open end of the cylinder. The cylinder containing the undisturbed sediment sample was then withdrawn and the sample placed in labelled zip-lock plastic bags. The bag was then placed in another bag to ensure security of the sample during transport.

For the purposes of the Wollangambe regional sampling program, biofilms were considered an integral part of the river's sedimentary system. Layers of biofilm were thus sampled for inclusion in the study when encountered. Biofilms are defined as an assemblage of immobilised microbial cells (mainly bacteria) bound into a matrix of excreted extracellular polysaccharides (Ragusa, 1996). Biofilms generally form at the solid/water interface due to the fact that nutrients often concentrate where surface charges exist on solid surfaces. The tangled mats of cells that form can catch and hold sediments as well as adsorb and fix dissolved metals. Biofilms were collected and subsequently analysed using the same techniques employed for other collected sedimentary materials.

Samples of biofilm were collected by placing an open ended PVC cylinder over the surface to be sampled and loosening the biofilm within the cylinder with a plastic scraper. The biofilm was then collected with a plastic suction pump. The cylinder used was of the same area dimensions as the one used to collect sediment samples.

Many sections of the river were flowing rapidly, which made collection of samples difficult. The sampling equipment used was developed primarily in response to the need to collect samples in a way that minimised the loss of the sample in the flowing water. The samples were visually classified, based on grain size, at the time of collection. The categories for the sediment material collected were sand, fine sand, mud and biofilm.
Analysis

The mass of sediment for the area sampled at each node was determined in the lab by drying (at 105°C) and weighing. The samples were then homogenised and the sample size reduced using the coning and quartering technique. US EPA Method 3050B (US EPA, 1980) was employed for the digestion of the sediment samples prior to analysis. This digestion technique is a strong acid digest. The metals that are in insoluble particulate form or adsorbed to the surface of the sediment matrix are dissolved into the solution to be analysed. The actual matrix of the sediment material is not broken down.

The digested solutions were then analysed utilising flame atomic absorption spectrometry (APHA, 1995). The accuracy of the analytical result is reflected by how well it agrees with the true quantity of constituent. This was assessed with Standard Reference Materials (SRM’s). The SRM used in the Wollangambe River regional sediment study was AGAL 10, a lake sediment sample provided by the Australian Government Analytical Laboratories.

Data Assessment

Assessment of the variability associated with the preparation and analysis of the samples

The precision of a result is reflected by its reproducibility. Replication of analysis was used to give a statistical estimation of the precision of the preparation and analysis techniques employed. Variability data from measurements of replicates from different samples may be combined (or pooled) to give a better estimate of variance, while minimising the number of replicate analysis that are carried out on each sample. It should be noted that this method of pooling is only applicable when the estimates of variability are from the same population. This was the case, as we were estimating the variance of a set preparation method and a single analysis instrument. Variance is often expressed in terms of standard deviation (s). The general equation used to pool estimates of variability from replicates of different samples is shown below:

\[
 s_e^2 = \frac{\left(s_r^2 \times df_1\right) + \left(s_1^2 \times df_2\right) + \left(s_2^2 \times df_3\right) + \ldots \left(s_j^2 \times df_j\right)}{df_1 + df_2 + df_3 + \ldots + df_j}
\]

where

- \(s_e\) = the standard deviation of the preparation and analysis method
- \(s_r\) = the standard deviation of the set of replicates
- \(df\) = the degrees of freedom in the replicate set
- \(j\) = the number of subsite samples

In the analysis of sub-site samples from the trial site, it was decided that two replicate samples would be made from each sub-site sample. These duplicates were analysed and the standard deviation estimated from the two values obtained. This represents an estimation of variance made with one degree of freedom (\(df = 1\)), as two samples were used to calculate the standard deviation.

Assessment of Small Scale Variability

The variability associated with the preparation and analysis of the samples (represented by the standard deviation \(s_e\)) was used to estimate the variability associated with the spatial distribution of the sub-site samples (represented by the standard deviation \(s_r\)). Mean values were calculated for each sub-site sample based on the duplicate analyses. The standard deviation of the sub-site samples was calculated using the mean values of the metal concentrations for each set of sub-site replicates. Fig. 2 represents the nested structure of the sampling and analysis for the site.

Values for level 2 were calculated from the mean of level 3 values. The pooled variance from level 3 was calculated to give an estimate of the variance associated with analysis of the sub-site samples (\(s_e\)). The value for level 1 was calculated from the mean of the level 2 values. The variance from level 2 was calculated and used to estimate the variance associated with the spatial distribution of the sub-site samples (\(s_r\)).
Fig. 1 - Sampling grid pattern used at the first site sampled in the study
The variability associated with the spatial distribution of the sub-site samples within the site was calculated as follows:

\[ s_s = \sqrt{s_{\text{sample}}^2 \cdot \frac{s_a^2}{2}} \]

where

- \( s_s \) = the standard deviation of the spatial distribution of the samples
- \( s_a \) = the standard deviation of the preparation and analysis method
- \( s_{\text{sample}} \) = the standard deviation of the sample values

The relative sizes of the sources of variability for the overall site and within each sediment category for iron are shown for iron in Fig. 3. The variability associated with the preparation and analysis of the samples is generally much smaller than the variability due to the spatial distribution of the sub-site samples. The exception was the fine sand samples where the variability associated with the preparation and analysis was larger than the spatial distribution variability. This was attributed to the fine sand samples from which the replicates were taken not being very homogeneous.

Fig. 2 - Nested design for analysis of variance of trial sampling sites

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Fig. 3- Relative sizes of the sources of variability for iron sample data set

The variability associated with the preparation and analysis of the samples is also much smaller than the variability due to the spatial distribution of the sub-site samples for cobalt (Fig. 4) and manganese (Fig. 5). The variability associated with the preparation and analysis of the samples is larger than the spatial distribution variability for nickel (Fig. 6) and zinc (Fig. 7). This indicates that the nickel and zinc are more uniformly distributed across samples taken from different locations at the site. This trend is also visible in the more uniform contents of nickel and cobalt in the variety of sediment types encountered, which is discussed in the next section.

Examination of the Variance associated with the Spatial Distribution of Sediments at a Site

The variability of sample composition at a site is created by the flow patterns that occur in the river. The water velocity controls the type of depositional environment that occurs at a point. As water moves more slowly a higher percentage of fine-grained sedimentary material is deposited. This gives rise to differences in the composition of sediment samples. Biofilms cannot build up in areas where sediment is constantly being added or removed. The biofilm is either covered over by freshly deposited sedimentary material, or removed with the departing sediment. Layers of biofilm are commonly encountered in rapidly moving environments, such as sections of riffle over rock ledges. The microorganisms that form the biofilm layer actively attach themselves to the solid surface over which the water is flowing. The layer can then grow without being removed under normal flow conditions.

Many subsystems of sedimentary environments may be encountered at a site due to the processes describe above. These types of material have different average metal concentrations as well as differing degrees of variability. The mean metal
Fig. 4 - Sources of Variance for Cobalt

Fig. 5 - Sources of Variance for Manganese

Fig. 6 - Sources of Variance for Nickel

Fig. 7 - Sources of Variance for Zinc
contents of the site were calculated using the 25 values averaged from the duplicate analysis of each sub-site sample. The mean metal contents for the overall site as well as for each of the sedimentary material categories are shown in Fig. 8 to Fig. 12. There is a trend of increasing metal content as the grain size of the sediment gets smaller (i.e., sand → fine grained sand → mud) for both iron and manganese. This is because the surface area of the finer grained sediment is higher, thus providing more sites for the metal ions to adsorb to. Biofilms also show high iron and manganese contents due to the processes of absorption and fixation of high concentrations of these particular metal ions into the biofilm layer. Nickel and Zinc both tend to show highest association with fine grained sand material, and tend to have a more even range of contents for the different types of sedimentary material.

Fig. 8 – The mean Co content for the overall site and each of the sedimentary material categories

Fig. 9 – The mean Fe content for the overall site and each of the sedimentary material categories
The different types of sedimentary material encountered at a site are the main cause of variability. Variability exists between materials of the same type, but it is not generally as high as the overall site variability. This trend can be seen by comparing the spatial distribution variance of each of the categories of sediment to the spatial distribution variance of the overall site.
The mean Zn content for the overall site and each of the sedimentary material categories

Overall site for iron in Fig. 3. Mud was the exception, showing high variability in its composition. A study of the effects of the number of sub-site samples taken on the variability of the results obtained was conducted. This involved graphing the average variability generated by 1000 random combinations of two sub-site samples, 1000 random combinations of three sub-site samples, and so on up until all 25 samples were considered. This analysis is biased by the fact that it does not consider every possible combination of sub-site samples that could be generated. The process, however, provides a rough estimate of the number of samples that should be collected in a similar study. A graph of the results of this analysis for iron is shown in Fig. 13. The optimal number of sub-site samples was found to be approximately 15.

Taking fewer samples than this generates higher overall variability. Taking more than 15 samples does not enhance the results, as the variability is not significantly lowered. This result also applied to all of the other metals considered in the study.
**Composite Sampling Technique Trial**

A composite sampling strategy was also explored. The value of taking a composite sample to reduce the number and volume of samples that needed to be transported back from each site and analysed was evaluated. Two composite samples were prepared by adding 10% (according to total wet sample mass) of each of the sub-site samples collected. This method ensured that the composite samples contained the same relative proportions (by wet mass) of material as was initially collected from each equal area that was sampled.

The composite samples were prepared and analysed identically to the individual samples. The masses of the specified metals in the composites were compared to the mean of the masses of the metals in the individual sub-site samples. The composite samples both yielded results within one standard deviation of the average content of the metals calculated from the analysis of the 25 sub-site samples. This indicates that the method successfully represented the average metal content of the samples that were used to make up the composite.

The composite could be prepared in the field using a technique where equal areas of sediment from each node are combined. This composite sample would then be much easier to transport from the sampling site and would require much less analysis effort than the equivalent number of individual samples.

It should be remembered that results obtained from composites have higher levels of variability than indicated by repeated analysis of the sample. This is because the variance of the spatial distribution of the samples still exists, although it cannot be determined. The method is not suitable for determining if there are zones within a study area that exceed specified threshold values. The method is, however, suited to the determination of the average metal content within the sediment of a river at a certain location, as in the case of the regional sediment survey being undertaken here.

**STUDY CONCLUSIONS**

Considerable variability in metal content exists in the sediments within the immediate area of the sampling site. The main influence on the sediment metal content at any given location is the variety of sediment types and grain size that may be encountered. Variability exists at a small spatial scale and can thus influence the average metal concentration results for the site if not allowed for in the sampling methodology. To ensure that small scale spatial variability is taken into account it is recommended that about 15 sub-site samples be taken from a set sampling pattern at a site.

Positioning sampling points on the nodes of an evenly spaced grid that is randomly positioned at the sampling site reduces sampling bias, providing representative sampling of the site. The variability associated with the preparation and analysis of the samples was generally considerably less than the variation of samples due to their spatial location. The size of the spatial variability could influence the choice of analysis method employed for such a study. Highly precise techniques may not be efficient when the accuracy of the data is degraded by the site variability. Composite samples can provide the same level of metal content accuracy for a considerably reduced analysis effort.

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* except for Mn content in one composite, which fell just outside of the one standard deviation range of the average metal content


Diesel Vehicle Research at BHP Collieries

S Pratt¹, A Grainger¹, L Jones¹, J Todd¹, R Brennan¹, B Horton¹, L Eager², A Rogers³ and B Davies⁴

ABSTRACT

Research into the control of diesel particulates (DP) has been conducted by BHP Coal for more than 7 years. Personal monitoring of employee exposures (n = 480 full shift samples) conducted at nine underground coal mines has indicated that the exposure of the workforce ranges from less than 0.1 to 2.2 mg/m³ of DP dependent on job type and mining operation. Approximately 50% of the mass of DP captured is elemental carbon (EC) which is the species currently being considered by some international regulatory authorities as the exposure standard. Five technologies for controlling DP were investigated in a combination of studies conducted in an above ground simulated tunnel, in a special controlled section of underground mine roadway and validated by application in standard coal mining operations at Tower Colliery. Tests conducted under controlled conditions indicate that dependent on the type of fuel in use, the introduction of low sulphur fuels can reduce DP levels in return airways by up to 50% and in actual mining situations a reduction of 20% can be achieved in exposure of the workforce. In addition, subjective responses from the workforce indicate that exhaust emissions from low sulphur fuels provide lower irritation and a more pleasant aroma. The use of water filled scrubber tanks reduces the level of DP emissions by 25%. Chemical decoking of engines resulted in a reduction of 20% in DP in return airways. A commercially available non-flammable disposable dry exhaust filter constructed from synthetic organic fibres with an operational lifetime in excess of 20 hours was found to reduce DP exhaust emissions by 80%. Investigations have indicated that the use of increased ventilation to control DP levels particularly in multiple vehicle situations does not follow a simple dilution factor and in some instances compliance with current regulatory requirements may not produce the required reduced exposure levels. The results from single component control strategies provide considerable reduction in exposure to DP, however the most efficient and cost effective control methodology is the use of a combination of individual systems modelled to operations conducted at each mine.

INTRODUCTION

For more than forty years diesel powered equipment has been used extensively in Australian underground coal mines and for much of that period colliery employees have expressed concern over direct health effects such as unpleasant aroma, eye and respiratory tract irritation, and to potential adverse health effects such as lung cancer arising from exposure to diesel exhaust emissions (NIOSH, 1988). In May 1990 BHP Collieries commenced a major research project with the aim of investigating and reducing exposure to diesel exhaust emissions. The interdisciplinary team consisted of mine management, mining engineers, diesel mechanics, occupational hygienists and workforce representatives.

The NSW Coal Mines Regulations require that each piece of diesel machinery is subject on a monthly basis to underground checking by the operator for the compliance of raw engine exhaust gas concentrations (using detector tubes) for CO (maximum 1500 ppm) and oxides of nitrogen ( maximum 750 ppm ) and in addition every six months a comprehensive analysis is required to be carried out by an accredited laboratory that is approved by the Mines Department (NSW, 1984). A minimum fresh air ventilation of 0.06 M³/second/kilowatt of power is required to provide adequate dilution of exhaust gases for each machine. Although control of gaseous emissions indirectly controls soot emissions there is no engine emission standard for DP and no occupational exposure standard for DP in Australia.

Initial investigations conducted by the team indicate that raw diesel emissions consist of a complex mixture of toxic and noxious gases, vapours and particulate matter (soot). The soot or diesel particulate (DP) is almost entirely respirable, with about 90% by mass having an equivalent aerodynamic diameter of less than 1.0 micrometer which are readily deposited in.

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the alveolar region of the lung (Rogers, Davies and Conaty, 1993). The surface characteristics and the graphitic nature of
the carbon core allows ready absorbence of organics such as polycyclic aromatic hydrocarbons that have been generated in
the combustion process (UN, 1994)

Given the increasing dependence of Australian underground coal mining operations on diesel equipment, the research
team set out to investigate through a number of sub-projects the following technical aspects:

1. Suitable monitoring methods to capture, separate and quantify diesel particulates;

2. The relative influence on DP emissions and driver exposure to known major variables such as work routines,
   engine type and condition, ventilation rates, fuel quality and various scrubber control technologies (both above
ground simulated testing and underground controlled section mine roadway testing)

3. Application of the findings to actual underground mining situations at Tower Colliery so as to determine the extent
   of influence in a mixed factor environment.

4. To determine the usage of diesel equipment, extent of DP exposure of the workforce and the extent of variables
   such as fuel quality at both large and small underground coal mines in all mining districts within NSW.

5. Monitoring of overseas developments in measurement, advances in control technology and legislative trends for
diesel particulates.

TEST METHODS

Diesel particulate capture and measurement

Diesel particulate (DP) by definition is the mass of sub-micron aerosol particulates found in diesel exhaust. The sampling
device used in our project was developed by the University of Minnesota /US Bureau of Mines to measure diluted diesel
exhaust as found in general mine atmospheres via a combination of a cyclone and impaction plate to separate the
submicron fraction of aerosols which is subsequently gravimetrically determined (Cantrell and Rubow, 1992).
Transmission electron microscopy studies indicated that for mine atmosphere samples, more than 85% of the DP was
captured using this technique, with the remainder being excluded since it is attached to larger dust particles. Quantitative
Analytical scanning electron microscopy indicated that the amount of positive interference from submicron non diesel
particulates such as coal dust was less than 10% provided that the respirable dust levels are kept below the statutory limit
of 3.0 mg/M^3 (Rogers, Davies and Conaty, 1993) As a result of overseas changes in methodology, supplementary studies
were introduced later to measured the elemental carbon (EC) content of the DP (Rogers and Whelan, 1996).

Relative effect of variables and the application to controlling exposures

Above ground simulated tunnel testing

Initial test work was carried out in a pit top test tunnel (45 m long, 5 m wide and 2.5 m high) designed to simulate a
section of mine roadway (Pratt et al, 1993). An auxiliary fan was fitted at one end to provide ventilation inside the tunnel
in accordance with the NSW Coal Mines Regulation Act. The test vehicle chosen was a Domino Minesmobile fitted with a
Perkins 242 direct injection engine. A stringent driving schedule incorporating loaded and unloaded cycles was used that
represented as much as possible the normal driving cycle of the vehicle if it were being used underground (total time for
the cycle was five hours). Personal DP samples were collected on the driver during the various test regimes. The raw
exhaust gases (carbon monoxide, oxides of nitrogen and carbon dioxide) were continually sampled using a probe inserted
into the exhaust manifold with flexible tubing connected to direct - reading instruments with data stored directly onto a
portable computer. This above ground system allowed the use of test equipment that did not meet the underground
intrinsic safety requirements of the Coal Mines Regulations.
Controlled section underground mine roadway testing

After the satisfactory completion of the above ground tests, a suitable section of mine roadway approximately 110 m long was identified and pneumatically operated steel doors were installed at one end to provide variable control of ventilation to deliver the exact legislative ventilation requirements for various types of machinery. To reduce undue influences with respect to extraneous airborne dust, a large section of the rib was meshed and sprayed with concrete while the floor was graded and filled with road ballast. A Noyes Multi Purpose Vehicle (MPV) was selected as the most appropriate vehicle for testing because of the number of these vehicles in the colliery fleet and previous research which had demonstrated that these vehicles produced diesel aerosol particulates at a level that minimised the analytical errors associated with the selected sampling device. A driving schedule including various load cycles was developed that reflected as much as possible the normal activities of this type of vehicle in transporting supplies around the colliery over a normal shift. The vehicle was driven backwards and forwards along the test section of roadway for a number of shifts. Diesel aerosol particulate samplers were placed on the MPV operator; the machine itself adjacent to the driver's cabin; inbye of the MPV between the vehicle stopping barrier and the ventilation doors; and outbye of the MPV near the entrance to the test station roadway.

Mining operations at tower

At the completion of various stages of the above ground and underground test tunnel studies, depending on the results obtained under controlled conditions, individual control technologies were introduced to actual mining operations at Tower Colliery and their efficacy tested. Tower is typical of contemporary operations in underground coal mining in NSW. With a workforce of 409 operating a long wall over three production shifts, it achieves an annual production of over 2 million tonnes of high grade coking coal which is used both domestically and for export. Extensive methane drainage is used to supplement ventilation control of gaseous coal seam emissions. Thirty three trackless rubber tyred diesel powered units are used throughout the mine mainly for transport of manpower and materials (70 kilowatt) with heavy duty diesel powered equipment (120 kilowatt) being used for longwall moves.

Results of above-ground, simulated tunnel and real mine operation testing

Fuel quality

The above ground tunnel testing of 3 commercially available diesel fuels that are commonly used in the mining industry, one commercial low emission fuel and one experimental low emission fuel indicated that both aromatic hydrocarbon and sulphur content affected DP exposures with increased sulphur content in particular indicating a linear increase in driver exposure to both DP, carbon monoxide and oxides of nitrogen. Sulphur has a major influence resulting in the creation of fine particulate sulphates during combustion which catalyses the formation and agglomeration of increased levels of carbon nuclei.

Operational field trials were conducted by introducing the low emission test fuel into a small underground lead/zinc mine that traditionally used high sulphur imported fuel (Pratt et al, 1993). A 40 to 50% reduction in DP measurements was found in the general work areas and return airways in this mine which was similar to the results predicted in the above ground tunnel tests. Trials conducted at Tower Colliery where commercial diesel fuel that complied with the coal mines regulations was substituted with the experimental fuel, resulted in reductions in DP exposure to the workforce of around 10-15%. In both mines the workforce commented favourably on the use of the low emission fuels, reporting an immediate reduction in irritation and an improvement in the aroma of the emissions. The low emission test fuel has now been incorporated into all BHP underground coal mines in NSW. A fuel specification designed to meet low emission standards also needs to take into consideration other parameters in addition to sulphur including cetane number (which effects cold start, emissions and diesel knock), density (which effects power and hence emission levels), cloud point (which creates problems in cold climates) flash point (which indicates flammability) and distillation range (which indicates higher ends that result in the creation of additional DP) (NSW Mineral Council, 1996).

Scrubber tank cleanliness

A series of tunnel runs were conducted separately on two machines with the raw exhaust scrubber tank was trialed dry, wet and after extensive cleaning using chemical agents and a high pressure water jet. The NSW legislative requirements are
that this trap be filled with water so as to act as a flame trap but it was also found that water in the scrubber tank acts as an effective impaction barrier for DP resulted in a reduction of 20 -30% to workforce exposures. The presence of water is the major influence in achieving the levels of reductions observed with vehicles fitted with scrubber tanks, however it was found that the additional degree of cleaning instigated in the tests did not increase efficiency of capture above that of normal routine flushing procedures.

Engine chemical decoking

Short - term tests previously conducted at Tower Colliery has shown a significant reduction in the generation of diesel particulates after chemical decoking using commercially available systems. Further tests were conducted to determine the long - term effectiveness of procedures for the chemical decoking of engines and to evaluate the effects on component wear. A Noyes MPV was tested under controlled conditions for a number of days, then removed to the underground workshop and chemically decoked. After decoking, the vehicle was returned to the test station and retested over a number of successive days until the diesel particulates indicated a downward trend. The vehicle was returned to the vehicle fleet to resume normal duties from which it was extracted at regular intervals over a nine month period for retesting. A significant downward trend of diesel particulate levels was obtained soon after treatment and maintained over the nine month period of sampling. Apart from routine maintenance, no other work has been performed on the engine over the sampling period and no adverse mechanical effects were noted up to the present.

The decoking process releases considerable quantities of soot through the vehicle exhaust and water filled scrubber tank which continues for a number of hours. This release of built-up carbon requires the careful management of exhaust emissions to ensure workers are not inadvertently exposed to a range of noxious products.

Other machines have been similarly treated and tested and decreased DP emissions were also found although the amount of reduction was found to be dependent on the condition of the engine prior to the decoking.

Disposable exhaust filter

Considerable work has been undertaken in the USA to develop a disposable exhaust filter system (Ambs and Hillman, 1992). These filter types have a proven record in the USA of reducing DP exposures in underground vehicles, but unfortunately a number of incidents have been recorded where hot exhaust has carbonised and even ignited the paper filters making them unacceptable for use in Australian coal mines. Our research developed a filter constructed from polypropylene filter material which has enhanced flame resistant characteristics, good filtration characteristics, is not adversely effected by water mist and is commercially available obtainable at reasonable cost (Pratt et al, 1995).

A P. J. Berriman Pty Ltd Power Tram fitted with and without filters to the scrubber tank exhaust was tested under full power load in the underground test tunnel for durations up to four shifts. Back pressure was monitored to ensure that engine performance and exhaust emission levels were not adversely effected. Significant reductions in diesel aerosol particulates were recorded using the filter for periods in excess of three consecutive shifts even though backpressure increased to 20 kPa no effect was detected on engine performance. Without the filter, visibility in the heading was poor with significant levels of water vapour and soot present while with the filter, visibility was improved significantly. A number of other diesel machines types at Tower have been fitted with the filter units and this has been successful in reducing DP exposures in the mine.

The exhaust filter devices are now commercially available although not all equipment have filter kits which are approved. This system however provides the best short - term means of controlling diesel particulates from low - use heavy haulage vehicles in the underground coal mining industry. It is necessary that appropriate safety systems be installed with the unit to ensure that the filters are never exposed to temperatures above their design characteristics and that they are changed out on a regular basis (e.g. after each 24 hours of use).

Mine ventilation

A linear decrease in operator DP levels was found for increased airflow when machines were operating alone. When machines were used in combination, the actual results were different from the calculated additive values, being lesser in the low flow conditions and higher in the air flows greater than 12 m/s. Thermal stratification in the tunnel was observed.
only at flow rates less than 6 m³/s. The reason for the difference in additive effects of multiple machinery remains unclear.

At some flow rates thermal stratification may result in higher exposures to some members of the workforce.

In situations where multiple vehicle use occurs in some headings, high DP levels have been reported. A tag system has been implemented at Tower to prevent the use of machines in areas where the ventilation requirements may be exceeded.

**Routine underground diesel test bay**

The underground research test bay has been developed into a routine test procedure for all machines. Each operator once per week drives their machine into the test bay and enters the machine number into the computer which indicates the statutory air flow requirements. The operator adjusts the air flow then runs the machine under an idle and short load cycle. Inbye gas sensors record the exhaust emissions and indicate the status of compliance to the driver, the information is automatically recorded in the mine computer records. The driver must obtain a satisfactory printout for the machine before continuing further into the mine, failed machinery is immediately returned to the service bay. This system has been found to be very successful in preventing poorly tuned engines from entering work areas and in predicting early deterioration of machinery so that maintenance measures can be taken prior to it failing the statutory gas testing.

Currently a system is being researched that may facilitate gas testing as well as testing of DP levels under the same test bay conditions.

**Employee exposure profile**

During the research project some 203 personal DP samples were collected over a 18 month period to profile the exposures for all job types at Tower Colliery. In normal operating conditions, exposures ranged from 0.05 to 0.4 mg/M³. This increased during extreme load conditions such as long wall change to between 0.4 and 0.6 mg/M³. For tasks linked to diesel engine activity a linear relationship was found between exposure and number of hours of driving (Pratt, 1995). Overall a reduction of DP exposures at Tower has been observed as various control technologies have been introduced.

The DP monitoring program was extended to a number of operations within the NSW underground coal mining industry, with diesel activities sufficiently different to that of Tower Colliery. Railtrack and diesel locomotive haulage were identified as two such areas that needed to be monitored. A group of eight mines was selected that met the required criteria. Each colliery was contacted seeking their assistance and followed up with a site visit by the Project Co-ordinator in order to outline the research and to seek their commitment to the project. During this pre-sampling site visit, considerable time was allocated to the selection of those activities which would provide the most useful data in terms of differences to Tower Colliery. At least one operation similar to that at Tower Colliery was included in the tasks to be sampled as a control. Sampling was then undertaken at the eight collieries in the period July to December 1994, with the aim of obtaining a minimum of six full shifts of sampling per mine per specific activity. All samples collected were on a full shift personal basis with a minimum of four hours sampling duration. In all 134 personal samples were collected at the eight collieries.

While direct comparisons between exposures found at individual collieries is not possible because of varying ventilation and duty cycle requirements, Colliery C appeared to have consistently lower results relative to other operations, including Tower Colliery. Discussions with colliery management indicated the use of a low sulphur diesel fuel, good road conditions throughout the mine which minimised engine revving, an intensive scheduled maintenance program, a system for limiting the number of vehicles in ventilation splits, a computerised weekly exhaust emission testing program, and a policy of replacement of "older" design engines had lead to the improved conditions.

**Future exposure standards and legislative requirements for diesel particulates**

A proposed occupational exposure standard (TLV's) for diesel particulates have been published in 1995 and 1997 by the American Conference of Governmental Industrial Hygienists (ACGIH, 1995). Worksafe Australia has commenced viewing these standards and presumably they will flow on to the Australian coal industry. MSHA in the USA has indicated that they are close to releasing mechanisms by which a diesel particulate exposure standard would be set. MSHA have also released a "toolbox" consisting of mechanisms by which DP levels can be reduced in coal mines.

COAL98 Conference Wollongong 18 - 20 February 1998
Monitoring data from USA suggests that most coal mines would have difficulty in meeting the proposed standard of 150 micrograms/M3 (elemental carbon) unless they introduce control technologies similar to that currently being implemented in BHP mines which has been adopted into Guidelines by the NSW Mineral Council (1996).

A survey funded by an existing Joint Coal Board Health and Safety Trust research grant is being conducted in NSW and Queensland coal mines using modern sampling and analytical methods designed to specifically capture and analyse this sub micron fraction of DP and to measure elemental carbon content. The aim is to determine the suitability of these monitoring systems for the Australian coal industry and for applicability to any future exposure standards.

4 Wheel drive project

An ACARP funded research has just commenced on reviewing the current design criteria for underground light duty diesel vehicles in terms of safety and practicability with the view of developing a commercially available four wheel drive vehicle design for use in the non-hazardous zone of NSW underground coal mines. The more efficient electronically controlled engines may provide an alternate and future approach to control of diesel particulates. A successful outcome of the project will also provide substantial benefits to the to the workforce in respect of vehicle comfort plus significant reductions in the initial purchase price of diesel equipment for personnel transportation.

CONCLUSIONS

The research conducted at Tower Colliery has added significantly to the knowledge of the extent employee exposure to DP and to the methods by which exposures can be controlled. The traditional control strategies such as engine and scrubber tank maintenance, regular gaseous emission testing and the requirement of minimum ventilation rates provide considerable control of workforce exposures. Additional controls such as correct fuel quality, regular engine tuning and engine decoking provide further significant overall reductions in DP exposures as well as creating a more pleasant smelling atmosphere in which to work. For specific tasks which emit high diesel particulate levels, particularly under heavy load conditions, additional controls in the form of disposable filters fitted to the exhaust outlet appear to provide the most effective method of control at this stage. The most efficient and short - term cost effective overall control strategy is to use a combination of individual systems modelled to suit equipment and operations conducted at each specific mine.

Users of diesel equipment in underground mines are rapidly reaching a situation where they will need to comply with regulations that specifically limit the extent of employee exposure to diesel particulates. The control strategies tested in research described in this paper are now being successfully incorporated into the regular mining activities at Tower Colliery, the four other BHP mines as well as being adopted as a Code of Practice for all mining operations in NSW and hence will assist the industry in meeting future legislative requirements as well as improving the working conditions of the miners. Research which is in progress will move the industry to a new level of competitiveness in the future.

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Disclaimer: The results and opinions presented in this paper are the result of research conducted by the authors and do not necessarily reflect the views and policies of the organisations for which they work.
Gateroad Development in Thick Seams Using the Joy Sump Shearer

B Ward¹ and M Downs²

ABSTRACT

The use of a cutting machine designed to allow simultaneous cutting and setting of roof supports has numerous advantages for development roads in coalmines. The use of such a machine with the added ability to cut a curved roof profile is of considerable interest for longwall development in thick coal.

INTRODUCTION

Gateroad development in Australia has evolved in the direction of integrated activity around cutting and bolting machines, in line with the thinking that gateroads were essentially service tunnels for the longwall. This has lead to the focus on the single pass continuous miner (CM) fitted with integral bolting rigs.

Several new generation machines have been designed to progress this line of thinking by sumping in the cutting head to permit simultaneous roof bolting and cutting. One of these new machines is the Joy Sump Shearer (JSS). It differs from a CM in having the cutting heads on ranging arms like a longwall shearer, which can allow it to cut a variable heading profile.

This paper presents a case for using the JSS for gateroad development in thick seams in weak strata, whereby a curved roof profile can be formed to improve stability. A spiral arch roof support configuration is designed to improve the efficiency of the cutting/bolting cycle, which together with an integrated coal clearance system using a mobile boot end, should permit faster, safer gateroad development.

ARCHED ROOF PROFILE

Historically, heading development has been carried out by a continuous miner, which is designed essentially as a high volume coal cutting machine. With a cutting head rotating about an axis parallel to the face with vertical ranging, a CM can only form a rectangular heading section. This has obvious advantages in coal mining where coal seams are tabular bodies bounded by stone roof and stone floor. However, a rectangular section is inherently less stable than a curved, or arched roof section, particularly in a high stress environment where mining induced stresses are concentrated around the corners.

An example of smoothing out the stress trajectories around curved rib profiles was demonstrated by the Dosco In Seam Miner at Ellalong Colliery (Wallman, 1982). The semicircular rib profile provided a spectacular case of improved gateroad stability, unfortunately, the system did not achieve adequate advance rates.

Another example was at Western Collieries in the Collie Basin. AM75 roadheaders with mobile boot ends were used to drive the main headings in a thick seam environment with a CM for pillar extraction. The roadheader normally cut a rectangular profile but an arch profile was used when traversing the intermittent high stress zones, which would otherwise have proved extremely difficult to mine through due to roof instability (Misich, 1993).

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It is a general rule in geotechnical design that the greatest benefit in stability is always gained through geometric change. That is, it is always better to flow with the stress than to try and resist it through increased support. An arched roof is inherently more stable than a flat roof and thus will require less support than a flat roof in the same conditions. Less support means a potential for increased rate of drivage.

The JSS, because of its cutting head configuration, can be programmed to cut a range of curved roof profiles, from a flat rectangular section to a full arch, and hence, unlike the CM, can provide the opportunity to utilise the advantages of heading profile geometry.

Cutting stone roof to achieve an arched profile is obviously most unattractive in all but extreme circumstances. For the majority of Australian longwall mines, operating in a seam thickness from 2.2m to 2.8m, arched roofs are not an option. In a thick seam operation, however, this is no longer a constraint.

The northern part of the Bowen Basin coalfield in Queensland is characterised by seams with thicknesses in excess of 4m. Of particular interest in the context of this paper is the Goonyalla Middle Seam (GM), which is typically around 5.0m to 5.5m thick in current and prospective underground mining areas. In a seam of this thickness it is possible to have in seam development with a curved roof profile, with adequate dimension to meet ventilation requirements and without incurring unwanted dilution from stone roof.

JOY SUMP SHEARER

The JSS represents an innovative and radical approach to underground heading development. It has been designed as a single pass machine configured to allow simultaneous cutting and bolting. It differs from a conventional CM design in having the coal cutting function performed by twin 1.1m diameter cutter drums on ranging arms, rotating about axes perpendicular to the face, in a similar fashion to a longwall shearer. Coal clearance is via an east-west face conveyor, transferring to a through the body conveyor to the tail discharge.

The frontal aspect of JSS is shown in Fig.
The coal cutting equipment is separated from the working platform behind it by a face shield and side doors, which enclose the front of the JSS to allow any build up of gas or dust to be exhausted through the body of the machine into the mine ventilation system. This enables the operatives to install support whilst the machine is cutting coal.

The JSS is programmed for automatic sumping and profiling, the operator typically changing only mining height and floor options to match geological conditions. The cutter heads are sumped in (0.5m sump depth) by driving the machine forward, after which they cut the programmed heading profile. Bolts are installed during the profile cut. Roof support consumables are carried in pods on the machine with mechanised loading at cut-throughs for re-supply.

Roof support is installed by twin on board drills mounted immediately behind the shield. The configuration is such that bolts are placed approximately 1.5m from the face.

**ROOF SUPPORT**

Roof support design for gateroads should be aimed at providing the minimum amount needed in the short term to advance the heading, without prejudice to safety, such that any additional or secondary support can be placed later without slowing development rates. In addition the system must be sufficiently flexible to allow for variation or additional support to match any changes in ground conditions, such as when penetrating faults or zones of structural disturbance.

The proposed design for the JSS is based on cutting a partial arch in a 5.0m wide heading. Fig. 2 shows the general configuration. The arch has a 3.4m radius giving a maximum height of 3.3m with 2.2m high ribs. The cross sectional area is 15.2 m$^2$, equivalent to a conventional rectangular heading 5.2m wide by 2.9m high. Ideally a fuller arch is desirable (3.0m radius) but this would entail considerable re-engineering and is not possible with the currently available JSS. The proposed design is thus the best compromise that is practically achievable.

![Fig. 2 - Heading profile and roof support](image-url)
The arch is reinforced with a basic spiral arch pattern of 5 x 1.8m roofbolts arranged radially, or as near radial as possible with the twin bolting rigs. The three bolts on the pillar side are conventional T grade steel, the two on the panel side are plastic (i.e. cuttable) for reasons discussed later. The bolting arrangement is reversed in the tailgate.

Bolt spacing along the gateroad is linked to the 0.5m sump depth for the basic pattern, which gives a longitudinal bolt spacing of 2.0m (four sumps). The bolting pattern is thus equivalent to 5 bolts per 2.0m spacing but the pattern is arranged as a spiral arch rather than transversely in line. Fig. 3 shows the basic spiral arch pattern. Bolts are installed at 0.5m centres on the advance whilst the computer controlled profile is being cut. As bolt installation is linked to the sumping it will ensure a uniform spacing and density is achieved.

![Spiral arch roof support pattern](image)

**Fig. 3 - Spiral arch roof support pattern**

Two on board bolting rigs are required for the sequence. The two outer bolts are installed simultaneously with the other three installed at the rate of one per sump. By this means machine advance and coal cutting will not be compromised by the bolting sequence. The two bolt types are segregated such that only steel bolts are placed by the bolting rig on the pillar side whilst only plastic bolts are installed by the other rig.

The system is flexible enough to allow the support density to be increased without interfering with the cycle. Fig. 4 shows the next level of support, a herring bone pattern, whereby bolts are installed at 1.5m spacing. In this case two pairs are installed simultaneously but again, only one bolt per rig is installed in any cutting cycle.

Primary rib support is not envisaged as being necessary as the partial arch shape keeps rib height to 2.2m in contrast to current heights of 3.0m plus with rectangular sections.
In the event that difficult ground conditions are encountered, such as in fault or shear zones where the roof strata may be more closely jointed or disturbed, additional primary support can be placed radially at each sump. The configuration of the JSS enables bolts to be placed 1.5m from the face. Cuttable plastic mesh panels can be installed in extreme cases, or where roof stability would be at risk from subsequent abutment loads.

Another major potential benefit of the arch profile is that any subsequent abutment stresses during extraction will be more evenly distributed around the roof and into the ribs. This could reduce the need for additional passive support such as cribs. Secondary support, if needed, is envisaged as 2 x 6m resin-grouted tendons at 2.0m or 2.5m centres. These would be installed subsequently by outbye crews or contractors with mobile equipment. A single timber or fibre crib would be placed across each cut-through as per normal practice to restrict intersection span, prior to extraction.

**DEVELOPMENT SYSTEM**

**Mining system**

The system proposed for gateroad development comprises two JSS driving each entry simultaneously. The section conveyor has a mobile with the JSS driving the conveyor heading discharging coal onto the section conveyor via a belt bridge. The travelling road and most of the cut-through would be driven by the second JSS, operating with a ram car/battery hauler loading coal onto the section conveyor via a side loading belt.

Both JSS would be supported by inbye mono-rail mounted services, each mono-rail system being independent and of similar format. The inbye mono-rails would be in turn supported by an outbye mono-rail system, this being designed to facilitate the easy advancement of major section services.
The intensive use of mono-rail systems eliminates much of the repetitive down-time associated with advancing services as mining progresses. Significant benefits accrue from mechanising such functions, including enhanced safety due to much reduced handling of sundry pieces of equipment, the elimination of production delays normally associated with moving services forward, possible face labour reductions and greater speed of cyclic section advances.

The outbye mono-rail system also mechanises a previously labour intensive and time consuming activity by enabling easy and rapid advancement of the section transformer. Again, general safety is greatly improved by the elimination of the significant and usually manually completed task of advancing the feeder cables.

**Equipment**

The general arrangement of the face area equipment and the inbye mono-rail system is shown in Fig. 5.

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**Fig. 5 - Schematic arrangement of face area equipment**

JSS 'A' would drive the travelling road supported by a ram car or battery hauler. Coal clearance would be via a side loading belt in the cut-through. This side loading belt would be a readily demountable unit, with wheeled telescopic legs at the discharge end and retracting wheels at the loading end. When in operation, the loading end would be lowered to the floor and held in position by dowels and turnbuckles.

JSS 'B' in the conveyor road heading would discharge coal via a belt bridge which would be attached to the end of the JSS conveyor boom by a swivelling joint with a sliding section over a mobile boot-end. The use of the belt bridge enables a continuous coal throughput to be achieved whilst allowing the ability for the miner to pull back from the face - either to clean up or for maintenance - and also to partially form the cut-through.
Services (ventilation, power, water and compressed air) would be provided by the inbye mono-rails. Both mono-rails would also be of similar format, with a combination of flexible ducting and rigid sections as dictated by the specific requirements of the installations.

The mono-rail track in the conveyor road would form the basis for the longwall mono-rail system, consequently being erected on the pillar side of the roadway and left in position as the section advances. This monorail would be suspended from the middle steel roof bolt on the pillar side.

The mono-rail system in the travelling roadway would be erected on the panel side of the road, so as to avoid impacting on shuttle car load clearances at the cut-through, and salvaged as the section advances inbye for re-use at the working face. This temporary suspension would be on the plastic bolts.

The inbye mono-rails would be in turn supported by an outbye mono-rail system in the travelling road, accommodating services from the section transformer location/end of pipe range area, to the outbye end of the face mono-rails. The outbye mono-rail system and equipment is shown schematically in Fig. 6.

![Fig. 6 - Schematic arrangement of outbye section equipment](image)
The outbye mono-rail would be suspended from the pillar side steel roof bolts and would supply the following:

- power in the form of feeder cables from the section transformer to the load centre(s);
- water, through large diameter hoses to supply face dust suppression requirements as well as high volumes required for fire hydrants mounted on certain of the carriages;
- compressed air for both the mono-rail system "mule" drives and for general use; and
- sundry monitoring and communication cabling.

Drivage sequence

The section drivage sequence is shown by Fig. 7, which also broadly defines the split in requirements between the two JSS development machines. JSS 'B', with its attendant belt bridge, is required to form approximately 5m of the cut-through so that JSS 'A', which cuts the remainder of the cut-through, does not have to hole through in an area that would compromise equipment in the roadway, such as the mono-rail and the conveyor.

The inbye mono-rail system advances with each of the JSS, automatically bringing forward ventilation and power and water services. At the end of the designated drivage, the outbye end of each of the face mono-rails is moved forward and causes the flexible sections to compress as they are bunched up against the rigid, inbye ends. In this way, there is no manual handling of face cables, hoses, or ventilation tubes.

The outbye system also removes two activities off the critical path for a section advance, these being the extension of the pipe ranges and the extension of the H.T. cable. Both of these functions can be completed at any time over at least two complete cycles (i.e. two pillar lengths of section advance), without any delay to the production at either face.

Productivity

A cross-sectional area of 15.2 m² and a web depth of 0.5m will produce approximately 10.6t per sump cycle, that is one load for the coal hauler. A cutting and loading capacity of 10 tpm is assumed for the JSS.

An approximation of the productivity of JSS 'A' has been approached by considering the likely working times in a median position in the overall cutting cycle. It is assumed that a point 90m up the travelling road from the loading cut-through is representative of an average face position for this machine. Roof-bolting time per bolt is assumed to be 2 minutes.

For JSS 'A' the major component in the cutting cycle is the travel time of the coal hauler. Analysis indicates a production rate of some 120 tph, giving an output of about 780 t/shift, equivalent to a face advance of 39 m/shift, for an effective face time of 6.5 hours. Discounting performance to reflect operating efficiencies and system reliability by a factor of 0.75, indicates that the expected shift advance of JSS 'A' would be 30m (producing about 600t ROM).

The duty cycle for JSS 'A' only involves coal cutting, roof support and coal clearance functions. Hence the face crew can be limited to 4 persons.

JSS 'B' employs a belt bridge that should virtually eliminate coal clearance delays. Some loss of production time will occur when the machine is pulled back to clean up the roadway and possibly to enable a correction to be made to the recently installed conveyor structure but these are assumed to be included in the general discount factor for the purposes of this paper.
The only functions to be considered for JSS 'B' are cutting, loading and roof support. The roof support cycle of 2 minutes realistic minimum bolting time exceeds the nominal cutting and loading time of 1 minute, as no bolting is considered possible during the machine sumping function. The sumping action is presumed to take 1 minute. The total cycle for this face is therefore 3 minutes for an output of 10.6t.

For an effective face time of 6.5 hours, this would produce an output of 1300t ROM, is equivalent to a shift advance of 65m. Again, applying a discount factor of 0.75 to reflect operating efficiencies and system reliability, the expected performance of JSS 'B' would be a face advance of 48m per shift, producing an output of 975t.

This rate of face advance will require a conveyor extension of approximately 11 bays of structure. Minimal effort would be required to advance the general face services due to the mono-rail facility. The installation of 11 bays of structure is considered a relatively easy task to achieve in a shift and would require an intermittent maximum of 4 persons to complete.

The indicative manning for JSS 'B' would be 3 operators (including supervisor), 1 tradesman (probably electrical allowing for mechanical training of operator) plus 2 persons for backbye support. This gives a total complement of 6.
Backbye work in what is essentially a "super section" will comprise the need for rapid services extensions and general section support work. It is estimated that these functions would require an outbye crew of 4 multi-skilled operators to keep pace with the face advance. Therefore, the overall manning level for the mining system outlined would be 14 per shift.

Critical path analysis indicates that the overall section cycle is likely to be of the order of 8 shifts - assuming that the key parameters of advance rates and support work duration are met.

The rate of section advance implied by the cycle time is approximately 2 x 100m pillars advance per week. This is equivalent to some 920m per machine per month, which is close to current industry best performance in simple terms, but which when included in the system described results in an excellent rate of section linear advance.

INTEGRATION WITH LONGWALL

One issue with thick seam longwall extraction is managing the disparity between longwall mining height and the gateroad height. Gateroads are normally restricted to a maximum of 3.5m high, which will leave a differential of up to possibly 1.5m compared to the extraction height. High gateroads are not desirable from a geotechnical point as they increase the potential for roof and rib instability. In addition they require specialised lifting equipment for operators installing secondary support and create onerous conditions for erecting crib support with attendant risk of injury.

The general solution to date has been to ramp the longwall face either down to match the roof line or up to match the floor line, depending on whether coal has been left on the floor for trafficking in the gateroads. Whichever way, managing the ramping involves constraints or controls on the longwall operation (delays to shearing cycle) and also some loss of coal that could otherwise be mined.

The provision of cuttable bolts on the panel side of the gateroad roof is designed to permit the shearer cutting height to be maintained into the gateroads without change of horizon. By this means the complication and inefficiency of roof ramping will be avoided whilst still retaining the desired stability of a moderate gateroad height.

Fig. 8 shows the maingate configuration. Development is at or close to floor level as trafficking is not such an issue with the mobile boot end. A coal floor thickness of 0.5m of has been allowed in the design as a degree of conservatism with regard to the weak floor associated with GM Seam. The floor is thus ramped up to meet this. At roof level the shearer cuts in over the panel side of the gateroad through the cuttable bolts without changing horizon.

The main complication is in setting the maingate chocks against the roof. With 1.75m chock shields there will be two chocks in the maingate, both of which are set against the curved roof section at a lower height. Chock #2 will have to set
against the remnant coal roof fillet in the crown. As the goaf break line is inbye of the gatend, these chocks are not needed
to control goafing, their primary purpose being to advance the stage loader and protect against falls of loose rock. A
reduced setting pressure can thus be applied to these chocks to enable Chock #2 to be set safely against the roof fillet.

Chock #2 will be provided with a part elevated canopy top to match the roof geometry, depending on the final
longwall/chock design configuration, to prevent any breakage from the coal fillet from flushing sideways beneath its
 canopy. Any downdip face creep will automatically be accommodated in the design as the roof cut will always be in the
same place relative to the chocks out by taking the shearer to the same finishing point on the AFC. Racking potential will
also be reduced with lower height maingate chocks.

Fig. 9 shows the tailgate configuration. Approximately 0.8m of coal is left in the floor as a trafficking surface for the coal
 haulers. The shearer cuts into the floor, as per current practice in the Bowen Basin, and cuts into the roof as per the
maingate, thus permitting full extraction height to be maintained. Only one chock is present in part in the tailgate so that
special chock design modifications are not needed.

Fig. 9 - Tailgate end configuration

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coincide with the views of BHP Australia Coal.
Application of Virtual Reality Technology to Mine Management

R Mark and C Mallett

ABSTRACT

Coal mines are dynamic systems that change continually in size, shape, and condition. Large quantities of data about production, coal quality, ground stability, ventilation monitoring, equipment performance and location, vehicle and personnel movements and geology, are generated from all parts of the mine.

Internet technology and virtual reality technology are now being combined to provide an intuitive, multi-dimensional information system which can simultaneously display any information that can be collected and stored by a computer. Physical information is directly displayed as 3-dimensional objects in the mine model. Numerical and text information is dynamically linked to these objects and accessed by "clicking" on them.

All of the data can be integrated in real time into one information system with a 3-D graphical interface and user friendly controls. This interface will facilitate easy access to and integration of everything from real-time gas monitoring data and vehicle location to exploration drill cores, all within one application. Operators will use one simple intuitive interface which will manage information from the myriad of computer packages with different data structures, interfaces, languages and operating systems.

INTRODUCTION

This paper is about the future, the near future. Coal mining has become a very competitive business. International market forces continue to push the price of coal down while many of the costs of doing business continue to rise.

Community expectations of environmental and safety performance has changed the role of the mine management to include more than just maximising production while minimising cost. "Due diligence" is the catch cry of the nineties, requiring that mine managers weigh up all relevant information and make the "diligent" decision.

Improving technology including automation, instrumentation and advance geophysical techniques all combine to provide an overwhelming quantity of data. Increased expectations from the community and increased pressure from shareholders to keep the mine profitable in a tough economic climate require that good decisions be made quickly and with high degrees of confidence. Currently, it is difficult for mine operators to access all of the data necessary to make good decisions.

It has not been possible to collect and communicate the volume of information pertinent to controlling safety and productivity in a coal mine, but new systems are now available.

Virtual reality technology has been combined with the flexibility of internet communications by CSIRO Exploration and Mining to create a new method for integrating and communicating mine information. CSIRO has not developed the virtual reality software but has developed mine applications using freely available software. CSIRO developed this methodology as a tool for integrating disparate geological and mine data which were independently collected and manipulated using a lot of different software packages. The aim was to see relationships that were not apparent before the datasets could be combined. Key advantages of this new visualisation method are the ease of delivery of data, the intuitive interface and the fact that it is based on existing, readily available software. Less than 15 minutes training is required for anyone to begin "seeing" meaningful relationships between data sets.

1 CSIRO - Exploration and Mining
HOW CAN VIRTUAL REALITY TECHNOLOGY IMPROVE MINE PERFORMANCE

Mine planning and operations

Day to day planning and operation of the mine involves the use and generation of data which falls into two broad categories, geology, and engineering. Geology data includes data gathered from drill holes, seismic surveys, and geologists' observations. This data is generally gathered and stored in three-dimensional coordinates with qualitative data linked to stratigraphic and structural units. Engineering data includes the mine plan, services, equipment specifications, ventilation design, and gas drainage. This data is typically stored in 2-D with attributes assigned to objects. As an example, the mine plan is shown in 2-D with the attribute that it is in a particular seam with a working section of X metres with Y metres of floor coal etc.

The potential of this method has been demonstrated via ACARP project C5026, Interactive Evaluation of Mine Plans with Integrated Geological Exploration (LeBlanc Smith, Caris and Soole, 1997). Two mines, one open cut and one underground were reconstructed in "virtual space" by combining and displaying a large number of different data sets. The primary objective of the project was to demonstrate the analytical advantage of 3-dimensional visualisation for mine planning and exploration. During the course of the project geophones were used down bore holes to monitor micro-seismic events associated with the operation of the longwall. The location, size, and character of events were interpreted from monitoring. When these events were replayed in 3 dimensions in time sequence in the virtual mine model and viewed in the context of known faults and joints, major structural problems become apparent which were previously not well understood. This new information was instrumental in the decision to change the mine plan to avoid difficult mining conditions.

The ability to combine datasets from different sources including faults, boreholes, mine plans, micro-seismic monitoring, etc. and to visualise them in four dimensions (3-D + time) adds a whole new ability to see previously obscure data relationships. Another potentially powerful aspect of the tool is the ability to build on previous datasets in real time as the mine progresses. Any data which can be extracted from the mining process in a digital form can potentially be used in real-time to update the model.

It must be stressed that this technology in no way replaces current mine planning or geological software packages it merely provides a method for communicating information and interacting with data from those packages and a variety of other sources simultaneously. This tool will enable persons involved in critical decision making processes to have access to more data more quickly in order to gain improvements in the quality of decisions and reduction of time required to gather information Table 1 displays how this technology might be used to enhance the understanding of mine data.

COMMUNICATIONS

Better communication is the most direct benefit of this type of technology. The old saying goes "a picture is worth a thousand words"; if that is true a virtual world is worth a million.

On-site

Every person who has access to a computer on the mine site can potentially access the data which is pertinent to their jobs without installing any highly specialised software. This in itself is not unique and is already being done. What makes virtual reality technology unique, is that the data is communicated in a "real" framework with physical representations of the real world that add value and context to the data. The user is also empowered to see relationships between various data sets which previously were very difficult and time consuming to construct.

This system also has the power to deliver real-time monitoring information to mine operators in context. For example, if an alarm generated by the system indicated that gas levels are exceeding hazardous levels and at the same time personnel and vehicle location are displayed, then an operator will be able to immediately assess potential hazards by seeing whether the area of the mine in which hazardous conditions are detected is occupied or unoccupied.
Off-site

There are no technological barriers to prevent the transfer of information across the internet in real or near real time. This technology would enable a mine operator to supervise exploration drilling from head-office, monitor progress in real time, and discuss the data with a consulting geologist while you are both accessing and viewing it in 3-D, in the context of other relevant mine data, from separate locations via the internet.

To the rest of the world

Mining companies are under increasing pressure to communicate with the outside world about the activities of their mine. Consultants, shareholders, government agencies and board members all have a need to understand what is happening at the mine. Different individuals will be interested in seeing different sets of data and the relationships between different sets of data. Currently it is very difficult for a lay person to visualise what a mine is like or to understand the mining process. This tool allows real mine data to be communicated in a format that can be easily understood by lay people. Investors, for example, would be able to tour a planned or operating mine on their desktop and to understand the planned recovery of resource and its potential value.

Corporate memory

Retrieving historical data or engineering experience is usually very difficult in mines. Information is effectively lost when staff move on. Repeated errors commonly result when new staff have no corporate memory of the mining experience to call on. In a virtual mine system, data once entered cannot be lost, and is always available to review along side the current situation.

Table 1 - Displays how this technology might be used to enhance the understanding of mine data

<table>
<thead>
<tr>
<th>Data Set</th>
<th>Advantage of Virtual Reality (standalone)</th>
<th>Possible data set combination</th>
<th>Resulting new or improved capability (when combined)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine Plan</td>
<td>None</td>
<td>Geology, time</td>
<td>Better real-time control</td>
</tr>
<tr>
<td>Equipment</td>
<td>None</td>
<td>Stores, services, mine plan geology, time, longwall.</td>
<td>Automation, maintenance scheduling, efficiency improved reliability</td>
</tr>
<tr>
<td>Ventilation</td>
<td>3 D plus time</td>
<td>Geology, gas model, real time monitoring.</td>
<td>Visualisation (real time), simulations</td>
</tr>
<tr>
<td>Gas Drainage</td>
<td>3 D plus time</td>
<td>Geology, gas model, real time monitoring, drill holes.</td>
<td>Visualisation (real time), simulations</td>
</tr>
<tr>
<td>Geotechnical</td>
<td>3D plus time</td>
<td>Microseismic, stress, faults Mine Plan, Longwall face, etc.</td>
<td>Prevention of accidents, Improved visualisation.</td>
</tr>
<tr>
<td>Safety</td>
<td>Combination of various data sets ie. equipment geotechnical and services etc.</td>
<td></td>
<td>Prediction of hazardous conditions</td>
</tr>
<tr>
<td>Emergency</td>
<td>Combination of various data sets ie. equipment geotechnical, personnel location, services, etc.</td>
<td></td>
<td>Emergency Management System providing rapid reconnaissance of all available data, quality assurance that all data that is available can be assessed. Training for self escape.</td>
</tr>
<tr>
<td>Surface Mine</td>
<td>3D plus time</td>
<td>High wall monitoring, vehicle movement, stockpile monitoring, etc.</td>
<td>Visualisation of process flow / efficiency of haulage. Vehicle and personnel tracking.</td>
</tr>
</tbody>
</table>
HOW DOES IT WORK?

The system which will be referred to as the "Virtual Mine" displays graphical data (any data that can be represented by an object located in x,y,z coordinates) as objects in an interactive "world" constructed using Virtual Reality Modelling Language (VRML), non graphical data (data which can be represented by a numerical value or a string of text) can be displayed in a number of ways: as an attribute of an object eg. color, transparency; as a structured query linked to a database via a JAVA applet (small program which links data to an object); or as an HTML (HyperText Markup Language) page which comes up in a separate window when a link is activated. All forms of data can be linked to either an object within the VRML world or a control button on the console of the VRML browser.

The software necessary to interact with the Virtual Mine is readily available on the internet, most PC's and workstations now have the appropriate software installed on them from the factory. An internet browser (Netscape, Explorer, etc.) and a VRML plug-in browser (Cosmo-Player) are all that's required. The images contained within this paper are screen shots of Netscape, and Cosmo Player.

While the techniques to build virtual mines are established the process is still somewhat tedious. Graphical data from the various geological and mine planning packages has to be converted to VRML using data translators which have to be programmed separately for every different data format. Databases containing the non-graphical data have to be structured and programs written to interpret that structure and retrieve the data. While anyone can easily use and view the data once it is constructed the set up entails considerable time and effort. This process is being continually improved with better translators and faster data acquisition methods.

Conversion of data from existing data sets remains the most difficult task in constructing the virtual mine. In other areas of design, software vendors are beginning to provide VRML converters for 3-D output. If this trend continues and finds its way into the mining industry then the task will become much easier.

Because the virtual mine uses internet technology there are no inherent hardware requirements. Obviously, hardware limits the graphics quality, speed and quantity of data which can be accessed at any one time. However, internet applications are developed to be platform independent so there are no inherent hardware biases. PC, MacIntosh, or workstation, it doesn't matter so long as the computer has sufficient power to handle the data required at a speed which is acceptable to the user. The structure of the virtual mine also allows staff members in any field of expertise to access and use the data in for their own purposes in any combination.

WHAT HAS BEEN ACCOMPLISHED SO FAR

Two "virtual mines" have been constructed, one open cut and one underground both based on existing mines and illustrated in Figs. 1 and 2. Each one incorporates a variety of different data types, including:

- geological data, coal seams, faults, dykes, joints and strata information boreholes;
- geophysical data, seismic, microseismic, stress;
- mine planning data;
- surface data, digital terrain models of surface topography, terrestrial photogrammetry, and aerial photography;
- 'ideo clips;
- animations of equipment movement; and
- monitoring data simulations.
The demonstration projects have been successful in demonstrating the potential of the virtual mine as a visualisation and analysis tool. In the underground virtual mine, the ability to visualise in three physical dimensions plus time, the occurrence of microseismic events (rock movement) associated with the advance of a longwall face at the same time as the fault structure was instrumental in the decision making process.
The open cut virtual mine demonstrated the ability to integrate photogrammetric data both areal and terrestrial with survey and drill-core data to add meaning and to improve the mine model. The open cut mine also demonstrated the ability to incorporate video clips and streamed location data (for mine trucks) into the system as shown in Fig. 3.

In addition, the underground virtual mine was used as basis for simulating the display of real time data. Gas monitoring and personnel tracking were simulated by generating a series of numbers and text information which were then streamed into the model and displayed graphically and as text obtained by clicking on an object (monitoring location) in the mine.

![Fig. 3 - Open cut mine showing boreholes, vehicle location, video panel and 3-D terrestrial photogrammetry integrated into the model](image)

**PLANS FOR THE FUTURE**

The virtual mine developments by CSIRO prove the value of the tool to existing mine operations, with integration of disparate data sets and the intuitive display of complex data. It represents a critical step toward the holy grail of a comprehensively monitored and controlled mining system. Although many potential mining applications remain to be explored, current applications could be implemented by mining service providers now, and plans for the commercial implementation of the technology are underway. It is hoped that it will be available as a commercial service to the mining industry this year. The package would include the integration of geoscience data, survey data, photogrammetry, mine design and time sequenced monitoring data from various sources.

CSIRO is continuing research in a number of areas. The first of these is in improved software capabilities. The software tools are being developed on a worldwide basis, with many organisations developing and releasing products into the internet environment. Keeping pace with these developments and establishing which may be useful for mining applications is a major activity.

Most programs are made with the games and other bulk usage markets in mind, and can rarely be directly used for other purposes without some adaption or interface. For real applications new software is required to allow the automatic updating of VRML files from sensor data, so visualisations can represent real-time data. Another key to the visualisation of large datasets typical of mine sites is techniques which allow sampling of data at a scale appropriate to the application in use. When zooming in on areas of interest, the appropriate level of detail is drawn from a database, without all the data having to be stored continuously. CSIRO is also investigating the virtual workbench, which is a physical workstation
where the operator manipulates tools actually within the 3D model. This includes tactile feedback from surfaces and objects in the workspace.

Another area of development is in the capabilities of different computer hardware systems. A common question is ‘What sort of computer do I need to run this?’ There is not enough experience yet to answer this. All the software can operate on a PC as well as a workstation, but how practical this is for real datasets has to be demonstrated.

Virtual reality techniques open many new opportunities for mining applications, with the possibility of completely new ways of approaching mining tasks. Developing these new opportunities is not only the function of the virtual reality research team, but extends across the areas of mining research. All groups are asking how this new tool can be applied to their areas and are working to exploit its capabilities. In addition to the first underground and surface models we have described here other applications being developed are in gas drainage, ventilation, equipment status, mining system simulation (a virtual longwall), geotechnical performance, microseismic monitoring, safety, emergency response procedures, equipment and vehicle tracking, rehabilitation and environmental monitoring.

To fully implement the potential of many of these applications new or modified equipment, monitoring and communication will be required. An example of these support research developments is a new communication system (LAMPS) which is to provide robust and wireless communication for data throughout an underground mine.

It could be an essential module of a personnel and equipment tracking and monitoring in a virtual mine model. Of more interest in surface mines are techniques for highspeed mapping of exposures such as highwalls with photogrammetry and laser ranging technologies. The output of these methods could be incorporated directly into the 3D virtual mine datasets.

A feature of a fully developed virtual mine will be the capability to quickly integrate data from different areas. Although we are working to find ways to utilise this capability in operations, it will obviously be of enormous benefit for the times when things do not go as planned. If an event occurs it will be possible to see who might be the cause, what the effect on the rest of the operation might be, and the status of the remedial services. In the extreme case of a major mine emergency this technology will provide an invaluable tool to assess the situation and a basis for rescue and recovery of the mine. In its simulation mode, a virtual mine system could be used to develop training and establish operating procedures.

**CONCLUSION**

Virtual mine technology is opening the window on the mine of the 21st century. The virtual reality technology provides one of the key building blocks required for safe, regulated, and optimised mining.

It is essentially a communication tool to present all relevant data in an understandable format for decision making. It provides a way to reduce very complex and diverse data sets into the essential elements without loss of data quality. It turns data into powerful information.

Virtual reality technology will impact profoundly on all aspects of society, and the mining industry has the opportunity to share in the benefits. It is not just a nifty bit of software that kids play games with. It is a paradigm shift in our ability to manage our increasingly complex activities.

The technology is available now, and applications will be introduced throughout the industry in the years ahead, in areas where it provides new capabilities. As its use spreads we will reach a position where a mine will be able to access and visualise all its data in 3D and through time, and all decisions made in the light of best knowledge and historical experience.

**REFERENCES**

Combining Modern Assessment Methods to Improve Understanding of Longwall Geomechanics

M Kelly¹, W Gale², P Hatherly¹, R Balusu¹ and X Luo¹

ABSTRACT

Ongoing, collaborative research between CSIRO’s Exploration and Mining and Strata Control Technology has resulted in a better understanding of rock failure mechanisms around longwall extraction. Failure has occurred further ahead of the retreating face than predicted by conventional longwall geomechanics theory. In some cases, significant failure has been detected several hundred metres ahead of the face position with demonstrated influences of minor geological discontinuities. Shear, rather than tensile failure has been the predominant failure mechanism in the environments monitored. Validating technologies of microseismic monitoring and new face monitoring techniques have assisted the development of predictive 2D computational modelling tools. The demonstrated 3D consequences of failure has assisted in the ongoing direction of the project to further investigate these effects.

INTRODUCTION

Despite the impressive growth in Australia’s longwall production, many longwall mines have experienced major geotechnical problems. Unexpected geological intrusions and fault zones have resulted in loss of production over extended periods in some mines. Stress concentration in gate roads is another a major problem. In some mines, high gas emissions have resulted in production delays up to 20%, leading to lower production and productivity. Water ingress from overlying aquifers or water courses has also been an issue. In the past five years, cyclic loading under massive strata and the associated problems with face stability and wind blasts has also occurred. Subsidence of surface features, roads, water bodies, dams are some of the other problems which have restricted the longwall operations.

These types of problems are not just confined to Australia and are common in many countries, restricting the production potential of modern longwall faces. In addition, longwall panels have been trending both wider and longer to increase productivity. To support these larger capacity longwalls and to reduce the risk of occurrences that limit productivity, there needs to be an improvement in the understanding of longwall fracturing processes.

In order to improve understanding of longwall fracturing and caving, research into caving processes across a broad range of underground environments is being undertaken by CSIRO Exploration and Mining and Strata Control Technology (SCD). In this paper there is a brief review of the literature, a description of the methods used in this work, and a discussion of the results and insights into longwall caving process that have been obtained.

BRIEF LITERATURE REVIEW

There have been many studies into the geomechanical behaviour of longwall faces, and there is considerable variation in both the approach and underlying assumptions. Attempts have been made to analyse the caving mechanism using numerical models, empirical models, physical models and various forms of field measurement techniques. Most researchers link the extent of fracturing to extraction thickness and a number of relationships have been proposed to predict fracturing extent (Wilson, 1964, 1983; Peng and Chiang, 1984; Kidbyinski & Babcock, 1973; Zhu, Qian & Peng 1989; Whittaker, Gaskell and Reddish, 1990). These studies suggest that front abutment pressure reaches a peak value approximately 3 to 5 m in front of the face and is about 4 to 6 times the overburden pressure.

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Most studies suggest that there is vertical fracturing due to tensile failure ahead of the face, because the peak front abutment stress is thought to be very close to the face. However, such models are contradicted by time domain reflectometry (TDR) studies (Dowding, Su and O’Connor 1989; Haramy and Fejes, 1992), which suggest shear rather than tensile failures in the roof rock 15 to 20 m ahead of the face. This suggestion also indicates a different type of stress distribution around a longwall face and raises questions about many of the assumptions in current longwall design methods. In addition, knowledge of actual fractures zones and failure mechanisms ahead of the face and in the floor is limited.

Conventional field investigations (Wilson, 1964; Wagner and Steijn, 1979; Christiaens, 1982; Freeman and O’Grady, 1992) using stress cells, extensometers and convergence measurements often refer to the region very close to the face, and are insufficient to understand the rock mass behaviour further ahead of the face. Investigations involving surface boreholes and extensometers, such as camera surveys of borehole walls before and after mining, give some idea on the vertical extent of fracture zones. However, they are difficult to undertake, time consuming and expensive. In addition, the amount of data that can be obtained from such field studies, including TDR monitoring, is limited and is insufficient to characterise the complete caving process.

It would therefore seem that there is no clear understanding of the stress distribution and failure mechanism ahead of a longwall face, and that more comprehensive and thorough investigations are necessary to achieve such an understanding. However, this now appears to be possible with investigations by Sato and Fujii (1988), Styles, Bishop and Toon (1992) in Britain and by Hatherly et al (1995) at Gordonstone Mine showing that microseismic monitoring can provide large sets of three dimensional and dynamic data on stresses and failure mechanisms at minimal cost. Such data provide validation for the numerical simulations, which can now be produced with increasing sophistication and accuracy.

**THE CURRENT STUDY**

This current longwall study is aimed at improving face design and control methods by better defining strata failure mechanisms around the face. It will allow design and predictive tools to be developed for a broad range of underground environments. Specific mines are being studied to allow a rational understanding of the caving process and in turn its relevance to fluid flow, stress distribution and support interaction. Microseismic monitoring, 2D and 3D computational modelling, longwall face monitoring and new soft rock testing procedures are being used.

**Microseismic monitoring**

Most microseismic monitoring is mainly undertaken for the forecast and control of rockbursts, mine bumps and failures (Young, 1993; Coughlin and Rowell, 1993; Miller and Descour, 1996). This is an important application but represents only one aspect of its geotechnical uses. The potential of microseismic monitoring in understanding the longwall caving processes has been largely unexplored until recently.

The microseismic monitoring is utilising arrays of three-component geophones grouted within boreholes drilled from the surface to the working seam. Microseismic activity is monitored continuously as mining progresses. The number of boreholes and location of geophones depends on the geology and geometry of the area being monitored. The system includes a monitoring unit and computer data storage devices installed on the surface in a modified shipping container. The system is self contained with diesel generators providing power and a mobile phone with modem providing remote phone-in access. Geophone orientations and seismic wave velocities are established by firing of shots at known locations.

**Numerical modelling of longwall caving**

The program FLAC (Itasca, 1995) is used as the basic numerical modelling program with additional proprietary rock failure and goaf consolidation subroutines which simulate the longwall caving process and allow more rational prediction of stress distributions and displacements occurring around longwall faces. The code can handle the wide range rock mass failure conditions which exist around a forming goaf and complex fluid and ground deformation capabilities.
The rock failure routines define the orientation and properties of fractures created in the rock mass from various failure models. These are: (i) shear fracture through intact rock, (ii) shear failure along bedding, (iii) tensile fracture of intact rock, and (iv) tensile breakage along bedding planes. In addition, the orientation of pre-existing joints and the failure planes generated are analysed to assess stability under conditions of tension and shear, as the ground moves backwards into the extracted void. The goaf consolidation routine assesses stiffness of the caved material as a function of load and void space. The type of fracture and its orientation are displayed in the output.

The approach used in modelling is to excavate an approximately one metre wide 'web' (shear slice), calculate the ground response and then to repeat the process. The simulations also include coupled fluid pressure as part of the rock failure process. To date, the simulations have been two-dimensional simulations of the central part of a longwall panel where the panel is of supercritical width and the out-of-plane third dimension has least effect.

The question of 3D modelling versus the 2D used in the study needs comment. The level of detail required to adequately represent failure development using a discrete web width of one metre is such that the model requires a three week continuous run on a fast work station. A 3D model of the same detail is therefore not practical. However, we have commenced to model a broader, less detailed situation using the 3D code ABAQUS (Hibbit, Karlson and Sorenson, 1995) to understand the influence of structure and adjacent longwalls on the failure mechanisms of the current block.

Face monitoring

The longwall face monitoring component of this work has involved the monitoring of chock leg pressures, face convergence and goaf pressure. A new longwall powered support closure measurement system which does not depend upon potted wires has been developed for convergence monitoring. This system has several advantages over current closure measurement such as spring tensioned wire transducers. By utilising robust transducers mounted in relatively protected areas on the longwall support, the system has greater reliability. The system also provides additional information on the reaction of the base and canopy to loading, which can be used to help analyse support performance. This system is being used in conjunction with support manufacturer's monitoring packages for the monitoring of both leg pressures and convergence on each longwall face during the microseismic monitoring.

Other technologies

A sophisticated triaxial testing facility at CSIRO is being used to quantify rock and coal strength parameters at field sites. The soft rock triaxial testing facility consists of an automated cell pressure control, measurement and data acquisition system. The system can accommodate samples up to 300 mm in diameter. Pre- and post-failure characteristics and volumetric strain response are typically recorded to within 1% accuracy.

In addition, 'Goafmon', an instrument system designed to measure the load exerted by the caved strata in a goaf, has been used. Goafmon comprises a 400 mm diameter flatjack with associated electronics installed in the floor of a goaf, and connected by robust cables to the controller module located in an adjacent roadway. The frequency with which the pressure is monitored is user selectable, and to maintain the integrity of the data there is a full suite of real time synchronisation and error checking routines.

WEAK ROOF STUDIES - RESULTS FROM GORDONSTONE MINE

The Gordonstone Mine is located in the Bowen Basin in Central Queensland. The geology is described by Kelly, Lawrence and Devey, (1994). In the area of the study the 3 m thick German Creek Seam is being mined at a depth of about 235 m. Mining is by the longwall method, with a face width of 250 m. The immediate roof and floor are particularly weak, with UCS values of only 5 - 15 MPa. Stronger bands (UCS of about 50 MPa) occur above the Corvus Seam, some 25 metres above the worked seam. The dominant horizontal stress is NNE, parallel to the panel direction and sub-parallel to the dominant coal cleat and strata joint directions. The thickness of the Tertiary sediments and volcanics is about 70 m.
Microseismic monitoring results

In 1994 a microseismic study was undertaken with the objective of determining whether caving from longwall mining extended to overlying unconsolidated Tertiary sediments and volcanics Hatherly et al (1995). Three boreholes were drilled, and nine triaxial geophones were installed in each. All but the shallowest were below the Tertiary/Permian unconformity. Piezometer readings to supplement the microseismic data were made at depths of 205 m, 170 m and 125 m, from a hole drilled in the centre of the longwall panel and within the microseismic array. The microseismic activity was monitored during September and October 1994. The activity was closely correlated with mining, and in all 1200 events were detected. Of these, 629 with sharp P-wave onsets were located. The remaining events were of lesser magnitude, with indistinct onsets; these are thought to have been from within the goaf and to have occurred after initial failure.

The locations of the microseismic events are summarised in Figs 1 and 2. They are estimated to have an accuracy generally better than 5 m within the microseismic network and 10 m outside. In plan view (Fig. 1) it is apparent that the majority of the events occurred within and above the panel (LW 103) which was being mined. There is a tendency for events to occur on the sides of the panel, and in cross section (Fig. 2) it can be seen that they generally lie within an envelope at some 15° from the vertical above the gateroads. Fig. 2 also shows that the events extend to a height of about 120 m above the German Creek Seam, and to a depth of about 30 m into the floor. Fig. 3 shows the locations of the events relative to a fixed face position. This figure shows that the events tend to occur up to 100 m ahead of the face and that this seismically active zone extends upwards at an angle of about 50° from the horizontal.

Fig. 1 - Microseismic events location in plan view - weak roof
It has also been possible to determine source mechanisms for a number of the events. The nodal planes are approximately parallel to the longwall face and a compressive shear fracture pattern is indicated with fault planes dipping at an angle of approximately 50° (+/- 10°). Piezometer data confirm these microseismic results. In the upper piezometers, increases in pore pressures occurred up to 170 m ahead of the face and varied according to mining activity. The piezometer cables were also sheared progressively up the hole at distances of 73 m, 53 m, and 25 m ahead of the face respectively. These distances coincide with the onset of the microseismic activity.

Numerical modelling results

The numerical model for Gordonstone, Fig. 4, extends from the seam to the surface and 500 m below the seam. The results of simulated longwall mining at Gordonstone are presented in Fig. 5. A number of key features are indicated:

- Failure of roof strata occurs at a substantial distance in front of the face. The model indicates that this takes place at least 10-15 m ahead of the face, and that it extends up to the Corvus Seam (20-25 m above the worked seam). The fractures are pervasive but have no pattern. The expected microseismic characteristics would therefore be many low intensity energy releases, rather than a (periodic) high energy release of lesser frequency.

- Failure of the immediate floor strata occurs at regular intervals.

- The nature of rock failure ahead of the face is shear fracture through intact material, and bedding plane shear.
Fig. 4 - A section of the Gordonstone geological model

Fig. 5 - Results of modelling showing zones of rock failure - weak roof
This initial simulation does not have coupled fluid pressure as part of the rock failure process as current models now have. As a result, it is probable that shear fractures form even further ahead of the face than indicated from this study. At Gordonstone, the outcomes of subsidence, stress measurement and microseismic data are all consistent with the results of the simulation.

**MEDIUM STRONG STRATA STUDIES - RESULTS FROM APPIN COLLIERY**

Appin Colliery is located in the Southern Coalfield of the Sydney Basin. The longwall panel at Appin is 200 m wide and extracts the 2.3 m thick Bulli Seam at a depth of about 500 m. The strata below the Bulli seam typically consists of interbedded strong sandstone, coal, carbonaceous material and interlaminated sandstone and shale. A study was made of longwall panel 28a to determine the nature of the fracturing to the underlying Wongawilli seam and to determine whether the fractures extended to the Tongarra seam. These seams are significant potential sources of goaf gas emissions. Within the panel numerous strike slip joint structures intersect the maingate particularly between 5 and 7 cut-throughs.

**Microseismic monitoring results**

At Appin Colliery 17 triaxial geophones were installed with nine geophones in a borehole drilled from the ground surface to the Bulli Seam and two perpendicular surface strings of four geophones each. The microseismic activity was monitored during August to November 1996 during which time there was 700 m of face retreat. Distinctive seismic events with low and high frequencies were observed.

The microseismic events locations in plan view, Fig. 6, indicate three broad areas of failure (i) cyclic failure of strata from mid face across to the tailgate (ii) reactivation of the strata under the pillars of the previous gateroad and (iii) activation of a strike dip structure in the maingate well outbye of the face position. All of the high frequency events are located in the fault structure zone and activation of the structure started far ahead, more than 300 m, of the face.

**Fig. 6 - Microseismic events location in plan view - medium strong roof**

The event locations in section, Fig. 7, show that the majority of fracturing (the low frequency events) extends to a height of about 50 to 70 m above the Bulli seam and to a depth of 80 to 90 m into the floor, often extending down to the Tongarra
They tend to occur up to 30 - 50 m ahead of the face in cyclic pattern. The events around the previous gateroad pillars may also be up to 300 metres away from the face position.

![Fig. 7 - Microseismic events locations in cross section](image)

**Numerical modelling results**

The Appin longwall caving model, Fig. 8, has been developed using the geological and geotechnical data collected from the mine. The section considered for detailed investigations extends from 50 m below the seam to 150 m above the working seam. The model simulations also included coupled fluid pressure as part of the rock failure process. The results of the simulations are presented in Fig. 9. Key features of the results include:

- Cyclic fracturing through to the base of the Wongawilli seam and occasional permeability increase down to the Tongarra seam.
- Bedding plane shear in the Stanwell park claystone unit approximately 100 m above the Bulli seam and extending 50 - 100 m in front of the working face.

The model predictions are consistent with microseismic monitoring measurements with respect to the cyclic loading but do not represent the extraordinary three dimensional nature of failure in this environment.

**Face monitoring**

In addition to the above studies, face monitoring investigations were conducted at Appin Colliery. The new convergence monitoring system was used in conjunction with support manufacturer's monitoring packages for the monitoring of both leg pressures and convergence. The results agree with those from a conventional potted wire convergence system also used on the face. With the additional information on canopy angle, it is possible to determine whether there is tip loading or goaf loading occurring. The face monitoring data are currently being analysed for comparison with the microseismic and modelling results. Preliminary analyses show a correlation between convergence rate, microseismic events and subsequent gas emissions.
NEW INSIGHTS INTO LONGWALL CAVING

As described earlier, most traditional studies while recognising that abutment loads can be detected significant distances away from the longwall, still predict that the maximum abutment loads occur close to the longwall excavation. They further predict that these abutments are 4 to 6 times the overburden pressure. Most studies also predict either explicitly or implicitly that the failure mechanism is tensile. This tensile mechanism is caused by an indirect tensile stress due to an essentially unconfined large abutment load close to the face.
In contrast, the vertical stress profile, Fig. 10, developed from the simulation, shows that the maximum abutment load is only twice the overburden stress and occurs 10 metres ahead of the face. This difference may be from two main causes:

- Firstly, many Australian mines are in a high horizontal stress regime with the major principal stress being horizontal, and about 2.5 times the overburden stress and typically designed to be parallel to the gateroads. The horizontal relief into the goaf is quite significant and through lateral relaxation will have a decreasing influence on the vertical stress.

Fig. 9 - Results of modelling showing zones of rock failure - medium strong roof

Fig. 10 - Vertical abutment stress relative to the goaf edge developed in the model
• Secondly, bedding plane shear and shear through intact rock will result in a reduction in the load carrying capacity of the rock adjacent to the longwall zone. This will effectively transfer the abutment peak away from the longwall and reduce its magnitude.

The microseismic events at Gordonstone have shown that the initial failure occurs 50-70m ahead of the face and that the dominant failure mechanism of all events is a shear failure with a failure plane orientated at an average angle of 50° to the horizontal thrusting up into the goaf direction. This failure is further validated by piezometric evidence. Our modelling has also indicated that the initial failure occurs a substantial distance ahead of the face with the dominant failure mechanism being shear through intact rock with some bedding plane shear.

The main difference between the model and actual field microseismic measurements are not the modes of failure but the predicted distance ahead of the face where failure occurred, being 15 m for the model and a mean of 30 - 40 m for the field measurements. The roof sequences at Gordonstone have a high moisture content which was not modelled as part of the failure criteria. It is postulated that pore water pressure has an influence on the intact rock failure by reducing the effective stress. This will cause failure to be initiated at lower stress levels and may explain the differences between the model and field measurements.

At Appin, the failure mechanisms recorded were of three main types:

1. A cyclic failure in the Tailgate side of the face with an interval of slightly more than 100 m. These events occurred about 50-60 m ahead of the face and with the majority from 80 m above the seam to 100 m below the seam. These mechanism was expected, although the depth below the seam was surprising. This depth demonstrated consistent breakage of the Tongarra seam for gas release.

2. Failure along a joint-shear in the Maingate/blockside of the face. This joint had been mapped as a minor feature underground. Failure along this joint commenced while the face was still 450 m inbye of it and this mechanism controlled the stress relief along the entire maingate side. It is suspected that this mechanism was a major influence on the reduction of gas release to 25% less than normal total gas make expected. This mechanism is outside all conventional longwall geomechanics theory and was a surprise to both the research team and site staff.

3. Failure of the gateroad pillars between the previous longwalls 26 and 27. Conventional pillar theory revolves around empirical coal strength formula. However the majority of the failure observed in this case occurred from in the surrounding strata, predominantly in the floor below the pillar up to a depth of 130m below the seam. Again this is outside conventional longwall geomechanics theory.

The results from the current project will either demonstrate that shear failure is the dominant failure ahead of a longwall face or otherwise demonstrate that this type of mechanism is limited to certain geotechnical regimes.

CONCLUSIONS

The power of combining accurate microseismic monitoring and detailed numerical simulation has been demonstrated in this study. It has shown that, in the circumstances at Gordonstone, the traditional models of tensile failure mechanisms and abutment loads of 4-6 times the overburden pressure are not correct. The effect of shear failure, reduction of horizontal stresses and perhaps pore fluid pressure has resulted in a much lower abutment load which peaks further away from the longwall face than traditional models indicate. This has a major implication in understanding longwall geomechanics generally and will influence issues of face control, gateroad abutments and fluid flow (both gas and water).

At Appin, the results have been even more dramatic and in contradiction to traditional longwall geomechanics theory. The essence of the 3D nature of failure in medium strength environments has been depicted, especially underlying the effect of minor structure in influencing failure mechanisms and questioning the relevance of conventional pillar design theory which relies principally on empirical coal strength formula. The 3D nature and effect of structure has also been demonstrated at North Goonyella, although that site is still being monitored and results are preliminary at this stage. The value of detailed site studies, especially structural analyses, has been demonstrated at all of the sites.
The effectiveness of the combination of technologies has been the most potent outcome of the work to date. Validation of results through 2, 3, 4 and even 5 independent technologies increases confidence in the findings and dismisses the academic controversies which tend to cloud important new work such as this. This confidence assists implementation into minesites and graphically demonstrates the narrow vision of past assessment methods. The future requires a better understanding of the 3D nature of some of these mechanisms and the development of effective numerical modelling that can translate results into studies of mining alternatives. Also required is the further development of the microseismic monitoring systems. Current analyses, although accurate, are slow and tedious.

Finally, although not discussed throughout the paper, the safety implications of better defining longwall mechanics demands a comment. A better understanding of longwall geomechanics has profound safety implications. Issues such as face control, wind blasts, gateroad support and high level gas emissions can all be better addressed with the knowledge that this project will provide.

ACKNOWLEDGMENTS

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INTRODUCTION

Coal exports are forecast to grow strongly, so that coal is likely to remain very important to Australia for the foreseeable future. However there is no sign that prices or profitability will rise of their own accord. Therefore the intense squeeze on costs will continue for the foreseeable future. This will mean lower manning, more outsourcing and probably an increasing rate of company attrition. As well, the political pressure from the greenhouse debate is only going to intensify and the pressure on the Federal Government after Kyoto is likely to be irresistible.

The next one to two years therefore will see the industry caught in a nutcracker of intense political pressure on the one hand to help the country meet its political obligations, while on the other there is the relentless pressure of zero profit margins. Answers will be demanded that require research at the same time that cost pressures are reducing companies' ability to fund it. It is timely therefore to consider what it is that the industry is getting from its research levy and to see whether it really is making a difference.

AUSTRALIAN COAL ASSOCIATION RESEARCH PROGRAM (ACARP)

In December, 1992 the Australian Coal Association took over the program of coal related research, which had been run under the National Energy Research Development and Demonstration Program (NERDDP). Under the former program approximately A$167 million dollars was spent from 1978 to 1992, with expenditure largely funded from a levy of 5c/tonne on black coal production. The Australian Coal Association now fully directs the research program, called ACARP and since June 1993 has taken over voluntary collection of the coal research levy.

ACARP's charter has recently been extended for another two year period and will now run until at least the year 2000. This good news is a significant vote of confidence in the program from Australian coal producers.

So far under ACARP, approximately A$41 million dollars of levy funds have been committed to research projects with a total value of about A$100 million. The additional funds come from ACARP cooperation with other funders of research such as the BHP Special Research Program, the NSW State Energy Research Fund, manufacturers of mining equipment and individual coal mining companies which support specific projects over and above their research levy contribution. The levy of 5c/tonne on black coal production, is currently generating about A$ 9.9 million per year.

ACARP has committed funding to 294 projects so far. Of these about 105 have concluded, 155 are in various stages of completion and the rest are under contract negotiation prior to starting. An additional 56 projects have just been selected to which a further $8.6 million will be committed when the necessary contracts have been finalised.

ACARP ANALYSIS

About two thirds of ACARP's research effort is on mining related activities reflecting the fact that funds are drawn from coal producers which have to remain world competitive in selling price and at the same time cope with increasing production difficulties in terms of deepening pits and increasing environmental pressures. Table I shows the distribution of project findings during the period 1992 - 1996.
Table 1 - ACARP 1992-96

<table>
<thead>
<tr>
<th>CATEGORY</th>
<th>No of PROJECTS</th>
<th>ACARP $ million</th>
<th>OTHER $ million</th>
<th>TOTAL $ million</th>
<th>LEVERAGE</th>
</tr>
</thead>
<tbody>
<tr>
<td>Underground</td>
<td>139</td>
<td>16.6</td>
<td>26.9</td>
<td>43.5</td>
<td>2.6X</td>
</tr>
<tr>
<td>Open Cut</td>
<td>60</td>
<td>11.7</td>
<td>12.4</td>
<td>24.1</td>
<td>2.1X</td>
</tr>
<tr>
<td>Coal Preparation</td>
<td>54</td>
<td>7.2</td>
<td>8.0</td>
<td>15.2</td>
<td>2.1X</td>
</tr>
<tr>
<td>Coal Utilisation</td>
<td>41</td>
<td>5.8</td>
<td>13.8</td>
<td>19.6</td>
<td>3.4X</td>
</tr>
<tr>
<td>TOTAL</td>
<td>294</td>
<td>41.3</td>
<td>61.1</td>
<td>102.4</td>
<td>2.5X</td>
</tr>
</tbody>
</table>

This table demonstrates that a useful degree of leveraging continues to be achieved on project funding. In the round just completed, ACARP’s $8.6 million allowed entry into nearly $19 million worth of projects.

**THE ACARP VISION**

In a recent review of its vision for ACARP the joint Research Committee looked at where ACARP currently is and where it would like to be.

**Where is ACARP**

- ACARP is an effective program of research with runs on the board;

  Around $20 million per year of projects are funded;

- Second most influential coal research funder in Australia after BHP;

- Strongly networked with research providers;

- A strong inter-company technical communication forum;

- An effective industry response capability

- Provides much needed commercialisation support;

- International links are strengthening;

- Still flexible and developing;

- Constrained by the need for short term results;

  CEOs not fully convinced;

- Minesite support about 50/50 (could be 80:20). Coal Prep 80:20 already; and

- More integration of different program segments is needed;

**What is needed**

- A clear vision for the future;

- A clear strategy

- Sufficient time;
The current issues and problems clarified;

The right program formulated / the right projects selected;

Superb communication, marketing;

Support from all stakeholders (to be recognised as essential to the industry);

A credible cost - benefit analysis mechanism established:

Collaborative funding maintained or strengthened;

The research community focussed to the industry’s best advantage;

Efficient paths to commercialisation, implementation in place;

- Improved alliances with manufacturers ( domestic and overseas

**MAKING A DIFFERENCE**

If we achieved this challenging agenda how would this make a difference to our stakeholders? Well let us look at some examples of what ACARP is doing.

**Safety**

An important area in which ACARP is definitely having an effect is in safety. Significant work is being supported in six categories as shown in Table 2.

**Table 2 - Major Incident Safety**

<table>
<thead>
<tr>
<th></th>
<th>ACARP $</th>
<th>OTHER $</th>
<th>TOTAL $</th>
</tr>
</thead>
<tbody>
<tr>
<td>23 Proj (i) Strata control</td>
<td>2463</td>
<td>3418</td>
<td>5881</td>
</tr>
<tr>
<td>(ii) In-seam drilling</td>
<td>2310</td>
<td>3702</td>
<td>6012</td>
</tr>
<tr>
<td>18 Proj (iii) Detection and prevention of fires and explosions</td>
<td>2412</td>
<td>2955</td>
<td>5367</td>
</tr>
<tr>
<td>15 Proj (iv) Gas monitoring, drainage and control</td>
<td>1288</td>
<td>2474</td>
<td>3762</td>
</tr>
<tr>
<td>10 Proj (v) Outbursting</td>
<td>969</td>
<td>1105</td>
<td>2074</td>
</tr>
<tr>
<td>5 Proj (vi) Escape and rescue</td>
<td>1185</td>
<td>708</td>
<td>1893</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>10627</td>
<td>14362</td>
<td>24989</td>
</tr>
</tbody>
</table>

**Strata control**

As you can see from the table the biggest area of effort has been in strata control. There is no doubt that the reason for this strong ACARP support is because strata control is not only a safety issue but a key production issue as well. The safety aspects are well demonstrated of course by the Joint Coal Board funded project at the University of NSW on improving pillar design. This work can clearly be shown to have saved lives and reduced serious injuries. ACARP is funding the UNSW group to extend this work into understanding the support capacity of irregularly shaped pillars and into soft strata environments and also to improve the design and performance of timber chocks.

ACARP has put considerable effort into trying to understand and implement pretensioning of roof bolts because of the promising results of early work in this area. Seminal work by ACIRL and Ripu Lama has been followed up by Barrett Fuller and Partners in their work on the Flexibolt and by Strata Engineering. Results to date are very encouraging. The UNSW is also looking, in conjunction with ANI Arnall, at the effect of pretensioning as a means of improving rib support.
Another consistent strand within ACARP funded work has been to improve monitoring techniques and instruments for strata control since this is a key to early warning of impending trouble. Work with the CSIRO, Strata Control Technologies and Mincad has been very successful in this regard. The safety benefits of some of this work have recently been well demonstrated by Pacific Power in the successful management of their windblast problem. Early warning was the key to removing personnel from danger before windblast events.

ACARP recently funded an exchange of data with South Africa which has some very graphic examples of the danger to life and limb posed by major strata collapses and their experience is a clear reminder of why ACARP supports efforts by every significant strata control group in the country to improve the theory and practice of strata control.

In-seam drilling

The main technological developments in in-seam drilling since the commencement of ACARP have been made by mine operators and contractors, with assistance from suppliers, to alleviate current problems. ACARP funded research has resulted in prototypes which have either been completed or will be completed over the next two years and which will take in-seam drilling technology to the next generation. This is a very well thought out and coordinated program aimed to lift in-seam drilling to a new level.

The value of this work can be estimated by considering whether any South Coast mine which is unable to demonstrate adequate gas drainage through a well conducted in-seam drilling program, is likely to be allowed by the inspectorate to remain open. It has also been of major value to Dartbrook in helping that pit cope with its high carbon dioxide levels and the Central Queensland mines now starting to cope with hydrogen sulphide.

By monitoring and coordinating individual projects and acting as the technical transfer link between researchers, pits, drillers and equipment suppliers this task force has been part of a very significant upgrading in the industry’s capabilities in an area which threatens the ability of many pits to continue mining. Those operations which are actively participating in this program are deriving very considerable value from it.

Some achievements are:

* Development and demonstration of the rotary drill rig monitor by BHP Research for detection of structures while drilling. (Awaiting commercialisation).

* Development and IS approval of a 486 computer by BHP Research, for safe data collection underground. Combined with the M&G high speed modem, which has also received IS approval, this is a very important breakthrough in data collection and analysis. (Modem on sale, IS computer awaiting commercialisation.)

* The AGA consortium has developed technology to enable monitoring of drilling parameters behind the bit, thus facilitating future detection of potential outburst structures while drilling long holes. (Undergoing surface testing prior to underground demonstration)

* A borehole pressurisation system has been developed by AGA which will facilitate future geophysical logging of holes. In conjunction with this, Sigra has developed a device for analysing the virgin gas content of drill cuttings while drilling, for a better assessment of outburst potential. (Undergoing surface testing prior to underground demonstration)

* Development of radar and radiometric tools by the CMTE for use during long hole drilling to detect proximity of the bit to roof and floor and for detection of adjacent structures. These probes have been partly funded by ACARP. (Prototypes still being tested)

* Development by AGA (privately funded) of an inexpensive electronic ‘measure-while-drill’ survey tool for use initially in rotary and later in long holes either as a component of other tools or as a separate item. This tool should be available in late 1997.
• Initial trials with the CMTE developed water-jet assisted drill are very encouraging for drilling straight gas drainage holes which go where they are aimed. (Trials continuing).

Unfortunately there is not time to go into the details of what is being developed but only to say that we expect to have the capacity shortly, to drill holes more rapidly with an accuracy that will guarantee safe drainage and to have the means of gathering information both during and after the drilling which will accurately locate and define any structures which have dangerous outburst potential.

Gas monitoring, drainage and control and outbursting

Although these are categorised separately of course they are all of a piece with in-seam drilling as part of improving the industry's ability to safely control gas emissions of whatever type. Two projects in the Outbursting category for example, could just as easily be put into in-seam drilling. One is a project to pressurise boreholes. This device if successful will allow a wider range of geophysical tools to employed for detecting outburst structures. This class of tools requires a fluid filled hole to be effective. Another is a project to develop a new geophysical tool for the detection of outburst structures in boreholes.

As with strata control, improved monitoring is a key area of importance, for early warning of a dangerous gas build up or changes in the make up of gases being emitted which could signal danger. Many facets of monitoring are being investigated. Attempts are being made to improve real time monitoring in working areas, during drilling, in goafs and in return areas. A prototype of a new meter for measuring gas make from drainage holes has been developed and is awaiting testing. And in response to the Moura Inquiry two different prototype valves for shutting off drainage gas in the event of an explosion have been developed and are also awaiting trial. Hopes are high that these will help prevent the secondary explosions which are the most damaging events after an initial incident.

Detection and prevention of fires and explosions

There are four main strands to ACARP work in this area.

The first is electrical safety where we have five projects looking at ways of preventing dangerous ignitions particularly from high voltage electrical equipment underground.

The second is on improving the design and efficiency of dust and water explosion barriers. This work is to be added to in the latest ACARP round by further efforts to improve seals and stoppings. In this regard one of the highlights of the recent ACARP funded exchange of data with South Africa was the revelation of a new and apparently highly effective stone dust barrier, recently developed there.

The third and probably most costly in dollar terms was support for the recent inertisation trials using the GAG jet engine and the Thomlinson Boiler. Both these trials were very successful, leading to the purchase by the Queensland Government of a GAG inertisation system for their State and further development of low flow systems.

The fourth is the understanding and control of spontaneous combustion, which clearly is still a major issue with the industry. From ACARP's point of view it is not only a danger to underground mining but also an environmental problem for open cut mining and a possible hazard in transport and handling. The projects in these other areas reinforce the work done for underground miners. For example, an ACARP project which is attempting to control spontaneous combustion in an open cut spoil pile by the injection of a fly ash slurry, could well be applicable to creating a grout curtain in an underground situation.

The fifth area, where there is only one project, is to develop an improved method of testing and classifying the flammability of conveyor belting, since underground belt fires are becoming an increasing concern. Related to this, but not categorised as a Safety project, is a device developed under ACARP by Vipac (and recently commercialised by them) to rapidly detect, faulty idler bearing rollers prior to failure. The failure of these rollers is a major cause of belt fires.
Escape and rescue

In spite of the very high cost involved, a number of mines have gone ahead with the purchase of oxygen self rescuing equipment. The South African experience with oxygen self rescuers, revealed on the recent exchange visit to South Africa, has been very enlightening in this regard. Not only is it clear that some equipment is better than others, it is also clear that introduction of oxygen self rescuers needs to be in conjunction with a system of monitoring the devices to ensure that they are always in satisfactory working order. This realisation is currently being digested by the state rescue services and will certainly lead to further ACARP work to help develop the necessary capability.

ACARP inherited the Numbat from NERDDP and has continued work to both improve its capability and to use it as a test bed for a possible Mine Rescue Vehicle. A very imaginative proposal has been put forward to develop a mine rescue vehicle based on the ANI Ruwolt man transport vehicle. It is proposed that the vehicle have a self contained air supply for both personnel, and the engine, so that it is isolated from the mine atmosphere. It would carry the Numbat sensing package. A proof of concept project will be undertaken to test the feasibility of such a design.

BENEFITS FROM ACARP

Passive benefits

Levy payers benefit both passively and actively from what ACARP does.

Some benefits which the industry receives accrue to the coal industry generally. They are real but difficult to quantify in value because they tend to be preventative in nature.

A good example is what happens after there is a serious incident in a coal mine in which either multiple lives were lost or a disaster was narrowly avoided. The subsequent inquiry plus union reaction almost always leads to massive pressure on the industry and the regulating authorities to make a significant response. There is a strong tendency for draconian legislation to be introduced which is ultimately unproductive.

That the counter productive effects of this sort of knee jerk reaction can last for decades is well illustrated by the banning of the use of aluminium alloys in British and Australian underground collieries on the basis of what in hind sight looks like an extraordinary reaction to an incident in a British coal mine in 1962. In spite of how odd this decision now appears and the scientific and metallurgical progress made since it was imposed, it may never be possible to get the ban lifted in Australia. A study being undertaken by ACARP which documents the background to the decision and more importantly the lack of incidents attributable to the use of aluminium alloys in US coal mines, however does offer some hope.

More importantly, as demonstrated in the aftermath of the recent Moura Inquiry, the existence of ACARP funding and the ACARP committee structure provides an effective and above all a credible response mechanism to the technical questions raised by the Inquiry. There is no doubt that this response played a part in producing a productive outcome for the industry rather than a purely negative one.

Levy payers who are not actively involved in the Inquiry response or in ACARP, still benefit if the industry is protected from inappropriate legislation. Other examples of where the industry has an improved defence from negative judgements because of ACARP assistance, are related to environmental or community problems e.g., mine site rehabilitation or reduced pollution from coal utilisation

In these cases a bad image created at one place can easily produce problems for every operator and it is politically essential for the coal industry to be seen to be taking effective action to combat the problems.

Another passive benefit is the way ACARP helps to keep public research at the universities and the CSIRO focused on industry problems. Over time the benefit of this work accrues to all operators in the improved understanding of problems such as pillar design or rock mechanics. Additionally, research infrastructure is maintained which is available when a pit needs help and an improved flow of better trained graduates results.
Active benefits

Clearly the operators who benefit most from ACARP are those who actively participate by offering staff for selection committees and project monitors and their mines as demonstration sites. When a research project relieves a serious problem at a particular mine site and at the same time demonstrates a technique which is useful industry wide, the benefits are immediate at the demonstration site and valuable over time for the rest of the industry.

The recent ‘Coal Loss’ project which demonstrated how to reduce coal losses in open pit mines is an excellent example of this. The demonstration sites gained additional coal worth several million dollars a year while industry wide there are now demonstrated techniques available for pits with similar problems.

Here are some examples where pits, through participation in ACARP have recorded significant gains:

- Westcliff (NERDDP)
- Angus Place, Ellalong
- Gordonstone, Tower, Teralba, West Wallsend
- Appin, Tower
- Tahmoor, Dartbrook,
- South Bulga, Appin, North Goonyella
- Tahmoor
- Wyee, Cooranbong, Tahmoor
- Gordonstone
- Howick, Warkworth
- Electricity generation from mine gas drainage
- Flexibolt (significantly improved roof support)
- Pre-tensioning of roof bolts (as above)
- Improved gas drainage/drilling for greater effectiveness at lower cost
- Real time gas monitoring
- Improved understanding of ground behaviour around longwall faces
- Remote control outburst driveage
- Use of the Coal Auger for drilling cut throughs
- Improved timber chock constructions
- Reduced coal loss for higher mine yield

Many innovations have come from this work which will benefit the industry, but the companies that get the most benefit and the first use of these and will be those which actively participated in their development through ACARP.

Other than just providing specific solutions ACARP has a role in encouraging pits to develop a culture of innovation. Innovation allows an operation to be in charge of events rather than just reacting to a harsh external environment such as worsening mining conditions (do they ever get better?), falling prices, loss of markets, customer dissatisfaction, inspecional edicts, environmental pressures, company takeovers, rising workers’ compensation premiums, etc, etc.

These are just a few examples of the gains pits have made through active participation at mine sites, mostly from the underground sector.

For open cut mining the major effort has been to improve overburden removal efficiency. In Australia this effectively means trying to improve dragline efficiency. Mainly through company efforts, but definitely assisted by ACARP, this has been a notable success story. There have been major gains in efficiency. The levy payers have received full value from the research which has made a recognisable difference at their individual pits.

Some of you may have seen the recent report by Tasman Asia Pacific which was commissioned by Rio Tinto entitled “The Scope for Productivity Improvement in Australia’s Open Cut Black Coal Industry” which made very unfavourable comparisons between USA coal mine productivity and Australia. The one bright spot where we are way ahead is in the productivity of Queensland draglines. I have no doubt that ACARP is partly responsible for this.

Unfortunately time precludes further examples but allow me to say that one of the key indicators of the success of the program is that at last count, 68 mine sites were actively hosting ACARP projects. Under the previous government regime, project selection tended to be researcher driven. Under ACARP it is operator driven and increasingly the pits themselves are becoming the research laboratories. We believe that this is a sure sign that the program is making a difference.
Communication

ACARP provides a communication network through committee meetings, newsletters and various types of workshops and seminars and will soon be augmenting these efforts through the Internet and by publishing our total research output on CD-ROM so that it is rapidly searchable for data of relevance to a levy payer’s problem. While all new knowledge has value, its usefulness is vastly increased by being easily accessible and its true value is often only apparent when its relevance to a current problem is recognised. Superb communication is vital to ensure that the full value of research is realised.

THE ENVIRONMENT

Maintenance of a healthy coal mining industry for the community means above all means the maintenance of high paying jobs.

However those not involved in coal mining object, at times strenuously, to the by-products of mining:

- visual pollution;
- nuisance noise and dust;
- increased heavy truck traffic near ports;
- subsidence over long wall mines affecting ground and surface water, buildings and roads, scenic rock formations etc;
- interference with alternative rural land uses; and
- salination of streams.

Whether it realises it or not, the community of course gains significantly from the provision of cheap electricity based on coal. However this is not without concerns in regard to:

- NOx, SOx and particulate emissions;
- Greenhouse emissions causing climate change;
- Effluent pollution of streams e.g., selenium poisoning; and
- Disposal of fly ash.

It is pretty clear then that if the community can comfortably enjoy the economic and life style benefits that coal mining undoubtedly brings, while having the side effects reduced or maintained within tolerable limits, then that would make a considerable difference to how the coal mining industry was perceived by the community.

It is difficult to see how the technical issues related to community concerns could be tackled in a more cost effective way than through a system such as ACARP.

The following are some of the environmental projects which ACARP has supported which are making a difference to the mining industry:

- Projects to improve the stability and drainage of spoil piles left by open cut coal mining;
- Projects to improve revegetation of spoil piles with native grasses, shrubs and trees;
• Projects to improve the quality of water in final voids;
• Projects to reduce noise and dust from open cut mining; and
• Projects to reduce the effects of mine subsidence.

MARKETING OF AUSTRALIAN COALS

Coal preparation

Some of the best results which ACARP has achieved have been in coal preparation research. This success makes a difference for the levy payer through:

• improved recoveries leading to higher mine yields;
• reduced risk of incurring penalties for being outside specifications for ash, and moisture;
• lower transportation charges; and
• reduced handling problems.

The customer also benefits through getting a consistent and improved product with lower ash, lower moisture, reduced impurities and better blends which improve such properties as coke strength, combustion efficiency and coal handling.

Coal utilisation

Work in this category is by definition highly customer oriented. The aim is to make Australian Coals work better, more cleanly and efficiently in whatever application they are used. If this happens the customer benefits and also the environment. If the environment benefits the adverse community pressure on the use of coal at all levels is reduced, whether it be in a prefecture in Japan, a local community in Australia worried about air toxics or the Federal Government trying to secure a slightly less unfavourable outcome for Australia from greenhouse treaty negotiations.

Much good work has been done in many areas, such as:

1. improved combustion;
2. reduced NOx emissions;
3. reduced particulate emissions;
4. improved measurement and control of trace elements; and
5. improved coal grinding.

CONCLUSION

A research program such as ACARP not only has to add value for all its stakeholders, it must be clearly seen to be doing so.

Its primary focus must be on meeting the needs of the coal companies who pay the levy which funds it.

They get value by
• having their operational problems relieved by site based research;
• by keeping their key staff abreast of the latest technology;
• by the cross fertilisation of good ideas and by exposure to best practice;
• by being able to respond to community concerns in a systematic, coordinated and visible way;
• by being able to respond to government concerns in systematic, coordinated and politically effective way;
• by the maintenance of a core of talent in research establishments who can respond as needed where problems arise
• by helping attract and train good students who will enter the coal industry preferentially
• by helping to develop and sustain a local service and manufacturing response capability which can provide customised output for the local industry;
• by helping to maintain the public image, credibility and prestige of the coal industry.

This last point is more important than most people think. When coal companies think of competition, they think of other coal mines, either in Australia or overseas, which are competing for sales in Japan or Taiwan or Europe. They tend not to give full recognition to the primary competition within Australia for the very right to exist.

Coal companies compete for access to land against rural interests and the tendency to declare an increasing portion of the country national parks. They compete for finance and the best university graduates and they compete for public favour in the way business taxation is levied.

Consider these recent examples
• If a prospective coal mine planning to export A$100 million worth of coal a year, is competing with a small vineyard likely to produce wine worth less than one million dollars a year, which will have the most public sympathy?
• If the government is planning to take away a tax rebate on diesel fuel and it has to choose, which is it more likely to take it from, the grain growing industry or the mining industry?
• If you are a brilliant high school student what will you tend to study at university: law, accounting, medicine or mining engineering?

Public perception and political good will play a major role in decisions such as these and having an active research program which can ameliorate or prevent problems is not only good business sense, it can be very good PR. Coal mining is Australia’s premier export industry. It is vital to the economic health of the country yet it has low public esteem, minimal political support and attracts few of the top graduates. Anything which adds to its prestige adds value.

In meeting the needs of the levy payers the research program is automatically adding value for the other stakeholders because coal mining does not occur in isolation. Coal mining will only be funded if its financiers believe they will get value; its products will only be saleable if its customers get value; it will only be the preferred land use option if the community feels that this use adds more value than alternatives and it will only get from its work force a profitable response if they feel it adds value to their lives. In all these interactions, a well managed and targeted research program can be a cost effective means of value addition for all the stakeholders.

ACARP has all these attributes and is certainly making a difference.
APPENDICES

Underground category

Major incident safety

Table 3 - Gas monitoring, drainage and control

<table>
<thead>
<tr>
<th>Code</th>
<th>Description</th>
<th>ACARP</th>
<th>OTHER</th>
<th>TOTAL</th>
</tr>
</thead>
<tbody>
<tr>
<td>3029</td>
<td>Improved technology for maintaining hole integrity during gas drainage</td>
<td>25,000</td>
<td>25,000</td>
<td>50,000</td>
</tr>
<tr>
<td>3030</td>
<td>Real time monitoring of gas emissions</td>
<td>150,000</td>
<td>197,000</td>
<td>347,000</td>
</tr>
<tr>
<td>3034</td>
<td>Development of a general purpose hydrogen monitor</td>
<td>80,000</td>
<td>0</td>
<td>80,000</td>
</tr>
<tr>
<td>3076</td>
<td>Real time return gas monitoring for outburst and gas drainage assessment</td>
<td>91,163</td>
<td>171,164</td>
<td>262,327</td>
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<tr>
<td>3077</td>
<td>Gas detection technique to continuously monitor gas in drill fluid</td>
<td>45,000</td>
<td>15,000</td>
<td>60,000</td>
</tr>
<tr>
<td>4033</td>
<td>Stimulation of gas make from horiz in-seam drain holes by hydraulic fracturing</td>
<td>122,535</td>
<td>130,096</td>
<td>252,631</td>
</tr>
<tr>
<td>4040</td>
<td>Development of a hydrogen monitor for use in coal mines</td>
<td>139,253</td>
<td>220,000</td>
<td>359,253</td>
</tr>
<tr>
<td>5030</td>
<td>Development of a gas flow drainage meter</td>
<td>99,000</td>
<td>0</td>
<td>99,000</td>
</tr>
<tr>
<td>5036</td>
<td>Hydrofracture modelling to assess potential improvements to mine gas drainage</td>
<td>24,408</td>
<td>24,408</td>
<td>48,816</td>
</tr>
<tr>
<td>6020</td>
<td>Mine gas control</td>
<td>25,000</td>
<td>975,000</td>
<td>1,000,000</td>
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<tr>
<td>6021</td>
<td>Automatic shut-down valve for UG gas drainage lines (under vacuum)</td>
<td>40,000</td>
<td>15,000</td>
<td>55,000</td>
</tr>
<tr>
<td>6022</td>
<td>Automatic shut-down valve for UG gas drainage lines (positive pressure)</td>
<td>93,250</td>
<td>20,000</td>
<td>113,250</td>
</tr>
<tr>
<td>6031</td>
<td>Maximising coal production in the presence of H,S seam gas</td>
<td>162,000</td>
<td>591,000</td>
<td>753,000</td>
</tr>
<tr>
<td>13</td>
<td>TOTAL</td>
<td>1,096,609</td>
<td>2,383,668</td>
<td>3,480,277</td>
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</table>

Table 4 - In-seam drilling

<table>
<thead>
<tr>
<th>Code</th>
<th>Description</th>
<th>Cost</th>
<th>Cost</th>
<th>Cost</th>
</tr>
</thead>
<tbody>
<tr>
<td>3070</td>
<td>In-seam drilling and bit location system</td>
<td>235,000</td>
<td>200,000</td>
<td>435,000</td>
</tr>
<tr>
<td>3071</td>
<td>Calliper probe for logging in-seam bore holes</td>
<td>41,680</td>
<td>28,000</td>
<td>69,680</td>
</tr>
<tr>
<td>3072</td>
<td>Borehole pressurisation system</td>
<td>135,290</td>
<td>40,000</td>
<td>175,290</td>
</tr>
<tr>
<td>3073</td>
<td>Bit, torque, load and RPM sensors</td>
<td>59,620</td>
<td>20,000</td>
<td>79,620</td>
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<tr>
<td>3074</td>
<td>Standards for in-seam drilling equipment</td>
<td>11,130</td>
<td>0</td>
<td>11,130</td>
</tr>
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<td>3075</td>
<td>In-seam drilling project coordinator</td>
<td>40,000</td>
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<td>40,000</td>
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<tr>
<td>4035</td>
<td>Co-ordination of in-seam drilling research</td>
<td>44,000</td>
<td>0</td>
<td>44,000</td>
</tr>
<tr>
<td>4036</td>
<td>In-seam drill monitoring and bit location system stage 2</td>
<td>250,000</td>
<td>230,000</td>
<td>480,000</td>
</tr>
<tr>
<td>4037</td>
<td>Sensing and logging for in-seam boreholes</td>
<td>210,000</td>
<td>1,337,000</td>
<td>1,547,000</td>
</tr>
<tr>
<td>4038</td>
<td>Electronics for bit torque, load and rpm sensors (bitor electronics)</td>
<td>20,000</td>
<td>0</td>
<td>20,000</td>
</tr>
<tr>
<td>4039</td>
<td>Testing of drill rod joints for long hole drilling</td>
<td>90,000</td>
<td>20,000</td>
<td>110,000</td>
</tr>
<tr>
<td>5027</td>
<td>Co-ordination of in-seam drilling research</td>
<td>44,000</td>
<td>0</td>
<td>44,000</td>
</tr>
<tr>
<td>5028</td>
<td>Water jet assisted drilling</td>
<td>195,000</td>
<td>950,500</td>
<td>1,145,500</td>
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<tr>
<td>5029</td>
<td>Development of a new borehole survey tool</td>
<td>234,100</td>
<td>54,700</td>
<td>288,800</td>
</tr>
<tr>
<td>6027</td>
<td>Co-ordination of in-seam drilling research</td>
<td>44,000</td>
<td>0</td>
<td>44,000</td>
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<tr>
<td>6028</td>
<td>Longhole water jet assisted drilling</td>
<td>240,000</td>
<td>484,000</td>
<td>724,000</td>
</tr>
<tr>
<td>Proj</td>
<td>Development of an intrinsically safe drill monitoring system Stage 3</td>
<td>75,000</td>
<td>55,000</td>
<td>130,000</td>
</tr>
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<td>------</td>
<td>---------------------------------------------------------------</td>
<td>--------</td>
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<td>--------</td>
</tr>
<tr>
<td>17</td>
<td>TOTAL</td>
<td>1,968,820</td>
<td>3,419,200</td>
<td>5,388,020</td>
</tr>
</tbody>
</table>

Table 5 - Outbursting

<table>
<thead>
<tr>
<th>Proj</th>
<th>Improved remote control and monitoring of outburst mining equipment</th>
<th>170,000</th>
<th>248,000</th>
<th>418,000</th>
</tr>
</thead>
<tbody>
<tr>
<td>3035</td>
<td>Workshop on management and control of outbursts in underground coal mines</td>
<td>25,000</td>
<td>20,000</td>
<td>45,000</td>
</tr>
<tr>
<td>3079</td>
<td>Outbursting scoping study</td>
<td>50,000</td>
<td>0</td>
<td>50,000</td>
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<tr>
<td>4034</td>
<td>Development of a borehole pressurisation tool for outburst assessment</td>
<td>156,850</td>
<td>53,150</td>
<td>210,000</td>
</tr>
<tr>
<td>5035</td>
<td>Prediction of outbursts using the occurrence of radon gas</td>
<td>79,500</td>
<td>78,000</td>
<td>157,500</td>
</tr>
<tr>
<td>5037</td>
<td>Degassing of methane and carbon dioxide</td>
<td>116,371</td>
<td>165,000</td>
<td>281,371</td>
</tr>
<tr>
<td>6023</td>
<td>Intercomparison of 'quick crush' techniques used to measure gas content of coal</td>
<td>76,250</td>
<td>41,200</td>
<td>117,450</td>
</tr>
<tr>
<td>6024</td>
<td>Modelling of outburst mechanisms</td>
<td>100,000</td>
<td>266,000</td>
<td>366,000</td>
</tr>
<tr>
<td>6025</td>
<td>Detection of gas emission events precursive to outbursting using seismic</td>
<td>65,000</td>
<td>123,250</td>
<td>188,250</td>
</tr>
<tr>
<td>6026</td>
<td>Bore hole dielectric probe to detect mylonite zones and other structures</td>
<td>130,000</td>
<td>110,000</td>
<td>240,000</td>
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<tr>
<td>10</td>
<td>TOTAL</td>
<td>968,971</td>
<td>1,104,600</td>
<td>2,073,571</td>
</tr>
</tbody>
</table>

Table 6 - Detection and prevention of fires and explosions

<table>
<thead>
<tr>
<th>Proj</th>
<th>Reduction in earth fault currents by improved earthing reactor use</th>
<th>54,000</th>
<th>0</th>
<th>54,000</th>
</tr>
</thead>
<tbody>
<tr>
<td>3022</td>
<td>High voltage cable test apparatus</td>
<td>35,000</td>
<td>0</td>
<td>35,000</td>
</tr>
<tr>
<td>3083</td>
<td>Design and efficiency of dust and water explosion barriers in modern Australian mines</td>
<td>65,410</td>
<td>0</td>
<td>65,410</td>
</tr>
<tr>
<td>4030</td>
<td>Research into the failure of 11kv plugs and adaptors used in underground coal mines</td>
<td>164,000</td>
<td>233,000</td>
<td>397,000</td>
</tr>
<tr>
<td>4031</td>
<td>Study of safety aspects of sheet metal IP55 enclosures in high fault level mines</td>
<td>118,000</td>
<td>18,000</td>
<td>136,000</td>
</tr>
<tr>
<td>5031</td>
<td>Development of better indicators for spontaneous combustion</td>
<td>166,747</td>
<td>143,920</td>
<td>310,667</td>
</tr>
<tr>
<td>5032</td>
<td>Further research into failure of 11KV plugs and adaptors in underground coal mines</td>
<td>75,000</td>
<td>0</td>
<td>75,000</td>
</tr>
<tr>
<td>5033</td>
<td>Improved flammability test methods for conveyor belting material</td>
<td>214,150</td>
<td>111,000</td>
<td>325,150</td>
</tr>
<tr>
<td>5430</td>
<td>Design and efficiency of dust and water explosion barriers in modern Australian mines</td>
<td>22,535</td>
<td>0</td>
<td>22,535</td>
</tr>
<tr>
<td>6001</td>
<td>Improved prediction of spontaneous combustion</td>
<td>172,000</td>
<td>176,144</td>
<td>348,144</td>
</tr>
<tr>
<td>6002</td>
<td>Demonstration of sealing and monitoring during low flow goaf inertisation</td>
<td>238,000</td>
<td>100,000</td>
<td>338,000</td>
</tr>
<tr>
<td>6018</td>
<td>Design and efficiency of dust and water explosion barriers Stage 2</td>
<td>161,525</td>
<td>0</td>
<td>161,525</td>
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<tr>
<td>6019</td>
<td>Demonstration &amp; evaluation of jet engine inertisation techniques</td>
<td>432,512</td>
<td>515,100</td>
<td>947,612</td>
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<tr>
<td>13</td>
<td>TOTAL</td>
<td>1,918,879</td>
<td>1,297,164</td>
<td>3,216,043</td>
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COAL98 Conference Wollongong 18 - 20 February 1998
<table>
<thead>
<tr>
<th>Code</th>
<th>Project Description</th>
<th>Budget 1997</th>
<th>Budget 1998</th>
<th>Budget 1999</th>
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<tr>
<td>3025</td>
<td>Field trials flexible roof bolt project</td>
<td>120,000</td>
<td>0</td>
<td>120,000</td>
</tr>
<tr>
<td>3027</td>
<td>Improving reinforcement design</td>
<td>120,000</td>
<td>0</td>
<td>120,000</td>
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<tr>
<td>3032</td>
<td>Improved roof stability through pre-tensioned roofbolting</td>
<td>100,000</td>
<td>0</td>
<td>100,000</td>
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<tr>
<td>3059</td>
<td>Rib mechanics and support systems</td>
<td>70,000</td>
<td>70,000</td>
<td>140,000</td>
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<tr>
<td>3067</td>
<td>Roof and goaf monitoring for strata control in longwall mining</td>
<td>120,500</td>
<td>427,500</td>
<td>548,000</td>
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<tr>
<td>3068</td>
<td>Cost effectiveness of various timber chock constructions for longwall tailgate support</td>
<td>25,000</td>
<td>10,462</td>
<td>35,462</td>
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<tr>
<td>3069</td>
<td>Determination of stress relaxation axes in drill core using laser micrometry</td>
<td>90,000</td>
<td>306,000</td>
<td>396,000</td>
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<td>3104</td>
<td>Rib bolting commissioned study</td>
<td>17,400</td>
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<tr>
<td>3105</td>
<td>Underground thick seam rib support</td>
<td>0</td>
<td>60,000</td>
<td>60,000</td>
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<tr>
<td>4024</td>
<td>Testing of roadway roof integrity</td>
<td>90,000</td>
<td>60,000</td>
<td>150,000</td>
</tr>
<tr>
<td>4025</td>
<td>Prestressing of strands to improve cable support performance</td>
<td>195,000</td>
<td>0</td>
<td>195,000</td>
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<tr>
<td>4026</td>
<td>Engineered mine design in soft strata environments</td>
<td>292,888</td>
<td>585,091</td>
<td>877,979</td>
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<tr>
<td>4027</td>
<td>Detection of incompetent mine roof (Stage 2)</td>
<td>50,000</td>
<td>18,000</td>
<td>68,000</td>
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<tr>
<td>5021</td>
<td>Post grouting technology to reduce bolting cycle times</td>
<td>163,500</td>
<td>200,000</td>
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<tr>
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<td>Early prediction of catastrophic roof failure</td>
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<td>20,000</td>
<td>110,000</td>
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<td>5023</td>
<td>Improved reinforcement techniques for weak roof</td>
<td>165,000</td>
<td>205,000</td>
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<tr>
<td>5024</td>
<td>Measuring the strength of irregular shaped and rectangular pillars</td>
<td>21,700</td>
<td>17,768</td>
<td>39,468</td>
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<tr>
<td>5025</td>
<td>Improved monitoring system for better roof management</td>
<td>125,000</td>
<td>120,000</td>
<td>245,000</td>
</tr>
<tr>
<td>6030</td>
<td>The dynamics of windblasts in UG coal mines</td>
<td>165,000</td>
<td>153,000</td>
<td>318,000</td>
</tr>
<tr>
<td>6033</td>
<td>Improving the up-time efficiency of roadway development units by reduced primary bolting densities and routine secondary support</td>
<td>200,000</td>
<td>800,000</td>
<td>1,000,000</td>
</tr>
<tr>
<td>6034</td>
<td>Improving safety and performance of chock construction</td>
<td>71,080</td>
<td>20,120</td>
<td>91,200</td>
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<tr>
<td><strong>20 Proj TOTAL</strong></td>
<td></td>
<td><strong>2,292,068</strong></td>
<td><strong>3,072,941</strong></td>
<td><strong>5,365,009</strong></td>
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### Table 8 - Escape and rescue

<table>
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<tr>
<th>Proj</th>
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<th>TOTAL</th>
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</thead>
<tbody>
<tr>
<td>3 proj</td>
<td>Numbat upgrade approval and demonstration</td>
<td>60,000</td>
<td>110,000</td>
<td>170,000</td>
</tr>
<tr>
<td></td>
<td>New self contained self rescuer</td>
<td>110,000</td>
<td>10,000</td>
<td>120,000</td>
</tr>
<tr>
<td></td>
<td>Numbat upgrade approval and demonstration</td>
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### OH & S general

<table>
<thead>
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</tr>
</thead>
<tbody>
<tr>
<td>3 proj</td>
<td>(i) Improved systems for capturing incidence and causality data particularly in relation to permanent disablement</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>3 proj</td>
<td>(ii) Dust control</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>5 proj</td>
<td>(iii) Reduced musculo-skeletal injuries</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>1 proj</td>
<td>(iv) Reduced noise induced hearing loss</td>
<td>87,650</td>
<td>45,350</td>
<td>133,000</td>
</tr>
<tr>
<td>1 proj</td>
<td>(v) Reduced injuries due to vibration and jarring</td>
<td>77,820</td>
<td>43,350</td>
<td>121,170</td>
</tr>
<tr>
<td>2 proj</td>
<td>(vii) General</td>
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### Table 9 - Improved systems for capturing incidence and causality data particularly in relation to permanent disablement

<table>
<thead>
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<tbody>
<tr>
<td>3046</td>
<td>OH&amp;S commissioned study</td>
<td>27,773</td>
<td>47,140</td>
<td>74,913</td>
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<tr>
<td>4044</td>
<td>OH&amp;S commissioned study</td>
<td>15,000</td>
<td>0</td>
<td>15,000</td>
</tr>
<tr>
<td>6032</td>
<td>Improved incident reporting and analysis</td>
<td>20,000</td>
<td>0</td>
<td>20,000</td>
</tr>
<tr>
<td>3 proj</td>
<td>TOTAL</td>
<td>62,773</td>
<td>47,140</td>
<td>109,913</td>
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### Table 10 - Dust control

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</thead>
<tbody>
<tr>
<td>3082</td>
<td>Reduction of dust in return roadways of longwall faces</td>
<td>42,500</td>
<td>231,500</td>
<td>274,000</td>
</tr>
<tr>
<td>4041</td>
<td>Electrostatic enhancement of water sprays for dust suppression</td>
<td>88,000</td>
<td>177,250</td>
<td>265,250</td>
</tr>
<tr>
<td>5019</td>
<td>Improved longwall dust suppression</td>
<td>35,000</td>
<td>242,000</td>
<td>277,000</td>
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<tr>
<td>3 proj</td>
<td>TOTAL</td>
<td>165,500</td>
<td>650,750</td>
<td>816,250</td>
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### Table 11 - Reduced musculo-skeletal injuries

<table>
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<tr>
<td>3031</td>
<td>On-board rib bolting</td>
<td>100,000</td>
<td>0</td>
<td>100,000</td>
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<tr>
<td>3038</td>
<td>Development of a compact semi-automatic roof bolter</td>
<td>208,000</td>
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<tr>
<td>3060</td>
<td>Extension of compact autobolter project</td>
<td>175,000</td>
<td>0</td>
<td>175,000</td>
</tr>
<tr>
<td>3066</td>
<td>Application of light alloys and alternate materials in underground coal mines</td>
<td>104,350</td>
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<td>104,350</td>
</tr>
<tr>
<td>5366</td>
<td>Application, risk and benefits of using aluminium in UG coal mines</td>
<td>10,000</td>
<td>0</td>
<td>10,000</td>
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<tr>
<td>5 proj</td>
<td>TOTAL</td>
<td>597,350</td>
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### Table 12 - Reduced noise induced hearing loss

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<tbody>
<tr>
<td>4043</td>
<td>Adapting active noise control headsets for the coal mining industry</td>
<td>87,650</td>
<td>45,350</td>
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<tr>
<td>1 TOTAL</td>
<td></td>
<td>87,650</td>
<td>45,350</td>
<td>133,000</td>
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### Table 13 - Reduced injuries due to vibration and jarring

<table>
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<th>Description</th>
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<th>OTHER</th>
<th>TOTAL</th>
</tr>
</thead>
<tbody>
<tr>
<td>5040</td>
<td>Development of test procedure for assessing whole body vibration</td>
<td>77,820</td>
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<td>121,170</td>
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<tr>
<td>1 TOTAL</td>
<td></td>
<td>77,820</td>
<td>43,350</td>
<td>121,170</td>
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### Table 14 - Reduced exposure to diesel exhaust

<table>
<thead>
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</tr>
</thead>
<tbody>
<tr>
<td>3033</td>
<td>Improved diesel engine performance with lower emissions</td>
<td>317,000</td>
<td>174,000</td>
<td>491,000</td>
</tr>
<tr>
<td>3080</td>
<td>Evaluation and control of employee exposure to diesel exhaust emissions</td>
<td>223,000</td>
<td>0</td>
<td>223,000</td>
</tr>
<tr>
<td>3081</td>
<td>Effects of diesel fuel quality on exhaust emissions of U/G mining engines</td>
<td>87,000</td>
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<td>87,000</td>
</tr>
<tr>
<td>3 TOTAL</td>
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<td>627,000</td>
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<td>801,000</td>
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</table>

### Table 15 - General

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<th>ACARP</th>
<th>OTHER</th>
<th>TOTAL</th>
</tr>
</thead>
<tbody>
<tr>
<td>5038</td>
<td>Development of a standard underground approval system</td>
<td>30,000</td>
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<td>30,000</td>
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<td>6058</td>
<td>Exchange of OHS data with South Africa</td>
<td>50,000</td>
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### Open Cut Category

### Table 16 - Open cut vehicle and equipment

<table>
<thead>
<tr>
<th>Projec ts</th>
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<th>OTHER</th>
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</thead>
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<tr>
<td>4013</td>
<td>Intelligent dumper and hauler suspension systems</td>
<td>125,000</td>
<td>20,000</td>
<td>145,000</td>
</tr>
<tr>
<td>5041</td>
<td>Reflective material for improving night time driving on haul roads</td>
<td>66,000</td>
<td>45,000</td>
<td>111,000</td>
</tr>
<tr>
<td>5413</td>
<td>Intelligent dumper and hauler suspension system</td>
<td>29,500</td>
<td>0</td>
<td>29,500</td>
</tr>
<tr>
<td>6007</td>
<td>Intelligent dumper and hauler suspension systems (Stage 3)</td>
<td>140,000</td>
<td>77,000</td>
<td>217,000</td>
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<tr>
<td>6008</td>
<td>Development of a whole body vibration dosimeter</td>
<td>96,850</td>
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<td>140,500</td>
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<td>5 TOTAL</td>
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<td>457,350</td>
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<td>643,000</td>
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### Table 17 - OH&S General

<table>
<thead>
<tr>
<th>Projec ts</th>
<th>Description</th>
<th>ACARP</th>
<th>OTHER</th>
<th>TOTAL</th>
</tr>
</thead>
<tbody>
<tr>
<td>4012</td>
<td>Multifactorial back damage intervention study</td>
<td>209,500</td>
<td>0</td>
<td>209,500</td>
</tr>
<tr>
<td>4014</td>
<td>Emissions from spoil-pile fires</td>
<td>154,000</td>
<td>25,000</td>
<td>175,000</td>
</tr>
<tr>
<td>2 TOTAL</td>
<td></td>
<td>363,500</td>
<td>25,000</td>
<td>384,500</td>
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Accuracy of Measurement of Gas Content of Coal Using Rapid Crushing Techniques

A Saghafi\(^1\), D J Williams\(^1\) and S Battino\(^2\)

SUMMARY

The rapid crushing technique for the measurement of gas content of coal consists of accelerating the rate of gas desorption from coal by crushing. This technique allows the determination of gas content in the space of hours rather than days or weeks as required for the traditional slow desorption technique. The crushing technique is used particularly when gas content determinations are urgently required for mine safety purposes. The method is now routinely used in underground coal mines of NSW and to a lesser extent in QLD. Despite the rapidity and efficiency of this method, concerns have been raised upon the reliability of the method to deliver accurate results. Experience in Australia has shown that there is in fact some discrepancy in results between gas laboratories using the rapid crush technique. The development of rapid crushing techniques for seam gas content measurements in Australia and the main causes of variability in the gas content test results were investigated. The gas laboratories participating in the gas content testing were CSIRO Division of Coal and Energy, BHP Technical Services and GeoGas Pty. Ltd.

INTRODUCTION

Gas content is probably the most important parameter to be quantified in order to characterise a coal seam both from mine safety and gas recovery viewpoints. Worldwide, numerous direct and indirect methods of determining the gas content of coal are practiced. The direct method is based on the direct measurement of gas volume evolved from coal whereas the indirect method consists of measuring other coal properties and using the established relationships between these parameters and gas content to evaluate the latter. The two principal variants of the direct method are the slow desorption technique and the quick crush method.

In Australia the slow desorption method was widely used for the last 20 years. More recently the quick crush method was introduced and used initially for residual gas content determination followed by its current use for full gas content determination. While this method is fully operational, some concerns were raised by industry on variability of gas content results between gas laboratories using the quick crushing method. This paper addresses and quantifies some of the factors, which are identified by the authors to have influence on the gas content results using this method.

DEVELOPMENT OF QUANTITATIVE MEASURE OF SEAM GAS CONTENT

In Australia, initial rudimentary estimates of coal seam gassiness were made in the 1960s and 1970s during exploration drilling by coal geologists in the field using the Hargraves test. This consisted of quickly enclosing a coal core still in the inner tube of the core barrel in a sealed vessel for 24 hours and then sampling the gas released. On the basis of the gas analysis the level of gassiness of the coal bed was qualitatively determined by examination of the ratio between air and gas in the sample. In the early 1980s increased coal production rates and deeper mining conditions demanded greater interest in the gas content of coals ahead of the working face. A modified version of the USBM Direct Method (McCulloch and

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\(^1\) CSIRO, Coal & Energy Technology
\(^2\) Mining Consultant, Gastrade Pty Ltd
Diamond, 1976) was adapted by operating gas engineers and geologists in the industry to quantify the seam gas content ahead of mining (Battino and Doyle, 1983).

The need to standardise and document guidelines to provide industry with an acceptable, reliable and reproducible method of gas content determination led to the formation of a professional working group of industry personnel in March 1984. In 1986, an initial draft document reflecting the views of committee members was prepared and reviewed. This resulted in a document entitled “Guide to the determination of desorbable gas content of coal seams - Direct Method” which was circulated for public comments and recommendations. In 1991, the Australian Standard document AS3980 was issued as an acceptable working guide for the determination of seam gas content using the Direct Method. This was then used as the operating standard for Australian industry (Standards Association of Australia, 1991).

In 1995, the SAA working group was reconvened to discuss and review recommended changes to the existing document with particular emphasis on the following issues,

1. precision and reporting of the test results;
2. sampling and sub-sampling of materials tested;
3. the inclusion of the Quick Crush method; and
4. the factors affecting the accuracy of seam gas content determination.

Over the next 18 month period, regular committee meetings were held to examine and debate the above issues and this has led to the current Pre Publication Review Draft of a new AS 3980 Guide which is in its final completion stage. The factors affecting the precision and accuracy of seam gas content determination are included in order of importance (as deemed by the working group) in Appendix A of the new document. While a test result accuracy of 10% is indicated in the Guide, this has yet to be proven. To this end, an ACARP funded project aiming to study and quantify the effects of these factors was operated by the CSIRO Division of Coal & Energy, with support from the BHP Technical Services and GeoGas laboratories.

DIRECT METHODS OF SEAM GAS CONTENT MEASUREMENT

Slow desorption method

In Australia the slow desorption method of gas content testing has been used for almost 20 years. This modified USBM method (McCulloch and Diamond, 1976) consists of enclosing a coal sample in a sealed container and measuring the volume of the gas evolved. Gas desorbs from coal naturally and the rate of desorption depends uniquely upon the coal gas desorption characteristics but a single gas content determination may take up to a few months to complete. This time span can introduce various sources of error, which Lama (1995) examined and quantitatively assessed in detail. There are three main sources of error uniquely related to the slow desorption method.

1. System leakage

Clearly, the longer the period of time required to determine the seam gas content in the laboratory, the more chances of error due to system leakage, repetitive visual errors in taking desorption readings and mistakes in opening or closing valves during testing.

2. Solubility of CO₂ in water

Significant volumes of CO₂ gas can be dissolved by the water of the gas collection system (even when using acidified brine) particularly if the gas is bubbled up through the water from the bottom of the test cylinder. This depends on the size of the bubbles influenced by the exit nozzle diameter and the height of the water column. The residence time of
the desorbing gas contained by the water also plays a role as the dissolved CO₂ diffuses through the water column to the atmosphere. The slow desorption method is thus more prone to this error.

3. **Effect of changes in temperature**

During long periods of testing (such as weeks or months), there are likely to be substantial variations in ambient temperatures which can have a pronounced effect on the final gas content value obtained. It is generally recommended that tests be carried out in temperature-controlled laboratories at 20 °C. Tests carried out by R.D. Lama at temperatures in the range of 20 to 40 °C indicated errors of 0.135cc/g°C for CO₂ and of 0.07cc/g °C for CH₄.

**Quick crushing method**

The continuing increase in the rate of heading development and longwall face advance has forced the mine gas laboratories in Australia to seek faster ways of determining the gas content of coal. Some researchers suggested to estimate gas content of coal based on the initial gas desorption rate and using a database for correlating the initial gas release to the total desorbable gas content (Williams, 1997). Lama (personal communication, 1993) has suggested to use gas release rate from wet and dry drill cuttings to estimate gas content of coal. However in both of these procedures gas content is not directly measured and intensive site specific correlation between gas content and other parameters are required to be established before using the method.

The most reliable way of rapid and direct gas content determination is to crush coal, which significantly increases its gas, desorption rate. The crushing method has been widely used in most European countries with deep coal mining. The method was first developed and applied in Western Europe (Bertard et al., 1970). The rapid desorption or quick crush method has been effectively used in Australia more recently (Williams et al., 1992). The method consists of accelerating the process of gas desorption by crushing coal in a sealed container. Using this method, the total seam gas content can be determined in a space of 2 to 3 hours and this is now extensively used by the most active gas laboratories in NSW and is also rapidly progressing in Qld.

There are six major steps of this method.

1. The coal sample or core obtained from the selected underground site is first used to estimate lost gas during drilling (Q₁). This is achieved by measuring the gas desorbed for a period of 20 to 30 minutes. The volume of gas released underground (Qₐᵤ) must be recorded for determination of total desorbable gas content.

2. At the completion of the underground measurement, the canister is sealed and transported to the gas laboratory where the volume of gas evolved since sealing at the underground site is now measured (Q₂ₗ).

3. Some laboratories need to subsample coal (200 to 400 g) for crushing by transferring coal from the transport canister to the crusher. To enable the calculation of the gas lost in the period between opening the transport canister for sub-sampling and sealing in the coal crusher, the rate of gas release should be determined. This is achieved by measuring the gas evolved over a period of time. The gas released during this time and transfer time is Q₃ₜᵣ.

4. Coal sample is then crushed and gas released during crushing, Q₄ₚ, is determined.

5. The total desorbable gas content (Qₜₒ) is then determined by adding all the various gas components:

   \[ Qₜₒ = Q₁ + Q₂ₚ + Qₐₘ + Q₃ₜᵣ + Q₄ₚ \]

6. In some instances gas may continue to be released from crushed coal for a longer period. This gas (Q₅ₚ) can be measured if total gas content, Q₅ᵣ, is required.

The gas content values are generally reported in normal conditions (20 °C temperature and 1 atm. pressure).
Advantages of the quick crushing method

The significant advantages of this method include:

1. Very fast evaluation of the total gas content of the coal at a particular test site in the mine, essential for mining authorisation from the Mining Inspectorate and for gas predrainage purposes.

2. Significant reduction in the chances of leakage, dissolution of CO$_2$ in water and oxidation of the coal during testing.

3. Quick turnaround in freeing up the available field and laboratory equipment which can then be reused for other tests.

4. Reduction in laboratory cost because of shorter time for monitoring, less frequent gas volume and composition measurements and better resources utilisation.

INTER-LABORATORY COMPARISON OF GAS CONTENT MEASUREMENTS USING THE QUICK CRUSH METHOD

Field measurements

In order to achieve the desired level of comparison of gas content test results from the different operating laboratories, drilling and coring were conducted at a number of underground test sites in various NSW collieries. A total of 38 bore cores were collected from 8 mine sites situated in 5 collieries of the Illawarra and Hunter Valley coalfields.

At each site, the work consisted of drilling a borehole from a development heading into a virgin coal area and then coring end to end at regular spacing of 1.5 to 2 m. Most cores were obtained at a depth of 10 to 20 m from the rib, with the exception of one case where 5 cores were taken at the depth of 50 to 60 m. From each borecore, coal samples were apportioned to the three participating laboratories so that the samples were uniform and representative across laboratories. For each core section obtained, underground readings for $Q$, were undertaken using one of the canister samples while the other two canisters were left open. After the completion of $Q$, readings, all coal sample canisters for that particular core section were sealed, transported and delivered for testing to each of the participating gas laboratories as soon as possible (generally 2 to 12 hours after underground testing).

Laboratory measurements

Once the coal samples were received by each laboratory, the coal canisters were tested for leakage and the gas volumes released during transport and in the laboratory were measured. This quantity defined as $Q$, is similar to $Q_{2L}$ of quick crushing method described in previous sections. The coal samples were then placed in the crusher and the gas volume released during crushing was measured. This quantity defined as $Q_c$ is similar to $Q_{cl}$ in quick crush method described in previous sections.

The three operating laboratories use three different sets of coal canister and crushing equipment. The volumes of the canisters and crushing systems are also different. BHP and GeoGAS laboratories use larger coal transport canisters and crusher and must open the coal transport canister to subsample a part of the coal for crushing. With the CSIRO method, the coal transport canister can be directly mounted on the crushe and coal is crushed without being removed from the underground canister. GeoGAS uses a ring crusher whereas BHP and CSIRO use ball mill crushers.

Variability in measured gas content

The gas content results across participating laboratories were compared on the basis on the value of $Q_c+Q$. This is the total volume of gas desorbed from the time the coal sample was sealed underground in the transport canister until the
crushed coal was removed from coal crusher. A comparison of the $Q_c$ or $Q_a$ values alone would give rise to errors as the timing to start coal crushing was not strictly the same across the three participating laboratories.

In Fig. 1, the results of gas content measurements undertaken for the 38 borecore samples are presented. The gas contents are given in terms of $Q_c + Q_a$. The mean gas content for a given borecore is the arithmetic average of gas content values obtained by participating laboratories for that bore core. The bisector corresponds to the mean gas content. A measure of the variability in gas content measurements can be presented by the absolute or relative distance of the gas content measured by each laboratory from this line. This approach was adopted in this study and all individual values are compared to the mean values of gas content.

To estimate the variability in gas content results two terms: 'absolute variability' and 'relative variability' were defined. The absolute variability for a borecore is equivalent to the standard deviation of gas content results from the mean gas content for that borecore. The relative variability of results for a borecore corresponds to the standard deviation normalised over the mean gas content of the borecore. In Fig. 2 the absolute variability against mean gas content is shown. As can be seen, there is no strong trend in the data. However there is a weak trend in the scattered data, suggesting that a larger seam gas content value would produce a larger absolute variation between laboratories. The graph also shows that the maximum variation for all measurements except for one was under 1.2 $m^3/t$. In Figure 3, the graph of relative variability versus mean gas content is shown. This indicates that the relative variability in gas content across participating laboratories has no correlation with gas content and in all cases except two remains under or near 15%.

Discussion on causes of variability in gas content measurements

Gas content of borecore samples measured by the three laboratories showed a relative variability of 15%. The calculation of variability was based on the standard deviations from mean gas contents, consequently in terms of relative variability between two individual labs the upper limit was as high as 30%. This amount of variation is rather high and in the course of the study, some causes of the variability were removed. While tests were conducted by the various laboratories to identify leakage prior to commencing desorption, no leaks were reported. Three sources of variation in gas content results across the participating laboratories were identified.
Substantial error can occur when high proportions of carbon dioxide (CO₂) are present. CO₂ gas is highly soluble in water and current methods of measuring the gas volumes by water displacement in an inverted cylinder can only increase the variability in gas content results across laboratories. The CO₂ loss is especially high when gas is fed from the base of the test cylinder and has to travel through the water column. CO₂ is also lost at the interface between water column and gas. The factors, which determine the quantity of gas loss, are the residence time, the area of water-gas interface in the measuring cylinder and the area of the water-air interface in the water tray.

The traditional method to prevent CO₂ dissolution is to use an acidified water solution instead of pure water. This however merely inhibits the formation of carbonate ion, which is very soluble in the acidified brine. In an experiment at the CSIRO gas laboratory, 500 mL of pure CO₂ gas was injected from the bottom into an inverted 2L measuring cylinder filled with 1liter of either distilled water or acidified brine. The height of the water column was 20 cm and bubbling of the gas...
through the water column took approximately 60 seconds. The gas volume lost traveling in pure water was 95 mL or 19% and in acid brine dissolution was 73 mL or 15%.

The effect of gas standing over a long period of time over acidified water in an inverted gas collection cylinder was also studied. Gas containing CO₂ was injected from the top of the cylinder and gas samples were collected and measured at regular intervals. It was found that the gas composition changed with time as the CO₂ dissolved. Gas initially containing 40% CO₂ and 60% CH₄ only had 27% CO₂ after 20 hours while for gas starting at 89% CO₂ and 11% CH₄, the level fell to 68% CO₂.

In another experiment, the effect of separating gas and liquid phases on gas loss was investigated. A layer of raw linseed oil of 4mm thickness was placed on top of the acidified brine and then gas containing 90.6% CO₂ and 11.4% CH₄ was injected from the top of cylinder. After 24 hours, the concentration fell to 90.2% CO₂ and after 336 hours, it fell to 80.2% CO₂.

All the laboratories involved in this project now feed gas from the top of the measuring cylinder. In addition CSIRO uses an oil barrier between water and gas to minimise contact between the two phases.

From the above discussion it can be seen that an error of up to 15% in gas content can be expected when seam gas is very rich in CO₂ and gas is fed from the base of measuring cylinder. It should be noted that methane (CH₄) gas is also soluble in water but in much lower quantity compared to CO₂. The error relating to CH₄ loss in water is estimated to be about 1%. The effect of residence time, that is the time of gas standing over the water column, can be neglected as in the crushing method this time is in order of only minutes.

**Effect of gas partial pressure**

The partial pressure of seam gases can alter the overall volume and the rate of gas desorption from coal. The void volumes in the crusher and transport canister varied among the participating labs. The volume of void space in the crushing container depends on the volume of the container and the mass of coal. The ratio of crushing container volume (mL) to coal mass (g) is used to express the void volume. This ratio varied from 2.1 to 6.1 across the participating laboratories. The variation in this ratio has an effect on the rate of desorption when the coal gas content is low or the coal contains mixed gases.

In order to quantify the effect of gas partial pressure, after each completion of a crushing and measurement of gas content some 10% of crushed coal was left in the crusher and gas released from coal was monitored for a day or longer. In most cases gas continued to desorb from crushed coal under the new low partial pressure. In Fig. 4, the measured 'residual' gas contents under very low gas pressure conditions, of 38 borecore samples, are shown. From Fig. 4 it can be seen that no correlation exists between gas released after crushing (Q'c) and gas released before and during crushing (Qₜ + Qc). The majority of coal samples showed a Q'c below 0.8 m³/t. The same data are shown in Fig. 5 where Q'c is normalised by Qₜ + Qc values. The graph shows that gas released under low partial pressure for the borecores measured is in most cases under 12%, but is more significant for the lower gas contents and occasionally can reach as high as 19% of measured Qₜ + Qc. Consequently it can be concluded that variability due to variation in gas pressure is limited to 12% in most cases.

**Effect of temperature**

Coal crushing generates heat and temperature rise changes the gas desorption rate. The effect of temperature rise on gas desorption rate is under study and is not yet quantified. Crushing time varied among laboratories from 7 to 90 minutes. At this stage all participating laboratories try to avoid temperature rise by using a multi-cycle crushing and cooling procedure or they use correction factors to compensate for the increase in temperature. During the measurements the ambient temperatures in participating laboratories were in the range of 20 to 22 °C.
CONCLUSIONS AND RECOMMENDATIONS

Analysis of seam gas content and composition figures determined by the participating laboratories indicates that there are some variations in both gas content and gas composition results. The causes of variability in gas content can be the dissolution of CO₂ in water, different void ratio or gas partial pressure in coal containers and temperature changes during crushing. The causes of variability in gas composition can be the dissolution of CO₂ in water and oxidation of coal particularly for low gas content coal samples. The problem of gas dissolution in acidified water was reduced dramatically by feeding gas from the top of cylinder. All laboratories involved in this study now practice this method. Using a linseed oil barrier to separate gas from water in the measuring cylinder further reduced this dissolution. Effect of temperature
increase on gas volume and rate of desorption is minimised by crushing coal in a sealed container and then allowing a cooling period before opening the valve and measuring the gas volume desorbed.

For the 38 bore cores tested for gas content, the absolute variability from the mean gas content in all cases except one was found to be less than 1.2 m$^3$/t. The relative variability in terms of mean gas content was in all cases except three below 15%. However variability in results between two individual labs can be higher and in some cases be as high as 30%. Further research is essential if close positive correlation to the suggested level of 10% is to be achieved.

The necessity to achieve accurate, reliable and reproducible gas content and composition test results at all operating laboratories remains paramount to ensure the safety of underground mine personnel as well as the required development rates. While it is accepted that the accurate measurement of seam gas content of coal is no easy task, it is imperative that continuing research be maintained to overcome all significant variability and create an Australian Standard which ensures that comparable and consistent data are obtained and reported at all times.

ACKNOWLEDGEMENTS

The authors wish to thank the ACARP organisation for funding the project. Their thanks also go to Dr R. Williams and Mr F. Mattas from GeoGAS Pty Ltd., Mr J. Wood and Mr A. Gurbka of BHP Technical Services and Mr D. Roberts of CSIRO for actively contributing to the test work. Gratitude is also extended to those underground coal mines in the Sydney and Hunter coal fields which made valuable contributions to this project by agreeing to undertake the drilling and coring for coal samples as required.

REFERENCES


Contrast in Gas Sorption at Dartbrook and South Bulli Collieries

P J Crosdale

ABSTRACT

Dartbrook (Hunter Valley and South Bulli (Southern Coalfield) Collieries provide contrasting settings for gas sorption studies. Gas compositions at Dartbrook are dominated by carbon dioxide, with minor methane, while those at South Bulli are dominated by methane with minor carbon dioxide. Total gas contents at Dartbrook are also significantly greater than those at South Bulli. The two collieries are also contrasted in terms of coal composition and mining environment. Dartbrook Colliery mines a relatively small section (4m) of very thick seam (up to approximately 25m); the coal is vitrinite-rich, high volatile bituminous. South Bulli mines most of the seam thickness, with coal being an intertinite-rich, medium volatile bituminous coal.

Coal samples have been collected from in-seam gas drainage holes for evaluation of relative gas desorption rate (methane versus carbon dioxide). Following adsorption isotherms, petrography, chemistry (proximate and ultimate analysis) and pore structure analysis (carbon dioxide surface area, mercury porosimetry). Bomb desorption studies indicate different behaviours for methane desorption at these two collieries desorbed gas compositions at South Bulli show a small enrichment in methane content. The opposite is observed at Dartbrook. Both collieries show enrichment in CO₂ in the residual content. Adsorption studies show marked variation in adsorbed gas contents for methane and carbon dioxide at each colliery and between collieries. These variations are related to differences in coal pore structure, chemistry and petrology.
Contrasts in Methane Sorption Properties Between New Zealand and Australian coals

B Beamish¹, C Laxminarayana² and P J Crosdale²

ABSTRACT

High pressure microbalance investigations have produced results for both New Zealand and Australian coals which show fundamental differences in their methane sorption properties. New Zealand high volatile bituminous C rank coals have a methane adsorption capacity of 38 cc/g(daf) which decreases to a minimum of 23 cc/g(daf) at medium volatile bituminous rank and increases to 31 cc/g(daf) at low volatile bituminous rank. Vitrinite-rich coal samples from Australia display a similar trend, but, the methane adsorption capacity is approximately 8 cc/g higher than for New Zealand coals at low volatile bituminous rank increasing to 20 cc/g higher at high volatile bituminous A rank. From these differences it is implied that New Zealand coals contain a lower proportion of microporosity than Australian coals, most likely due to the presence of volatile components blocking the micropore structure making them unable to sorb as much methane.

INTRODUCTION

Little published data exists on the methane sorption properties of New Zealand coals, making it difficult to assess the likely potential of both methane recovery from coal seams and resulting emissions from underground mining. Methane adsorption studies were recently conducted on these coals using high pressure microbalances. The same equipment and technique has been used to study Australian coals. This paper compares the results obtained for coals from both countries and assesses the influence of coal rank and type on methane adsorption capacity.

EXPERIMENTAL

Coal samples

The New Zealand coal samples studied were obtained from the Coal Research Ltd sample bank in Wellington, New Zealand. Two sets of samples were analysed:

1. a rank suite of run-of-mine coals covering the range from high volatile bituminous C to low volatile bituminous (Table 1); and

2. a lithotype suite of high volatile bituminous C rank from the Greymouth Coalfield (Table 1). This lithotype suite is distinguished by increasing amounts of vitrinite bands present: bright non-banded (752-5), bright <25% vitrinite (55/382), bright 25-30% vitrinite (55/381) and pure vitrinite (752-8). All New Zealand coal samples were supplied as splits from samples prepared for proximate analysis.

The Australian coal samples were obtained from underground in-seam borehole cores in the Sydney Basin and vertical surface boreholes from across the Bowen Basin. Bright, vitrinite-rich fractions covering the rank range from high volatile bituminous A to low volatile bituminous (Table 1), were used for direct comparison in this study, as the New Zealand coals are predominantly vitrinite-rich with subordinate amounts of liptinite. To show the effects of coal type, two Australian inertinite-rich, high volatile bituminous A rank samples (Table 1) were also analysed for comparison with the New Zealand high volatile bituminous coal type suite. The Australian samples were crushed to <212 mm and a separate split of this product was used.

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for thermogravimetric proximate analysis (Beamish, 1994).

<table>
<thead>
<tr>
<th>Sample No.</th>
<th>Moisture (%, ar)</th>
<th>Ash (%, db)</th>
<th>Volatile matter (% daf)</th>
<th>Langmuir volume (cc/g, daf)</th>
<th>ASTM Rank</th>
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<tr>
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<td>8.0</td>
<td>21.8</td>
<td>31.1</td>
<td>lvb</td>
</tr>
</tbody>
</table>

**New Zealand high volatile bituminous C coal type suite (see text for descriptions)**

| | | | | | |
| 752-8 | 7.8 | .4 | 37.2 | 39.8 | hvCb |
| 55/381 | 6.6 | 3.4 | 40.1 | 41.9 | hvCb |
| 55/382 | 5.9 | 1.3 | 42.9 | 39.2 | hvCb |
| **752-5** | | | | | |

**Australian vitrinite-rich coal rank suite**

| | | | | | |
| G102/14CTBR | 5.4 | 0.5 | 33.0 | 46.3 | hvAb |
| G102/15CTBR | 5.4 | 0.2 | 34.6 | 46.0 | hvAb |
| NM1-04BR | 1.6 | 9.4 | 27.4 | 38.6 | mVb |
| NM1-21BR | .5 | 5.3 | 22.0 | 38.1 | mVb |
| PH1-03BR | 1.5 | 16.2 | 19.0 | 38.0 | mVb |
| RS1-02BR | 1.2 | 3.8 | 18.1 | 38.0 | lvb |
| **SM1-05BR** | | | | | |

**Australian high volatile bituminous A inertinite-rich coal types**

| | | | | | |
| SM1-05BR | 1.4 | 6.6 | 15.0 | 39.4 | lvb |
Methane adsorption determinations

High pressure microbalances were used to measure the methane adsorption by the samples. All coals were initially dried to avoid effects of moisture and enable direct comparisons of the maximum methane adsorption capacities (Langmuir volumes as determined from the Langmuir equation) of each coal. Methane volumes adsorbed at approximately, 0.5, 1.0, 2.0, 3.0, 5.0, 7.0, and 9.0 MPa were recorded. Calculations used to determine the methane adsorbed are reported in Crosdale and Beamish (1995).

RESULTS AND DISCUSSION

A comparison of the Langmuir volumes of the two suites of coals is shown in Figure 1, plotted against volatile matter (% daf). There is a pronounced minimum in methane adsorption capacity (Langmuir volume) at medium volatile bituminous rank for both Australian and New Zealand coals. This minimum is quite sharp for the New Zealand coals, but appears to be rather broad for the Australian coals. There is a shift in the position of the minimum (based on volatile matter), which is partly real and partly due to the New Zealand coals having a higher volatile matter content than Australian coals for any given rank (based on vitrinite reflectance). The shift in position of the minimum in Fig. 1, which is partly real and partly due to the New Zealand coals having a higher volatile matter content than Australian coals of equivalent rank should be noted.

![Figure 1](image_url)

Fig. 1 - Variation in Langmuir methane volumes for Australian and New Zealand coals. Both suites of coal show a minimum in Langmuir volume at medium volatile bituminous rank

The Australian coal trend is very similar to that obtained by Griffith and Hirst (1944) for coal internal surface areas, whereas the New Zealand coal trend is similar to that obtained by Kini (1964). In absolute terms the Australian coals at high volatile bituminous rank adsorb as much as 20 cc/g more methane than the New Zealand coals. This difference in adsorption capacity
appears to decrease with increasing rank, such that at low volatile bituminous the difference is as much as 8 cc/g. Measured CO$_2$, internal surface areas for these coals are 70 m$^2$/g for the New Zealand high volatile bituminous A rank coal (Clemens, Matheson and Rogers, 1991) and 200 m$^2$/g for a similar rank coal from the Sydney Basin.

A direct implication of this difference in adsorption capacity between the Australian and New Zealand coals of equal rank is that lower gas contents could be expected for New Zealand coals at equivalent depths. Gas contents as high as 10 cc/g have been recorded for New Zealand high volatile bituminous coals at depths of 700 m. The same gas contents are found in high volatile bituminous Australian coals at depths as shallow as 130 m in the Bowen Basin and 400 m in the Sydney Basin. Conversely, New Zealand coals with equivalent gas contents to Australian coals would have substantially higher seam pressures. This has serious consequences for trying to apply gas content threshold limits to New Zealand conditions.

There is a general increase in the methane adsorption capacity of the high volatile bituminous C New Zealand coal with increase in vitrain content, ranging from 35.4 cc/g to 41.9 cc/g (Table 1). A similar relationship exists for the high volatile bituminous A Australian coal, which ranges from 37.4 cc/g to 46.0 cc/g for one lithotype pair, and 38.2 to 46.3 cc/g for the other (Table 1).

It is well documented that the majority of the methane adsorption takes place in the micropores of the coal. The observed decrease in methane adsorption capacity from high volatile bituminous C to medium volatile bituminous rank has been attributed to "plugging" of the micropore system by "low boiling" hydrocarbon constituents (Thomas and Damberger, 1976). As coalification continues, cracking of the occluded oils during debituminization re-opens the micropore system increasing the availability of adsorption sites (Levine, 1993). Levine (1991) also points out experimental and field evidence indicating that methane adsorption capacity of coal is diminished by the presence of entrapped oils. He states that, "Hypothetically, molecules of occluded volatile constituents occupy molecular sites that would otherwise be accessible to methane, thus decreasing the methane sorption capacity". Toda et al. (1971) came to a similar conclusion suggesting that the decrease in micropore volume was strongly influenced by the concentration of hydrogen atoms bound directly to carbon atoms on the pore walls. The marked contrast in the methane adsorption capacity between the New Zealand and Australian coals in the high volatile bituminous rank (Figure 1) support this view, with the New Zealand coals for a given rank having a higher volatile component.

While both coal type suites showed the same trend of increasing methane adsorption capacity with increasing vitrain content (Table 1), the mechanisms for the trends appears to be different. The New Zealand coal type suite shows a consistent decrease in volatile matter with increasing vitrain content, and hence the increase in methane adsorption as discussed above. However, the Australian coal type suites shows a consistent increase in volatile matter content with increasing vitrain content. In this case the difference is due to a greater increase in the proportion of the macropores present in the coal, not a loss of micropores.

**CONCLUSIONS**

The methane adsorption capacity of New Zealand coals is much less than Australian coals. At low volatile bituminous rank the difference is 8 cc/g and increases to 20 cc/g at high volatile bituminous A. These differences are primarily due to decreased microporosity in the New Zealand coals from the presence of volatile components blocking the micropore structure. The same mechanism accounts for the rapid decrease in methane adsorption capacity from high volatile bituminous C to a minimum at medium volatile bituminous rank for the New Zealand coals. The Australian coals show a similar trend, but the decrease is not as rapid.

Coal type also shows a significant effect on the methane adsorption capacity of the high volatile bituminous coals from both countries. Generally, there is an increase in methane adsorption capacity with increase in vitrain content of the coal. However, the mechanism for this increase is different for New Zealand coals compared to Australian coals. The New Zealand coal type trend is controlled by the presence of volatile components producing a similar effect to the rank trend, whereas the Australian coal trend is a result of increased macroporosity relative to microporosity in the coal.
ACKNOWLEDGMENTS

The authors would like to thank the Auckland University Research Committee and the James Cook University Merit Research Grant for financial assistance with this project. Dr Tim Moore and Coal Research Ltd. provided the New Zealand samples used in this project.

REFERENCES


APPIN AND TOWER METHANE ENERGY PROJECT

A world first in energy innovation

BHP has teamed with Energy Developments Limited and Lend Lease Infrastructure, to establish an electrical power generation facility that utilizes methane produced as a by-product of mining at BHP’s Appin and Tower coal mines in southern New South Wales.

In what is believed to be a world first, Energy Developments Limited has developed technology which will capture not only the drained methane, but also a portion of that which is present in the ventilation air, reducing BHP Collieries Division’s greenhouse emissions by approximately 50 percent. The methane gas is converted to electricity using state-of-the-art lean burn gas engine technology.

Construction and installation of two plants, comprising a series of one megawatt gas engines, commenced in July 1995. Full capacity was achieved on 5 September 1996. The combined output is 94 megawatts of electrical power.

Gas supply tends to vary with geological conditions and mining performance. An advantage of the multi-engine concept is that engines are brought on-line as required to match the available gas supply, thus optimising the total efficiency of the plants.

The gas engines are capable of consuming a total of 161,262 tonnes of methane per annum (93,712 tonnes from Appin and 67,550 tonnes from Tower).

The electricity generated at the plants will be supplied to the local distributor, Integral Energy, and be sufficient for up to 60,000 homes.

Environmentally – Positive power generation

The Methane Energy Project is a powerful illustration of how an imaginative approach to greenhouse reduction can achieve a number of beneficial effects simultaneously, namely the prevention of methane being vented into the atmosphere; high safety standards prior to mining activity in the mines; and the advantages of electricity production from an otherwise unused resource.

BHP’s Appin and Tower mines are considered gaseous, meaning for reasons of safety they require gas drainage (drilling bore holes into the coal seam and strata ahead of the mining operations and piping the methane to the surface). Large volumes of ventilation air must also be passed through the mine to dilute the undrained methane.

In the past, methane at the Appin and Tower mines was captured by the methane drainage plants and exhausted along with mine vent air directly into the atmosphere. Some of the methane from Appin Colliery was used to power a gas turbine to generate up to 14 megawatts for supply to the State grid.

The Methane Energy Project is unique in that it is not only reducing greenhouse gas emissions, but utilizing available energy sources in capturing the methane and burning it in high efficiency engines to produce electricity.

The project will reduce Australia’s output of greenhouse gases by 0.5 percent, helping to meet the national emission reduction target. In fact, this project is the single largest contributor to reduction targets in Australia.

1 Operations Manager, Energy Developments, Appin
An award-winning development

The success of the project in reducing greenhouse emissions by 160,000 tones (4.3 million tonnes CO₂ equivalent per annum) contributed to winning the project a National Energy Award 1995 in the Production / Conversion of Energy Category.

One of the main elements of the award judging focussed on those projects that have taken energy issues beyond good management into new areas of innovation.

The Appin Project was also awarded the prestigious Premier’s Award for Environmental Excellence in the New South Wales Minerals Industry.

More importantly, this power is generated by disposing of a waste product – hence it is referred to as “Clean Power”

A cleaner atmosphere leads to more benefits

In addition to the environmental gains, other benefits resulting from the Methane Energy Project include:

- The ability of the mines to operate from an independent electricity source. This has safety benefits in the event of an emergency evacuation of the mines,

- The effective use of an otherwise wasted energy resource,

- The provision of cheaper and more reliable electricity to consumers,

- The creation of up to 30 new jobs now that the plants are in operation (more than 60 jobs during construction), and

- The technical, commercial and contractual proving of a process that will have further applications in the mining industry and the general community.

Considerable interest has been shown in the project throughout the coal industry and the community in general. Experience gained from the Methane Energy Project will encourage further use of waste methane for power generation in mining and other industries. The use of such gas for small and large scale electricity generation may apply at other coal mines – particularly where deep, gassy seams are confronted.

The power partners

BHP Collieries Division initiated the project and also supplies gas to the operation

BHP Collieries Division is based in Wollongong, New South Wales, employing a workforce of approximately 1,700 and operating five underground coal mines. Using modern long wall mining techniques, the Division mines in some of Australia’s best reserves of coal, producing more than six million tonnes of clean coking coal and one million tonnes of energy coal each year.

The Appin power partnership energy developments limited

Energy Developments Limited is an integrated energy company which develops, owns and operates power generation and power transmission projects. Energy Developments Limited operates Power Stations with a total installed capacity of over 200 megawatts in South Australia, Victoria, New South Wales, Queensland and the Northern Territory, including power plants fuelled by natural gas, landfill gas and coal seam methane. The Company undertakes the initial design and construction and commissioning as well as the final operation of these plants.
Lend lease infrastructure

A subsidiary of the Lend Lease Corporation, Lend Lease Infrastructure is both a developer and operator of power and water infrastructure projects in Australia. Lend Lease Infrastructure has approximately 165 megawatts of generating capacity under development or completed. Some of these are directly managed and operated by Lend Lease Infrastructure, including hydro-electric power stations supplied from existing irrigation dams in New South Wales and Western Australia. These projects provide significant environmental benefits and regional economic development to our communities. The Lend Lease Corporation is able to utilize its considerable resources to provide specialist risk assessment, financial structuring and project management expertise.

GAS UTILISATION

Introduction

This section considers some of the technical issues in utilising Coal Bed Methane. It attempts to cover all of the important issues to consider and then to explain the logic and the reason for the success of the EDL solution.

Understanding the fuel source - methane drainage plant

This plant provides the primary source of coal bed methane. This fuel source is characterised by large variations in volume and quality. The variations tend to occur over a reasonably long time frame associated with the mining activity. Below is a typical specification of the mines gas from Appin and Tower.

<table>
<thead>
<tr>
<th></th>
<th>Minimum</th>
<th>Maximum</th>
<th>Average</th>
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<td>Total Flow M³/sec</td>
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<td>4.4</td>
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<tr>
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<td>45</td>
</tr>
<tr>
<td>Carbon Dioxide %</td>
<td>0</td>
<td>12</td>
<td>3 Note 1</td>
</tr>
<tr>
<td>Higher Hydrocarbons</td>
<td>0</td>
<td>1</td>
<td>1 Note 2</td>
</tr>
<tr>
<td>Air</td>
<td>20</td>
<td>60</td>
<td>51</td>
</tr>
</tbody>
</table>

Note 1 Appin has experienced some CO₂ but these numbers are relatively low compared to Westcliff.

Note 2 Higher hydrocarbons are not significant in the CBM drained from underground. The surface boreholes that collect gas from the Bulgo Sandstone overlying the coal measures can contain significant levels of higher hydro-carbons (4-8%).

Mine ventilation air

The mine ventilation air differs from ambient air in the following ways that are relevant to utilisation.

- Vent Air temperature approximately 10°C lower than ambient
- Vent Air Methane Content 0 to 1% CH₄
- Vent Air Dust loading is significant
- Vent Air moisture content is very corrosive

Natural gas

Natural Gas is used as a supplementary fuel source and can be considered a very clean source of methane. It is worth noting that the ethane content varies to maintain the specified heating value.

Safety considerations

As the methane drainage process and the mine ventilation system are all driven by improving mine safety it is critical not to compromise the safety of the mine whilst utilising the gas. There are five main safety issues to be aware of.
Flammability limits

The upper flammability limit (UFL) of methane air mixture increases with pressure and temperature as defined below:

Pressure Variation of UFL

\[ \text{UFL} = 14. + 20.4 \log P(\text{atm}) \]

Temperature correction defined as

\[ \frac{\text{UFL} \text{t}^\circ C}{\text{UFL} 25^\circ C} = 1 + 0.000721 (t - 25) \]

The above formula provides a UFL of 37.8% for a pressure of 9 atmospheres and temperature of 200°C.

This factor is very significant in considering utilisation technology. In gas turbine applications it is necessary to compress the fuel for injection into a combustion chamber. The increasing UEL means it is unsafe to compress beyond a reasonable pressure for a known methane concentration. This fact eliminates the aero derivative style gas turbines and some of the very highly efficient industrial gas turbines.

Interface with the methane drainage plant

Methane Drainage plants are a very important part of the safety equipment at the mines and are designed to “fail safe” and always provide a path for gas from the mine. This occurs in a plant shutdown or blackout by by-passing gas from the suction side of the plant to atmosphere via a non-return valve. In an operational mode this is achieved by having multiple stacks and “fail open” control valves to ensure a path for the gas.

The interface to the utilisation plant must preserve the by-pass facility for shutdowns and blackouts. It must also collect the gas in such a way that should the utilisation plant stop a path is always available for the methane drainage outlet. This can be achieved with tiered pressure control loops and appropriate valves.

Interface with the mine ventilation air

This interface must also be accomplished without compromising the mines safety. The major requirements of this interface are:

- Never to impact on the ventilation fan performance by adding back pressure.
- Never to utilise the ventilation air if the CH₄% is too high or following a ventilation failure.
- To let the colliery management establish when ventilation is normal and when it is acceptable to utilise ventilation air.

Safety devices

All fuel sources must continuously be monitored to ensure they are safely above the UEL or safely below the LEL. These systems are duplicated by the Colliery and the utilisation plant.

As a back-up, flame arresters are installed at appropriate place to prevent propagation of flames. It is essential to have a thorough understanding of flame arresters so that their installation guarantees their performance rather than ensures their inadequacy.

Explosion protection

Clearly the processing of coal bed methane establishes many hazardous zone of various levels and the equipment needs to be explosion protected for the relevant zones. In Appin and Towers case the plants have been designed to comply with the Coal Mine Regulations and the Australian Standards as well as the AGL gas codes.
Gas turbine versus gas engines

In considering which type of equipment to use there are several factors which need to be considered:

Ventilation air

Physically it is possible to direct ventilation air into either a gas engine or a gas turbine. In a gas engine the air stream’s sole purpose is to mix with the fuel to provide the correct air fuel ratio for the combustion process. In a gas turbine not all of the air from the compressor is used as combustion air. Part of the air is used for cooling and it mixes with the exhaust having by-passed the combustor.

Utilising the ventilation air concerns gas turbine manufacturers for two reasons.

- The environmental emission problems resulting from the cooling air circuit mixing with the exhaust.
- The particulate and chemical contaminant that could easily attack the exotic metals in the blades.

Currently no manufacturers have operated gas turbines on ventilation air. Energy Developments is planning to trial such technology in 1998.

Plant efficiency

Efficiency is normally improved by either increasing temperature or pressure or both.

Thus gas turbines with high efficiencies have higher pressure ratios and as discussed earlier cannot be safely applied to an air-methane mixtures.

The turbines that can be safely used on air-methane mixture have a pressure ratio of 20:1 or less and a typical efficiency of 30-35%. A gas engine can match these efficiencies.

The most important aspect to consider with the efficiency is the part-load efficiency. As the mine gas quantity varies it is not possible to continually provide maximum fuel requirements (unless an undersized plant is installed and surplus gas is vented to atmosphere). The part load efficiency of a gas turbine is quite poor compared to a gas engine and this is significant in the selection process.

Compression

To inject the fuel into a gas turbine significant expense is required for compression equipment. This adds both to the capital cost (= 40%) and the parasitic load of the Power Station. Gas engines if they are low pressure engines require no compression equipment. High pressure gas engines require relatively low cost and low pressure compression equipment.

Tolerance to fuel variations

This gas engine has greater tolerance to low methane concentration and high carbon dioxide concentrations than does a gas turbine. Typically a gas engine can operate down to a composition of 35% CH₄ and 35% CO₂ and a gas turbine would be limited to 50% CH₄ and 20% CO₂. This can be a very significant consideration as the Westcliff Gas Turbine has not operated for approximately two years due to their high CO₂ levels.

Maintenance

There is no doubt there is more maintenance on reciprocating engines than on gas turbines. The maintenance costs are also higher but the maintenance tasks can be scheduled so as not to interfere with revenue production. The maintenance of a gas turbine also involves more skilled trades persons and when it occurs it will involve a significant down time.
Flexibility and redundancy

This consideration is more relevant to the number of units and the unit capacity rather than the technology. There is no doubt that the Appin Tower project has significant advantages in design redundancy.

ECONOMICS & STATISTICS

The utilisation of C.B.M is a very essential environmental initiative and should also be economically viable.

The viability of such projects is influenced by the following:

- Mines Gas Availability
- Electricity Tariff
- Natural Gas Tariff
- Power Plant Efficiency
- Power Plant Availability
- Power Plant Capital Cost
- Power Plant Operating Cost

Mines gas availability

Mines Gas Availability is certainly a very critical factor. Ideally the mine should continuously supply the full fuel requirements of any utilisation plant. As the CBM production varies with mining conditions it is necessary to manage the gas availability by selectively collecting gas form boreholes that are not essential to mine safety. Such management produces a fairly uniform supply of gas and importantly gas is not vented to atmosphere. If a mine is not prepared to manage the gas resources then either shortfalls of mines gas will reduce revenue or surplus production will be vented to atmosphere reduce the environmental benefits.

Electricity tariff

The recent changes in the electricity industry have provided competition in the industry and resulted in lower electricity prices. The implementation of a competitive bidding process for generators has resulted in very low generation or pool prices. This relatively low price is expected to remain whilst supply comfortably exceeds demand. The current pool price would cover operating and fuel costs but would not service the capital investment. This low price is therefore not sustainable in the long term.

As far as utilisation of C.B.M is concerned it is a particularly difficult time to enter the market unless it is a small project which does no export power. In such cases the avoided cost of not purchasing electricity would improve the economics.

Natural gas tariff

The Natural gas tariff is obviously only relevant if Natural Gas is used as a supplementary fuel. The deregulation of the natural gas industry should result in a reduction of natural gas prices.
Power plant

Appin and Tower Power Station have an annual availability of greater than 98% and maintain an efficiency of approximately 35%. This excellent performance maximises the revenue to the project. The performance of the engines on mines gas has been good and the operating costs are below budget.

Viability From the miners perspective

The economic viability for the mine owners is dependent on a reliable supply of CBM. The quality needs to be within specification and sufficient quantity to fully fuel the power station. The returns to the mine are also influenced by the length of contract and the type of guarantee linked to gas supply.

It is possible to structure a project so that the mine owner can recover all or a substantial part of the underground methane drainage costs.

ACKNOWLEDGEMENTS

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In-seam Drilling Technologies for Underground Coal Mines

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ABSTRACT

There are several in-seam coal research projects currently under investigation by the Cooperative Research Centre for Mining Technology and Equipment (CMTE). The overall aim of the projects is to produce new technologies to enhance safety and the efficiency of in-seam gas drainage drilling. Results from high pressure waterjet rotary drilling trials indicate that by applying high pressure water to a rotary drill bit, the hole can be drilled to follow the planned trajectory very closely. Steerable drill bits for drilling long holes (>1 km) have been produced and trialed. A flexible high speed drilling system provides a capacity for rapid in-seam drilling, particularly suited to cross panel drainage. Geophysical tools are being developed to provide a geological steering capability and a better understanding of potential gas outbursts zones.

INTRODUCTION

Established in 1991, the CMTE is recognised internationally as a centre of excellence with a track record for developing and delivering new technology to the Australian mining industry. The Centre has research programs in new mining and drilling systems, geological sensing, automation and design and reliability. Its research partners are The University of Queensland, Sydney University, Curtin University and CSIRO. The following Australian mining companies, equipment manufacturers and mining contractors are members of the Centre: BHP Coal Pty Ltd, Hamersley Iron Pty Ltd, Mining Technologies Australia Pty Ltd (MTA), Pasminco Australia Ltd, Technological Resources Pty Ltd, Shell Coal Ltd, WMC Resources Ltd, Aberfoyle Ltd and Advanced Mining Technologies (AMT).

Developing improved methods for in-seam drilling for gas drainage in underground coal mines has been highlighted by the industry and the CMTE as a research priority. The drilling related research includes a number of projects on in-seam gas drainage from coal seams including a flexible high speed drilling system for cross-panel gas drainage, high pressure waterjet rotary drilling for straight cross panel holes, steerable high pressure waterjet rotary drilling for long holes, tight radius drilling from the surface to access coal seams for pre-drainage and horizon sensing and logging of in-seam boreholes.

COAL SEAM METHANE

The presence of methane within coal seams represents a significant hazard in underground coal mining. Prior to mining, the gas may need to be drained from the seam using drill holes drilled using one of two approaches. One is rotary drilling, in which torque is applied at the hole collar and transmitted to the drill bit along the drill string. The other is downhole motor (DHM) drilling. Here, as the name implies, a motor is located in the hole immediately behind the drill bit to provide the torque required for drilling. Holes drilled using DHM employ a bent-sub assembly which allows the drill head to be steered. The advantages of rotary drilling are that the capital cost of the drilling equipment is much less than for DHM and the rate of drilling is considerably greater than for DHM. The principal disadvantage of rotary drilling is that, at present,

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the drill cannot be steered during the drilling operation. A significant fraction of the holes that are drilled using the rotary method fail to reach their intended destination which results in the need to redrill these holes. The importance of drilling rate in the cost effectiveness of the operation must be balanced with the ability to drill the holes in the places where they are needed.

The methane in coal seams is also a potential energy resource. In Australia there are a number of coal basins close to the major population centres of the east coast. These coal basins are estimated to contain in excess of $10^{12}$ m$^3$ of methane (Paterson, 1990). Coal seam methane is easy to find and prove but comparatively expensive to extract. Currently the cost of coal bed methane, AU$5-7/GJ, is at least double the cost of natural gas, AU$2-3/GJ (Davis, 1995). The main reason for the higher extraction cost for coal seam methane is a lack of applicable economic drilling techniques. This is particularly true for Australian coal seams which generally have lower permeability and higher horizontal stresses than the US coal seams. Well enhancement techniques such as hydrofracing from a vertical well have not been economically successful in Australian coal seams. Conventional horizontal drilling from surface holes is also not cost effective as there are severe limitations in accessing multiple seams from a single vertical well. Improved drilling techniques for gas drainage in coal mining will also have an impact on the extraction of coal seam methane as an energy resource in its own right.

**SUMMARY OF CMTE GAS DRAINAGE DRILLING PROJECTS**

**High pressure waterjet rotary drilling**

In an effort to improve longwall productivity and address current safety issues associated with methane drainage, the CMTE has been investigating the applicability of high pressure water (20 to 40 MPa) for rotary drilling at both Appin (BHP Coal) and Dartbrook (Shell Coal) mines. The project builds on previous experience obtained under NERDDC (National Energy Research, Development and Demonstration Council) funding (Kennerley, 1993). The main objective of CMTE's work is to drill straight cross panel gas drainage holes and long gas drainage holes along the length of longwall blocks.

**High pressure waterjet rotary drilling for straight cross-panel gas drainage holes**

This project was supported by the Australian Coal Association Research Program (ACARP project no. C5028) and both Appin and Dartbrook Mines. The ultimate aim was to drill straighter and more accurate in-seam cross panel holes for methane drainage at a productivity greater than that achievable by DHM drilling technology.

A high pressure water pump (250 l/min at 80 MPa), suitable for use in the underground coal mine environment, was designed and manufactured for the drilling trials. As in Kennerley's work high pressure BQ drill string was used. Three types of high pressure drill bit (drag bit, PCD bit and pineapple bit) were tested during the field trials. Diamec 252 and Diamec 262 drill rigs were used. Phase 1 underground trials were conducted at Appin Colliery (Dunn, Liu and Stockwell, 1997). Subsequent tests (Phase 2) were at the Dartbrook Mine. All holes drilled were surveyed with an Eastman single shot camera (either pumped down or post-survey).

The following are the main findings:

1. High pressure waterjet rotary drilling has demonstrated significantly improved hole straightness over conventional rotary drilling. The average hole deviation, for the holes drilled by a PCD drill bit, was 7.6 metres. This was less than a quarter of the deviation of conventional rotary drilling (Fig. 1).
2. High pressure waterjet rotary drilling significantly increased the penetration (up to 80% improvement in instantaneous penetration rates). Feed pressures were substantially reduced to around a quarter of that required for conventional rotary drilling and torque pressures were reduced to around 80%.

3. The nominal coal cuttings size (50% passing) were reduced (Fig. 2). This may facilitate the cleaning of hole cuttings.

4. High pressure waterjet rotary drilling consumes about same amount of water as conventional rotary drilling (approximately 150 l/min) and less than DHM drilling (220 l/min).

Fig. 1 - Hole deviations for conventional and waterjet
Fig. 2 - Coal cuttings size distribution

High pressure waterjet longhole rotary drilling

Given the improved hole straightness and reduced feed and torque pressure forces, together with reduced cutting size, the extension of the cross panel drilling work into longhole drilling clearly has some potential. A project financially supported by the Australian Coal Association Research Program (ACARP project no. C6028) and with significant support from Dartbrook Mine has been established.

The project builds on the previous project by attempting to develop steerable drill bits which can be used in conjunction with the existing downhole equipment. The objective is to drill in-seam holes (>1000 m) along a longwall block. Drilling longitudinally along a coal block could increase the time available for gas drainage and improve equipment access in longwall roadway development. Three types of steerable drill bit were designed and manufactured, and trialed at Dartbrook Mine:

1. PCD drill bit with an off central jet,
2. PCD drill bit with a free rotational waterjet cutting nozzle (Fig. 3), and
3. PCD bit with a bent sub.
Fig. 3 - Off-Centered FR nozzle

The field trials were conducted with the same drilling equipment used previously. Tests were undertaken using high and conventional water pressures with all three types of bit. Holes were drilled across panel and also into a pillar to observe damage caused to the coal by drilling. At this stage a problem of the trending to the floor has not been completely resolved. There are four current findings:

1. Holes are straighter using high pressure water (this supports the earlier finding from the previous underground trials).
2. Rotary drill holes using the high pressure down-hole assembly and drill string trend towards the floor at conventional water pressure as well as at high water pressures.
3. The coal was not damaged as much by the high pressure water as was thought during previous underground trials.
4. PCD drill bits appear to be very aggressive. This may limit the potential for steering rotary drills with this type of drill bit.

High speed high pressure waterjet cross-panel drilling

The high speed high pressure waterjet cross-panel gas drainage drilling system uses a self-propelled cutting nozzle and flexible hose to drill holes in coal. The ultimate aim of this project is to develop a drilling system which can drill considerably faster than current rotary or DHM drilling methods. The system does not require the making and breaking of rod connections. The technique of the self-propelled waterjet cutting nozzle comes from the tight radius drilling (TRD) project being conducted by CMTE in conjunction with BHP Coal. The self-drilling device uses high pressure water forced out of the back of the drilling head to propel the device forward. Cutting waterjets at the front of the drilling device, in a self-spinning head arrangement, excavate the coal in front of the drilling head.

The first prototype drilling assembly was a retro-jet and self-rotating cutting nozzle assembly for non-directional drilling developed by Kennerley (1993). Subsequent trials were carried out into a coal seam exposed in a highwall of an open cut mine (Trueman et al, 1995). The system used a pressure of 60 MPa and a flow rate of 150 l/min. It drilled up to 100 m (drilling distance was limited by the length of hose available) with penetration rates from 0.3 to 2.5 m/min (Fig. 4).
The drilling assembly has been redesigned for greater pressure (115 MPa) and flow rates (234 l/min) and their number of retrojets has been increased in order to significantly increase thrust. These modifications resulted in a significant improvement in drilling performance. A 100 m borehole typically took 50 minutes to drill and the best rate of penetration was achieved when 194 m was drilled in 42 minutes. The drilling assembly also has a tendency to remain within a narrow band of the coal seam. We are currently addressing the guidance (steering and surveying) of such a drill assembly for use underground (ACARP funded research).

Sensing and logging for in-seam boreholes

For the drilling of in-seam boreholes for coal seam gas drainage and exploration to be more effective, there also exists the need for information on geological conditions, the orientation of the drill bit and the position of the hole within the seam. While tools are available to determine the orientation of the hole, geological conditions can only be established on the basis of drill cuttings and drill performance. Seam position needs to be established by periodically deviating holes to check for the location of the coal seam roof. Neither of these are satisfactory solutions.

To provide a means of testing for the position of the hole within the seam, CMTE has designed and built a proof of concept radiometric tool (Hatherly et al., 1996). Test results from the West Cliff Mine, Fig. 5, show that it is feasible to guide a downhole motor drill on the basis of the radiometric profile in a coal seam. The radiometric profile is related to the ash profile in the seam and allows for the possibility of tracking a borehole's position without the need to get close (30 cm) to the roof or floor before readings can be taken.

Current work to develop a production tool is being funded by ACARP and is being undertaken in collaboration with Dr Ian Gray of SIGRA. The intention is to develop a combined tool which will reside permanently behind the drill bit and provide:

1. a survey capability,
2. the ability to monitor the position of the hole with respect to the roof and floor,
3. the ability to monitor drill thrust, torque and rpm, and
4. the option to add additional geophysical sensors such as resistivity and sonic

Fig. 5 - Four boreholes drilled at West Cliff Colliery show increased material gamma counts as the holes deviate away from the mid-seam position.

The tool will be modular and be able to communicate through the drill string via the cable systems currently available or via a cableless system currently under development at SIGRA. A users interface will be provided.

To infer geological conditions CMTE in collaboration with CSIRO Division of Telecommunications and Industrial Physics is developing borehole radar and dielectric techniques (Hatherly et al., 1996). We have designed and built probes for HQ and NQ size drill rods. These tools can also be used to provide information on the location of the roof/floor (Fig. 6). The basic design uses radar centre frequencies of about 500 MHz. Initial tests of the dielectric tool suggest that it might be able to give a ready indication of the existence of mylonite zones intersecting a borehole. This is also the subject of on-going ACARP funded research.
CONCLUSIONS

As Australia's coal mining trends towards underground operations and deeper coal seams, gas drainage is becoming an increasingly prevalent and costly component of mining. CMTE's drilling research is directed at improving the efficiency of gas drainage by developing faster, steerable drills equipped with the sensing technology to allow them to remain on path within an undulating coal seam. There are a number of significant technological challenges which need to be addressed but through the combination of fundamental research and field trials these are being progressively solved. Current targets are to:

1. perfect a waterjet steerable drill bit to enable the drilling of in-seam drainage boreholes along a significant proportion of the length of a longwall panel,
2. overcome the tendency of holes to drop towards the floor of the seam,
3. continue the development of the high speed, high pressure waterjet cross panel drilling system,
4. implement seam following and geological logging sensors for use with DHM drilling,
5. integrate these sensors with the various waterjet drilling technologies.

Commercialisation of these developments is being pursued through collaboration with mining companies, drilling suppliers and contractors and actively contributing to industry meetings of drill operators.

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Inseam Drilling For Gas and Exploration - Recent Advances

J Hanes

ABSTRACT

In-seam drilling is conducted mainly to drain coal of gas and/or water prior to development or extraction. The requirements of in-seam drilling for gas drainage have mainly surpassed the technical limits of rotary drilling and are mainly being addressed by guided down hole motor drilling, albeit at a higher cost. There is an increasing application of in-seam drilling to detect geological structures or other hazards or their absence in advance of mining to reduce the risks of underground mining. Close collaboration between miners, drillers, geologists, researchers and suppliers over the last four years has seen many advances made to improve the accuracy and reliability of in-seam drilling. With such developments will come closer control over the ventilation and safety of future mining under deep gassy conditions, but not without costs. The paper summarises the current applications of in-seam drilling, recent technological advances and the next generation survey and geophysical sensing tools which are currently being developed.

INTRODUCTION

The majority of in-seam drilling is conducted to drain coal of gas and/or water prior to development or extraction. However, an increasing amount of drilling is directed at detection of structures or proving their absence, and testing of gas contents especially in seams prone to outburst. With tightening of the economics of underground coal mining, what mine can afford to intersect unexpected geological structures or to experience outbursts on development? Inseam drilling provides insurance by reducing the risks.

In modern in-seam drilling, guided down hole motor drilling is tending to replace rotary drilling. Rotary drilling involves rotation of a string of drill rods which extends from a stationary drill rig to the drill bit. The drill rig applies both rotation and thrust to the rods. Downhole motor guided drilling involves advancement of the hole by rotation of the drill bit by a hydraulically activated motor which resides close behind the drill bit. The drill rig applies thrust to the rods, but generally not circulation.

Until recently, gas drainage holes were mainly drilled as rotary holes of around 90mm diameter drilled from one set of gateroads, across the proposed longwall panel and beyond the next proposed gateroads. They were therefore typically around 250 to 300m long, but with the development of guided drilling technology in recent years, some holes are being extended to cover two or three longwalls. Drainage holes are typically drilled in fan patterns or as parallel holes at 15 to 20m spacings. Rotary holes are seldom straight and the resultant curve can mean that they do not cross the next gateroad panel. At most mines, all gas drainage holes are surveyed after drilling to determine where they go. Some of the holes close after drilling and cease to drain the coal. Sometimes when rotary holes intersect a geological structure, the bit bogs and cannot penetrate the structure. The more powerful the drill rig the lesser is the chance of the bit becoming bogged, and of the driller detecting a structure.

Downhole motors and some form of hole surveying are used to drill directionally controlled holes for exploration and gas drainage. The longest hole in coal to date is 1670m, drilled at Moura by Ponti Drilling from a highwall face. The longest hole drilled in an underground mine is 1538m drilled in a West Virginia (USA) mine by Advanced Mining Technologies (AMT). Although large faults and most dykes can be detected in these holes, it is difficult or impossible to detect small structures.

Consulting Geologist
Small rotary rigs are popular for drilling relatively short holes for detecting structures or for taking cores for gas content testing. The currently used bottom hole assemblies result in curvature of holes and often unknown trajectories.

Within the next few years the major improvements to in-seam drilling should come through the recognition and location of geological structures while drilling directionally controlled holes using a combination of behind-the-bit monitoring of torque, thrust and RPM, radiometrics, radar and capacitance along with reliable surveying. Use of these tools plus maintenance of borehole stability and testing of gas contents while drilling will be facilitated by borehole pressurisation. Recording of data will be facilitated by intrinsically safe computers. Survey while drilling rotary holes should be possible while modification of the torque, thrust, RPM device should enable recognition of structures. Drills will be monitored to assist recognition of structures. There is an immediate need to adopt drill monitors. The development of geosteering tools with incorporated geophysical probes, combined with mathematical modelling of bottom hole assembles and drilling parameters should provide a great leap forward to in-seam drilling technology.

There are still some problems to be addressed including communication of data out of the hole and straighter drilling of rotary holes. The former need to be addressed for successful application of the tools being developed now. The latter is being addressed by the mines and by the water jet assisted drilling project being conducted by The Centre of Mining Technology and Equipment (CMTE) under Australian Coal Association Research Projects (ACARP) funding.

Most of the major in-seam drilling problems identified by mine operators are being addressed and should be solved by developments which have occurred or commenced in the last four years, some as ACARP funded research, some as mine site initiatives and some by suppliers in response to industry requirements. Support by ACARP for a Co-ordinator of in-seam drilling research has facilitated these developments though improved communication between all players and minimisation of duplication.

THE CURRENT SITUATION

In 1993, around 300 km of rotary drilling and 160 km of guided drilling was conducted annually in seam in Australian collieries. In 1996 around 120 km of rotary drilling at around $30 per metre and 400 km of directional drilling at around $60 per metre were drilled. In 1997, the guided drilling distance increased to around 460 km while the rotary drilling remained at 120 km. A conservative estimate of cost for this work is $3.6M for rotary and $27.6M for directional drilling. In addition, around 400 km of cross measure drainage holes are drilled annually using rotary drilling. There is a trend towards replacing much of the rotary drilling with directional drilling to achieve better accuracy or at least to know where the holes go.

Rotary drilling technology has changed little during the last 20 years except perhaps for the introduction of the lightweight ProRam drill. With the recently introduced very stringent requirements of proving outburst-prone coal is safe to mine has come the need to survey drainage holes to prove that they have reached target. Survey is currently conducted using a single shot photographic survey tool or in some cases a multishot photographic or multishot electronic survey tool. The survey tool is pushed down the hole after drilling, either on PVC pipe or on the drill rods. Either method is prohibitively time consuming. Survey of boreholes is now accepted as necessary for all holes in outburst prone mines as not knowing where a hole goes is nearly as useless as having no hole at all.

Directional drilling with downhole motors is revolutionising in-seam drilling. Holes can be drilled straighter but at a lower penetration rate than with rotary drilling. The use of survey tools is critical during drilling to maintain hole trajectory. In 1993, the only options for surveying were a single shot photographic tool or the Dupont electronic survey tool which resided behind the bit. The single shot tool must be pumped down the rods, a photograph exposed, the tool withdrawn and the film developed before the attitude of the bit is known. The Dupont tool communicated with the collar of the hole via a sonic pulse which was satisfactory in holes which did not intersect stone or structures and which did not contain gas bubbles. Now, in response to industry requirements, there are two electronic survey tools available which reside behind the bit. The Mecca system supplied by AMT now uses a solid communications cable installed in the rods to transmit bit attitude while the rods are stationary and is used by many mines. The Drill Scout supplied by Surtron Technologies is a true measure-while-drill (MWD) tool and communicates data out of the hole via a single strand wire cable fitted to a mandrel.
There is currently no in-hole tool available for detecting small geological structures in boreholes during or after drilling. Longhole drilling requires that the seam roof be intersected periodically (such as each 50m to 80m) to provide horizon control. This necessitates a pullback of the hole and deflecting (drilling of another branch).

The popular drill rods used today (eg BQ, NQ) are not capable of sustained use for very long (+1500m) holes (Gray,1992). The rod joints do not have adequate tensile strength, especially for the heaviest duty of withdrawal from long snaking holes.

**REQUIREMENTS FOR THE NEAR FUTURE**

The need is increasing for drilling patterns to confidently drain gas to below levels which can cause outbursts or unsafe emissions, while detecting and accurately locating all geological structures which are significant to mining. Economics demand that these demands be met at a reasonable cost.

Rotary drilling is generally favoured for routine gas drainage drilling because of its cost advantage, but drilling accuracy must be improved. A robust survey tool located behind the bit which can either provide accurate survey data during drilling or on retrieval of the rods is required. Such a tool would, at least, allow rotary drilling to be used without the time consuming need for post drilling survey. If hole trajectories are irregular enough to leave gaps in the drained coal, directional drilling could be used to target the undrained areas. Better directional control of rotary drilling or replacement with a more accurate drilling method at a similar cost is desirable.

Detection and accurate location of geological structures in the hole are prime requirements of both rotary and directional drilling. There is a need to accurately identify structures varying in magnitude from mylonite 1cm thick to large faults. There is also a need to detect and define changes in gas parameters along the hole. Detection of structures during drilling is desired to allow prompt action for maintenance of borehole wall stability.

There is a need, especially during the drilling of directionally controlled holes, to know where the bit is located with respect to the boundaries of the coal seam. Currently the intersection of stone in a hole is confusing as it takes considerable effort to determine if the stone is roof, floor, a band, a fault or a dyke. If stone is intersected, it might be necessary for the rods to be partly withdrawn and a new branch of the hole commenced.

There is a need to record drilling parameters in each hole to provide accurate data for detection of geological hazards and to maximise drilling efficiency. The current method of manual recording is too haphazard and needs to be replaced by automated recording, data processing and drill control.

**FINDING SOLUTIONS**

The Exploration Taskforce of the Australian Coal Association (ACA) initiated an ACARP funded scoping study of in-seam drilling research requirements in 1993. This study defined the industry’s research requirements which have been addressed by research projects funded in 1994, 1995, 1996 and 1997. The research is coordinated by the Taskforce with the author acting as the Taskforce’s research coordinator. Coordination is minimising duplication, assuring the research is directed and facilitating communication among all players.

Commercially available survey tools are improving and are partly satisfying industry’s current needs. Tools to detect geological structures during drilling and the means of controlling the borehole environment to permit use of these tools are currently being developed under varying levels of ACARP support. To enable use of geophysical probes during drilling, the AGA Consortium (AGA) project on Pressurisation of Boreholes was approved. A borehole collar pressurisation system was developed to allow drilling of holes under a fluid pressure which is sufficient to prevent gas desorption during drilling. This pressure should help maintain borehole stability and will provide a water-filled hole for use of geophysical logging tools. AGA have incorporated a sampling vessel for accurate determination of gas desorption pressure of cuttings samples thus promising a rapid assessment of the potential for outburst. Lunagas was funded to design a system for detecting
changes in gas make during drilling and the resultant gas recovery system is being considered for trials by Lunagas and AMT.

Tools to detect structures are being developed. Funded by ACARP, AGA are developing detectors of drilling parameters including bit torque, load and RPM which will be installed behind the bit initially in rotary drilling bottom hole assemblies and later in directional drilling. The tool will incorporate a survey device being commercially designed by Sigra Pty Ltd. Changes in coal strength indicated by the sensors should be correlatable with geological structures. Australian Coal Industry Research Laboratories (ACIRL), with ACARP funding, has developed a prototype caliper tool for detecting changes in borehole diameter associated with structures. Partly funded by ACARP, CMTE developed two tools for structure and roof/floor proximity sensing. These are a radar and a radiometric tool. The radar successfully located the roof and floor proximity, but has been temporarily shelved because of the lack of an electromagnetically transparent drill rod housing. The radiometric sensor can only detect the roof or floor boundary when within 40cm of the stone, but shows potential for recognition of the radiometric signature of parts of the coal seam and should therefore be applicable for horizon control of the bit. The radiometric sensor will be incorporated with the AGA sensors and the Sigra survey tool plus other geophysical sensors to produce a drill guidance system. The CMTE and CSIRO tested a dielectric capacitance tool over a mylonite zone at West Cliff Colliery and demonstrated its ability to delineate the mylonite quite clearly. ACARP is funding further development and proving of this sensor. AMT, the developers of the MECCA survey system, are considering the incorporation of geophysical sensors into their popular survey tool to enable detection of roof/floor proximity and structures. BHP Research with ACARP funding, developed an intrinsically safe portable computer for underground use and used it to collect data from a monitored Proram rotary drill rig. It provides data on changes of drilling parameters at the drill rig with drilling of varying geological structural conditions and allows identification during drilling of geological structures and gas inrushes. There is a desire to automatically monitor drilling parameters on all rigs to reduce the human elements of error and lack of interest in detailed data recording and to maximise useful data capture.

Consequent to the Moura disaster inquiry, two valves to allow automatic closure of gas pipelines or flow of gas from holes are being developed under ACARP funding. They are designed to automatically stop gas flow in any occurrence of abrupt change of gas pressure within the gas drainage circuit as would occur during an explosion. Both units should be commercially available in 1998.

The downhole sensing tools being developed will have to communicate their data to the drill collar. The AMT MECCA survey system does this via a cable-in-rod system and the Drillscout survey tool communicates via a single core cable. Other communication systems are also being considered. AGA completed an ACARP project which aimed to develop standard electrical and mechanical connections for downhole tools. Because of the wide diversity of requirements, standardisation is not really practical, but the project provided useful guidelines. Data communicated out of the hole will have to be stored and processed and geologically interpreted.

As the problems of hole surveying, roof/floor proximity sensing and drill monitoring are overcome, there will be a need to drill longer holes to beyond 2000m to explore and predrain proposed longwall blocks. Gray (1992) showed that currently used rods will not be suitable for such long holes. With further ACARP support, Sigra have built a drill rod test rig and will be testing various drill rod joints to assess their suitability and to facilitate design of more appropriate joints. Both Lama (1995) and Gray (1992) emphasised the need to introduce more science into drilling if longer holes are to be successful.

To improve straightness of rotary holes, the mines have conducted their own site-specific trials of various bottom hole assemblies and communicated their results to colleagues at regular drill operator and supervisor meetings. Most major mines have opted for the improved accuracy of guided drilling. Only one company successfully uses rotary drilling for long holes, employing down hole motor drilling to correct major deviations. It appears that the drilling of long holes by rotary drilling is only successful when used by patient expert drillers. CMTE and BHP funded initial trials of waterjet drilling and waterjet assisted drilling at Appin Colliery to compare these types of drilling with conventional rotary drilling. ACARP funding was approved for extensions of the project in 1996 and 1997 to increase the productivity and reliability of waterjet assisted rotary drilling and development of direction control methods. This method offers some promise for improving accuracy of rotary drilling.
CONCLUSIONS

Finding solutions to in-seam drilling problems requires a multifaceted approach. ACARP funding of research into in-seam drilling totals around $500,000 per year for 1994, 1995, 1996 and 1997. This funding alone is insufficient to address all problems nor to find solutions to the many day to day problems experienced by the mines. The funding allows investigation and initial development of solutions for medium term challenges. The mining companies and drilling contractors put a lot of effort into solving their immediate problems, assisted by their suppliers and they share their achievements with other operators in the industry. Researchers funded by ACARP typically contribute funding to accelerate their projects. Suppliers are funding their own developments and assisting some of the research projects. Progress on all fronts is communicated at regular meetings of all players organised by the author as in-seam drilling research coordinator.

The ACARP Exploration Taskforce, industry and service providers are showing that industry challenges can be successfully addressed when all players are involved to firstly define the priorities, then to find the solutions. Regular communication between researchers, suppliers and industry assures that the projects are focussed and that minimal duplication of effort occurs.

ACKNOWLEDGEMENTS

The willing cooperation and sharing of information by all my in-seam drilling colleagues in the mining industry, suppliers and researchers should ensure development of successful solutions to the industry's in-seam drilling challenges.

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A New Approach in Planning Gas Drainage Practices

L W Lunarzewski

ABSTRACT

Both safety and productivity in underground gassy coal mines can be improved substantially if an appropriate gas management system is introduced. A strong direct relationship exists between the gas emission rate, roof and floor strata relaxation zones characteristic, mining activities and gas drainage practices. Detailed knowledge of geological factors, gas and coal-rock properties, as well as mining systems are necessary for the methodology used in predicting overall underground gassiness, and planning gas capture and ventilation systems. Early gas emission calculations for various mining activities, particularly for the longwall mining system are essential. A reasonably accurate prediction of gas make as well as the design of ventilation, gas recovery (drainage holes) techniques can be made using 'Floorgas' and 'Roofgas' computer simulations, provided sufficient geological, mining, and gas data are made available.

STATE OF THE ART

Several techniques for predicting gassiness during longwall coal production have been developed. Many are only relevant to regional mining, gassy and geological conditions. Most methods adopt the same basic parameters: the stratigraphy above and below the seam, insitu gas contents of the working and adjacent seams, the strata relaxation coefficient, and the degree of gas liberation. The accuracy of the final results, however, depends substantially on individual coefficients, which are developed specifically for the above techniques, and therefore cannot be reused in other mines or regions (countries). Comparative test results, using various coefficients, have shown that large errors are possible in these methods, within the range of -50% to +120% (Dunmore, 1979).

The advent of computer modelling methods, and particularly finite element techniques, enables predictions based on the nature and extent of the relaxation zone surrounding the long wall to be made when using local geomechanical, geological and mining input data.

Such a model has been developed and evaluated at Lunagas Pty Limited, Newcastle, Australia, under the name of 'Floorgas and Roofgas Geomechanical and Gas Release Models', and has been commercialised to operate on a Windows based platform.

Outputs from both programs (Figs. 1, 2 and 3) are used to design gas capture technologies, including cross measure or directional holes drilled from underground, and gas wells drilled from the surface. They are also used for precise gas make predictions and assessment of gas conditions necessary for planning gas management strategies.

Gassiness predictions based on the 'Floorgas' and 'Roofgas' models were compared with underground gas measurements from high production longwalls in Australia and other countries between 1992 and 1997. The results are sound, indicating accurate predictions of 90 to 95% efficiency, if the appropriate input data are available.

'FLOORGAS' AND 'ROOFGAS'

The programs can be used as engineering tools in the underground mining areas for:

1. prediction of gas emissions to the underground workings,
2. design of gas drainage holes and gas capture strategies,

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Principal Consultant and Managing Director, Lunagas Pty Ltd.
3. ventilation planning in underground coal mines,

4. planning for strata control,

5. definition of shearing zones and vertical loads, and

6. planning for coalbed methane utilisation and environmental control. (Lunarzewski, 1992; Lunarzewski et al., 1995).

When using the ‘Floorgas’ and ‘Roofgas’ programs, the following benefits are achievable:

1. Identification of gas discharge and shear zone positions in the floor, strata relaxation angles and various gas discharge zones position in the roof, which are both relevant to local geological and mining conditions.

2. Calculation and definition of prediction coefficients such as specific gas emission and the relationships between gas beration, longwall width, and coal production levels.

3. Optimisation of cross-measure drainage hole length, location, number, and angles of deviation and inclination.

4. Optimisation of in seam drainage hole locations, the number and direction.

5. Assessment of active gas resources and their contribution in the pollution emission process.

The programs generate graphical outputs typified in Figs. 1, 2 and 3, depicting vertical cross-sections of longwall strata at various selected distances ahead of and behind the longwall face. ‘Floorgas’ output displays extend down to 100m below the working seam, while a ‘Roofgas’ output can show strata relaxation up to 200m above the working seam.

The ‘Floorgas’ program generates a vertical load distribution function along the chain-pillar and adjacent longwall panels/goaf areas. On the basis of the generated load, the program calculates vertical and shear stresses for each cubic metre of rock element being modelled. The program is a product of long-term detailed analyses and it uses results gained from various coal mines world-wide, with particular reference to mining and geological conditions in coal mines with daily productions greater than 5000 tonnes per longwall.

‘Roofgas’ calculates the position and shape of five gas release zones, with various degassing intensities, using the Lunagas empirical model (Lunarzewski, 1992). Then boundaries are quantitatively defined both in terms of discontinuous deformation of rocks and gas release percentages.

**INPUT DATA**

Input Data is assembled from three (3) defined sources.

1. Geomechanical in which
   * Mechanical properties of the roof strata are expressed as
   * uniaxial compressive strength (UCS) values and
   * Strata stresses expressed as horizontal and vertical stress magnitudes, horizontal to vertical stress ratio, and horizontal stress direction related to longwall axis position.
2. Lithological - A geological (lithological) description and/or sonic velocity log of the strata for 'Floorgas' (100m below the seam), and 'Roofgas' (100m or 200m above the working seam).

3. **Mining and Gas**
   
   a. Longwall block and pillar geometry;
   
   b. Longwall advance rate or coal production level;
   
   c. Coal seam positions in the strata; and
   
   d. Gas flow or emission characteristics, if available.

**OUTPUT PRINTOUTS**

Both programs generate scaled colour outputs on the basis of local geological, geomechanical, and mining data, in relation to longwall face position (Figs. 1, 2 and 3).

'Floorgas' printouts (Fig. 1) are prepared for one-half of the longwall face width, and show vertical cross-sections through the strata parallel to the longwall face, down to 100m below the seam. Both normal and shear stresses are calculated for each point within the dimensions of the model and are represented by a colour scheme defined in the legend. Printouts show strata relaxation and gas emission variations for nominated distances behind and ahead of a longwall face with reference to either a start line or drainage hole position. Every printout shows strata relaxation and gas emission zone shapes corresponding to the relative level of vertical stresses expressed in megapascals (MPa). Optional maingate and tailgate views are possible and each represents only one-half of the longwall width, with the other half regarded as symmetrical.

'Roofgas' printouts (Figs. 2 and 3) are prepared for one-half or full scale of the longwall face width. They show vertical cross sections through the strata parallel to the longwall face up to 200m above the seam. The printouts are prepared for nominated distances behind the longwall face with reference to a start line or drainage hole position. Every printout shows strata relaxation and gas emission zone shapes corresponding to the relaxation range and percentage of gas release, based on the Kidybinski Sequential Bed Separation Principle (Kidybinski, 1990), and Lunagas' empirical gassiness prediction model (Lunarzewski, L, 1992).

**DRAINAGE HOLE DESIGN**

The program can be used most specifically for the precise design of underground drainage holes and surface gas wells, as well as identifying the strata relaxation, gas release, and shearing zones (Figs. 1, 2 and 3). Cross-measure hole positions in relation to the above mentioned zones, drilled and planned lengths, deviation and inclination angles can be applied directly to the printouts on the screen by use of the drainage hole tool. The user can draw an image of the hole on the cross-section, allow for changing hole positions, while the program simultaneously displays parameters according to the varying position. In seam hole positions in relation to the above mentioned zones, planned numbers and lengths can also be defined and designed, using strata relaxation gas release zone characteristics from 'Floorgas' and/or 'Roofgas' outputs.

Accurate planning and optimisation of gas drainage holes may significantly increase the volume of captured gas, thus reducing the cost of drilling unsuccessful holes.
Fig. 1 – Floorgas simulated output, 0m behind the face
ROOFGAS SIMULATION
300m behind the face

Colliery: XXX
LW/Panel: LW XX (200m x 1500m)

Fig. 2 - Roofgas simulated output, 300m behind the face
Fig. 3 - Surface gas well optimisation utilising roofgas software
The programs are written for a Microsoft Windows system and can be run with only the following minimum requirements: 486 IBM PC (or compatible), 8 megabytes of RAM, and Microsoft Windows operating system version 3.1 or later. 'Floorgas' and 'Roofgas' incorporate a graphical interface that assures easy and efficient data input, clear and concise outputs and design tools that improve accuracy and accelerate the design process.

An on-line help function includes all information on how to run the program, and gives examples and explanations of how to use the software for engineering purposes.

CONCLUSION

The 'Floorgas and Roofgas Geomechanical and Gas Release Models' define the relationship between coal mining activities, strata relaxation and gas release-emission phenomena. Both programs are the most advanced engineering numerical tools available which improve the accuracy and quality of gas control, gas drainage, and ventilation systems design, as well as assess environmental air polluting emission rates from underground coal mines.

'Floorgas' combines a precise rock mechanics analysis with gas conditions to calculate stress and gas release zones in the floor strata of a working coal seam. 'Roofgas' can generate a roof strata break-line as a boundary between continuous and discontinuous rock masses. All methods published hitherto deal with continuous or discontinuous rocks separately with no possibility of indicating the boundaries between the two zones.

Gas control strategies including gas management planning, ventilation and gas capture system requirements can be defined and designed for specific local conditions, when utilising 'Floorgas' and 'Roofgas'.

Any world renowned gas drainage technology or practice can be optimised or programmed using outputs and results from 'Floorgas and Roofgas Geomechanical and Gas Release Models'.

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Development, Application and Potential of a Real-time Return Gas Monitoring System

M I Slater and R J Williams

ABSTRACT

This paper presents an overview of the design and development, application to date and potential of a real-time return gas monitoring system.

Initially developed as an ACARP/Industry funded project, whose prime aim was to create a 'turn-key', prototype, stand alone, real-time, Return Gas Monitoring System (RGMS) capable of providing quantitative assessments of gassiness levels on a shift by shift basis, enabling unusual gas emission patterns to be readily flagged.

GeoGAS designed and commissioned the RGMS and 'live-trialled' it at Tahmoor Colliery and Dartbrook Mine. The system has a number of features new to Australian underground coal mines and is comprised of three distinct components:-

1. Hardware high accuracy CH₄ and CO₂ gas analysers, air velocity meter, belt weigher, PLC, modems and computer.

2. Real time software - SCADA based control, trending and data logging software.

3. Offline software - GeoGAS's RGMS data post-processing software implemented to perform the calculations and data reduction, facilitate analysis, and present results.

The system, currently monitoring longwall return conditions at Dartbrook Mine, has been in continuous use since February, 1995. Areas of potential application include:-

• Provision of an additional safety barrier in quantifying gassiness levels.

• Rationalisation of drilling & drainage operations.

• Post-analysis of real events.

• Support of operators and staff.

Introduction

In January 1994, GeoGAS Pty. Ltd. was awarded an ACARP grant (Project 3076) to research the development of a real time, Return Gas Monitoring System (RGMS) over a three year period.

The prime objective was to create a turn key, prototype, stand alone, real time, gas monitoring system capable of providing quantitative assessments of gassiness levels on at least a shift by shift basis and enabling unusual gas emission patterns to be readily flagged.

Sub objectives were to:

1 GeoGAS Systems Pty Ltd, Wollongong, NSW
- Define and document the process of using these data to back analyse gas drainage effectiveness.
- Assess the potential for using the technique to quantitatively define seam gassiness on a sub shift period basis.
- Define indices relating the gas emission response (rate of emission, peak emission rate, quantity, composition) to outburst proneness in terms significance to outbursting.

The project scope covered design and fabrication of the gas monitoring system, trial of the system at Tahmoor Colliery and Dartbrook Mine, and development of reporting procedures and associated software to facilitate its real world application.

**System Development**

The GeoGAS RGMS is comprised of underground and surface located hardware (Fig. 1), Scaleable Architecture Data Acquisition (SCADA) software, and data post-processing software.

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**Fig.1 - GeoGAS return gas monitoring system - layout**
Underground hardware

The system cabinet contains a mix of electrical and mechanical components, some intrinsically safe (IS), but most non-IS. It must be operated within non-hazardous conditions at all times. If the PLC detects CH₄ levels at or above 1.25% for 10 seconds it sends an alarm flag to the surface PC and then shuts the system down, isolating power to the interlocked door switch. Major underground hardware items include:-

- Two UNOR 610 gas analysers (CO₂ and CH₄).
- One Allen Bradley SLC 500 Programmable Logic Controller (PLC).
- One Control Systems Technology IPM-10 belt weigher monitor.
- One Mikan short-haul modem.
- One Trolex TX1322 vortex shedding air flow sensor.

A separate gas bottle cabinet houses two calibration gas bottles

- One “DS” size Alpha standard span gas.
- One “D” size zero gas (High Purity N₂).

The system PLC maintains autonomous control of the underground system accepting input data from the gas analysers, belt weigher controller and air velocity sensor every 2 seconds. It maintains continuous interactive communication with the surface CITECT monitoring PC, buffering then forwarding data.

The PLC program features:-

- Scaling of analog inputs under full auto-ranging analyser output.
- Calculation of moving averages of analog inputs to allow damping of signal levels, in particular, fluctuations in the air velocity signal.
- Totalised coal production, as recorded by the belt weigher, on per shift, per day and year-to-date basis.
- Control of the auto-calibration procedure for the gas analysers. The PLC is programmed to calibrate the gas analysers at an automated interval (weekly), or under command from the surface PC. Pressure sensors in the Zero and Span gas lines allow the PLC to indicate fault alarms when the calibration gases are spent.
- Alarm monitoring of CH₄ and CO₂ concentration levels.
- Decoding of gas analyser control signals to enable text display in the surface CITECT PC, of the current status of the gas analysers and calibration gas bottles.

A formal risk assessment of the system was conducted in December 1994 prior to the underground installation of the equipment at Tahmoor Colliery. It’s aim was to ensure the system incorporated sufficient designed safeguards against hazardous operations in underground environments and recommend any further safeguards deemed appropriate.

Above ground hardware

- One IBM Compatible PC running SCADA and data post-processing software.
- One Mikan short-haul modem.
**SCADA software**

The operator's interface with the RGMS is provided by the CITECT for Windows (v3.4) SCADA package. This software hosts communications between the surface PC and PLC in real-time. It enables user interaction for logged data, graphically trended results and privileged user access control for PLC settings, alarm set-points and acknowledgements.

Logged data from the SCADA software (CITECT) provides values and trends for:

- $\text{CO}_2$ flow rate (l/s).
- $\text{CH}_4$ flow rate (l/s).
- $\text{CO}_2$ concentration (%).
- $\text{CH}_4$ concentration (%).
- Air velocity (m/s) and air quantity (m$^3$/s).
- Production rate.

**Data post-processing software**

GeoGAS's RGMS data post-processing software was implemented in 32 bit Windows Pascal (Borland Delphi 2.0). It accesses ASCII or dBase files generated by the SCADA package and provides a means for calculating, checking and reporting the gas emission data.

At the completion of a shift the software is used to differentiate background from production related emission levels (on a shift by shift basis) and calculates the following face area emission :-

- Peak $\text{CO}_2$ emission rate (l/s).
- Peak $\text{CH}_4$ emission rate (l/s).
- $\text{CO}_2$ quantity (m$^3$)
- $\text{CH}_4$ quantity (m$^3$)
- $\text{CO}_2$ gas make (m$^3$/t).
- $\text{CH}_4$ gas make (m$^3$/t).

The software defines a process for calculating the gas emission indices and assessing the validity of the data. In addition to quantifying the gas emission, the software compares the results to historical readings and provides a rating of the emission (abnormally high, normal higher than average, normal lower than average, abnormally low).

A number of methods were devised and tested, to automatically set background levels. The 'Rate of Change' algorithm was applied. It involves :-

- Mapping the distribution of each gas flow value ($\text{CO}_2$ l/s and $\text{CH}_4$ l/s), against the rate of change between values (the actual change over a moving average of 5 consecutive readings, divided by the elapsed time in seconds). The background emission level in l/s is that corresponding to the minimum rate of change value (Fig. 2).
- Integrating the shift’s total emission and subtracting the background component.
Fig. 2 - Analysis of rate of change

System application

Tahmoor Colliery

Tahmoor Colliery agreed to provide a site for trial and financial assistance. 510 Panel was the first project field site (Fig. 3).

The trial was conducted over an 8 month period from 13th February 1995 to the 13th October 1996. Actual monitoring took place for the months of June, July and August with the balance in commissioning, decommissioning and production delays.

The Bulli seam is mined to a height of 2.2 m. The virgin gas content is around 12 m³/t at 80% CO₂ and 20% CH₄, but within 510 Panel had been pre-drained to less than half this value with the gas composition at 73% CO₂ and 27% CH₄.

Analysis of the data highlighted rhythmic fluctuations in air velocity as a significant source of noise. Rapid swings of about 5% were evident in air velocities (in this case equating to 45 m/s to 43 m/s). These transient changes were not reflected in corresponding CO₂ and CH₄ gas concentration readings, resulting in a higher level of 'noise' in the CO₂ and CH₄ gas flow results. The noise is partially controlled by the PLC's data averaging and moving average settings. There is further scope to adjust these settings.

The belt weightometer stand, originally configured for West Cliff Colliery, proved unsuitable for the belt structure at Tahmoor Colliery and was damaged. Production data in both trials was subsequently sourced from colliery records.

During the monitoring period, 510 Panel mining was periodically delayed when gas content tests did not achieve the required threshold value. The maximum shift gas content plots would therefore be indicative of the threshold values that could be applied using the RGMS (Fig. 4). While more data and further analysis is required in defining threshold values for the system, it does give an indication of how thresholds may be applied to the continuously monitored data.
The Tahmoor Colliery trial succeeded in quantifying and characterising the gassiness on a shift by shift basis for coal that had been pre-drained to below the threshold limit for outburst alleviation.

For constant levels of inherent gassiness, the gas make varies with production (Fig. 5).

While production in 510 Panel was subject to an outburst management plan, and proceeded only below prescribed gas content threshold, a potential application of this system would be development of additional threshold criteria based on shift gas make.

**Dartbrook Mine**

Dartbrook Mine provided a site and financial assistance. G101 Panel was selected as the second project field trial site (Fig. 6). The system was committed to the Dartbrook trial for a total of 11 months, from 13th October 1995 to the 14th September 1996. Installation was delayed initially by 3 months while Dartbrook drafted site-wide specification for belt weightometer equipment and its installation.

Continuous monitoring took place from mid April to mid September. The data set related to production within the panel, spans the period 25th April to 17th July 1996. In this time G101 panel developed from inbye 18 cut through to its termination at 25 cut through in essentially undrained gas conditions.

Dartbrook’s gateroad developments mine to a height of 3.9 m from an essentially continuous 25 m sequence of coal. The gas content of the coal monitored averaged 7.8 m$^3$/t (Q1+Q2+Q3 at 20°C) with a composition of 75% CO$_2$ and 25% CH$_4$. 

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**Fig. 3 - Tahmoor monitor site**
Fig. 4 - Shift gas quantity (CO2+CH4)

Fig. 5 - Shift gas make
Most mining conducted within the monitored period was 'conditional'. Production was halted and face drainage instituted twice as a result of gas levels in cores taken ahead of development. High face gas levels on development interrupted production in June and the panel was terminated in mid July shortly after connection of the Longwall 1 installation roadway.

As in the first trial, air velocity data introduced noise (up to 5%) in raw emission rate determinations. Of more concern was drift in the air velocity response. Stone dust from the airflow and adjacent stone dust barrier racks progressively fouled the sampling chamber of the velocity head, drifting the indicated value by approximately $0.005 \text{ m/s}$ per day.

The trial succeeded in quantifying and characterising the gassiness on a shift by shift basis for coal that had not been gas drained. The background emission accounted for 80% of the panel emission (contrasting with just 46% at Tahmoor). This is due to the large gas reservoir being mined. Gas quantities generated in the face areas were around 5 times higher than those seen at Tahmoor.

Transient gas events related to the intensive drilling and drainage program at Dartbrook were seen most days. No production related 'abnormal' emissions were detected.

The system logged and processed shift-based data for over 93% of the shifts when power was available. In the absence of power interruptions, the first trial showed the equipment, once properly installed and commissioned, to be robust and reliable. In this trial only four data shifts were lost, resulting from dust loading of the air velocity sensing head.
System potential

Provision of an additional safety barrier

There is a renewed focus on control of hazards associated with gas emissions by implementation and application of mine specific management plans. Direct real-time return gas monitoring, with subsequent analysis, provides an important additional barrier.

New mines, especially those in Queensland where depth (and gas content) are progressively increasing, need to ensure mining is carried out in an environment of zero dynamic gas incidents. Conditions will change with depth. In addition to measures (such as the GeoGAS Desorption Rate Index) aimed at defining when to take action to alleviate outburst potential, return gas monitoring should prove to be an important additional barrier.

A real life application for return gas monitoring occurred at South Bulli Colliery as response to a small outburst in a cindered zone. At one stage, preparations were under way to transfer the RGMS equipment from Dartbrook Mine, but it was decided to adapt the mine’s own gas monitoring system to this application.

GeoGAS accessed South Bulli’s data files daily via modem. Daily reports and analysis were provided to the mine on the level of gassiness encountered. As a guide, a draft threshold established for Tahmoor Colliery was utilised. Emission levels remained low.

The incident provided a ideal example of the potential application of the system. The South Bulli Colliery cinder zone was so fragmented, that gas content cores could not be taken and in-seam drilling for gas drainage was very difficult. Return gas monitoring became the only real option in assessing gassiness during mining. As rudimentary as South Bulli’s system was, it did provide data (in a timely manner) which the colliery then incorporated in management of the hazard.

Rationalisation of drilling and drainage operations

In existing mines with functional outburst management strategies, data provided by the system should allow fine tuning of gas drainage system planning and operation. There is potential for the increase of respective gas content threshold levels without compromise in safety.

GeoGAS has been involved in assessments aimed at raising management plan related gas content thresholds in some mines (on the basis of an inherently low gas desorption rate as measured by the GeoGAS Desorption Rate Index, Williams 1997). Return gas monitoring is an additional barrier, directly measuring the mechanisms involved, and raising confidence in the determinations made.

Post analysis of real events

The return gas monitoring system enables the best possible back analysis, quantification and diagnosis of a gas dynamic event. Two such events were captured by the RGMS at Tahmoor Colliery. It was initially scheduled for installation at South Bulli Colliery, but sent to Dartbrook Mine as part of the gas management effort required there. The system would have been ideally placed and utilised at South Bulli Colliery.

Support of operators and staff

Underground operators frequently discern increased (or decreased) gassiness. Timely access to return gas monitoring data can aid in understanding the environment, alleviate concerns, and facilitate effective responses.
CONCLUSIONS

The system’s control, communications and gas analysis components have proved responsive and robust. Improvements can be made in the air flow sensors, and re-specification of the belt weigher equipment is required to provide reliable coal production rate data.

The system has demonstrated it’s utility in actual gas dynamic incidents and capabilities in gassiness determination within development panels. Additional areas of application have been identified.

The system has increased (and increasing) potential application; today more than when engendered in 1994.

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The Nature of Underground Heating as Indicated by Numerical Modeling

D Humphreys

ABSTRACT

A numerical model of the spontaneous combustion of coal has been developed and is being used to investigate the nature of self-heating in underground coal mines. Results to date confirm the complexity of the self-heating reaction in coal and illustrate the impact of the factors such as the coal reactivity, mass and airflow through the coal pile. Subtle differences in the chemical, physical and environmental conditions and make a significant difference in the development of a high temperature self-heating or a low temperature benign condition, the subtlety escaping normal underground observations. Additional work is planned to investigate the thresholds for such factors in the development of spontaneous combustion and its implications for detection by gaseous products.

INTRODUCTION

Spontaneous combustion of coal in underground coal mines continues to represent one of the most significant hazards to the safety of workers in the mines in which they are employed. Many sources of ignition of methane gas have been removed from mines over the years, through their identification, and elimination by exclusion or changes in mining practices. These are generally associated with the mechanical aspects of mining coal or the ventilation of the underground workings. However, while ever coal mining takes place, crushed coal will be exposed to air and, under the right circumstances, spontaneous combustion can occur possibly leading to an open fire. If methane is present, a very dangerous situation can arise.

Inadequate detection methods and inappropriate control techniques can exacerbate the dangers of spontaneous combustion. Late detection means that a heating will be more advanced (hotter and/or larger) with the attendant greater risk of igniting methane. Inability to evaluate the state of a heating can easily lead to the use of inappropriate control techniques when the heating is too far advanced.

Explosions of methane, attributed to ignition by spontaneous combustion, have resulted in the deaths of 43 miners and the loss of three underground coal mines in Queensland over the last 25 years. Contributing factors in all three cases were inadequate detection and control techniques, much of which can be traced to a poor understanding of the nature of spontaneous combustion in underground coal mines. Very little is known regarding the nature of underground heatings and their controlling factors apart from rare anecdotal information on actual heatings, the results of gas analysis used in detection and the qualitative extrapolation from small scale laboratory tests. Some large scale self-heating tests have been carried out but with the emphasis on understanding of the stockpiling situation and are, due to their expense, limited in number and range of conditions studied.

A means of overcoming these difficulties is to develop a numerical model that may be used to examine a wide range of circumstances and conditions. Numerical modeling has been used in the past to examine the nature of spontaneous combustion in coal stockpiles but generally only over a limited range of coal types, particle size distributions, and configuration. The principles

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of modeling spontaneous combustion in stockpiles, however, can be adapted to understanding spontaneous combustion in underground coal mines to explain many aspects of this phenomenon.

To this end a number of numerical models that simulated self-heating of coal under conditions of forced ventilation, as occurs in underground heatings, have been developed and continue to be refined. Some of the factors that will be examined in the study will include the inherent reactivity and moisture content of coal, the geometry and mass of reactive coal as well as the rate of airflow through and permeability of the heating site. These will be related to in-situ conditions such as coal rank, the degree of fracturing and compaction of the heating site, and the pressure differentials required to cause heating. Modeling will also be used to predict the nature of the off-gas from a heating and used to evaluate current detection techniques such as Graham’s ratio and carbon monoxide make.

This paper will discuss the basic understanding of spontaneous combustion, the basis of the models being developed and preliminary results obtained to date.

THE BASICS OF SPONTANEOUS COMBUSTION

The phenomenon of spontaneous combustion is not limited to coal but is known to occur in a number of other materials such as charcoal, cattle feed, fertilizers, hay, cocoa beans, manure, oil-soaked rags (Haessler, 1989). No matter what material is involved, the basic principles that govern spontaneous combustion are the same in all cases. Spontaneous combustion is the process by which heat is generated spontaneously (without external initiation) within a substance by some reaction, under conditions that prevent the dissipation of the heat to the environment. Under these circumstances, the temperature of the reacting solid will rise leading in turn to an increase in the rate of reaction and greater heat generation. Left unabated the accumulation of heat can lead to the ignition of the solid reactant.

The self-heating reaction

The nature of the reaction that produces the heat that may ultimately lead to ignition depends upon the nature of the substance under consideration. For example, spontaneous combustion in hay bales is initiated by bacterial activity until the temperature reaches about 75°C. Beyond this point the rate of aerial oxidation is greatly increased and continues to cause the temperature to increase, possibly to ignition.

In the case of coal, the source of heat is low temperature oxidation with oxygen from air. While the heat generated by this reaction is very little at ambient temperatures, under the right circumstances this heat may accumulate and cause the temperature of the coal to rise. The specific chemistry of the coal-oxygen reaction is very complex, not well understood, and need not concern us here. However, it is necessary to understand the factors that affect the rate of oxidation and the rate of heat generation that takes place in the coal.

The main factors involved are described below:

(i) The oxidation reactivity of a coal pile, \( k \), is determined by the inherent oxidation reactivity of the coal and its particle size. The inherent oxidation reactivity varies considerably from coal to coal, largely dependent upon the rank of the coal. In general low rank coals tend to be more reactive than high rank coals under the same set of conditions. The rate of oxidation is also a function of particle size being inversely proportional to particle size down to some threshold beyond which further size reduction has no effect. The combined contribution of inherent reactivity and particle size results in an overall oxidation reactivity for the coal in question, which might be regarded as the pile oxidation reactivity, \( k \).

(ii) The temperature of the reaction, \( T \). Many studies have shown that the temperature dependence of the rate of oxidation can be described by the Arrhenius formula (Haessler, 1989) given by:
\[ \frac{dq}{dt} = \text{rate of oxidation (expressed in appropriate units such as gm O}_2 \text{kg coal/min)} \]

E = the reaction activation energy,

R = Boltzmann's gas constant, and,

T = the absolute temperature.

The effect of temperature on the oxidation rate is to cause an approximate rate doubling for each 10°C rise.

(iii) Accumulated oxidation, q. Coal, like many other substances, exhibits a reduction in oxidation rate as oxidation proceeds at a constant temperature and oxygen concentration (Carras and Young, 1994). This is described by the Elovich equation given by:

\[ \frac{dq}{dt} \propto e^{-aq} \]  

q = cumulative oxidation per unit mass of coal, and

a = a constant.

(iii) Oxygen concentration, O₂%. As the oxygen concentration decreases the rate of oxidation also decreases for a constant temperature and degree of cumulative oxidation (Schmidt and Elder, 1940). This is best described by the equation:

\[ \frac{dq}{dt} \propto \left( \frac{O_2 \%}{20.93} \right)^N \]

O₂% = oxygen concentration by volume(%),

20.93 = oxygen concentration in normal air by volume(%), and,

N = constant (approximately 0.6).

(v) The heat of oxidation, \( \Delta H_{ox} \). The reaction that takes place between coal and oxygen is exothermic. The heat of oxidation is the amount of heat released per unit of oxygen adsorbed. In many of the previous studies it has been assumed that the heat of oxidation was the same for all coals and approximately equal to the heat of combustion or about 14250 J/gm oxygen adsorbed. Recent investigations by Taraba (1994) suggest that this is not the case. The heat of oxidation at temperatures of about 25°C appears to vary from about 3100 to 9300 J/gm O₂ depending upon the rank of the coal. Lower rank coals appear to have higher heats of oxidation. Further, the heat of oxidation is also a function of temperature and must, at high temperatures be equal to the heat of combustion. These effects must be considered in the modeling of spontaneous combustion.

The overall rate of heat generation due to oxidation can be expressed in the combined equation:

\[ \frac{dH}{dt} = k \times \Delta H_{ox} \times \left( \frac{O_2 \%}{20.93} \right)^N \times e^{-\frac{E}{RT-aq}} \] 

where k = the pile reactivity (for a given coal and particle size), and, other variables are as defined above.

This equation can be used to calculate the rate of heat generation by oxidation at any point in a coal pile at which the oxygen concentration, temperature, coal pile reactivity and accumulated oxidation are known. The rate of oxidation (consumption of oxygen) is given by:
For the purposes of examining the production of off-gases, similar equations can be developed for the rate of production of gases such as carbon monoxide and carbon dioxide.

Other sources of heat or heat transfer

While oxidation is the primary source of heat that drives the spontaneous combustion process, there are other sources of heat and heat transfer that play an important role in the development of a heating. Remembering the conditions for spontaneous combustion to occur, that some of the heat is retained in the solid causing its temperature to rise, it is clear that means of heat loss play an important part. Even though the rate of oxidation and heat produced may be very high, if it is all dissipated to the surroundings there will be no temperature rise and no self-heating.

Other possible forms of heat or heat transfer that can occur in a coal pile are those associated with the wetting and drying of coal, convective heat transfer between the coal and air, conductive heat transfer through the coal and convective heat transfer at the surface of the coal pile. All of these need to be considered in attempting to examine the self-heating behavior of coal.

The least well understood but most important of these is the wetting and drying of coal. Drying of coal is an endothermic process (requiring heat) which will affect the heat balance in an oxidizing pile of coal. The effect of drying will be to reduce the heat available to cause self-heating. The corollary is that wetting of coal is an exothermic process and will tend to accelerate self-heating. Whether or not wetting and drying can be regarded as a purely physical process is not clear as there some evidence that the presence of moisture can alter the inherent rate of oxidation. As a physical process wetting and drying can be determined from the moisture isotherm for a coal (Allardice, 1991) which relates the equilibrium vapour pressure of the surrounding air to the moisture content of the coal.

Convective heat transfer within the pile is controlled largely by the airflow through the coal and will be less significant at low flow rates than at high flow rates and convective heat losses at the pile surface are controlled by the pile surface temperature. Conductive heat losses are determined by the temperature distribution in the pile. All these heat transfer processes are dependent on the temperature distribution and geometry of the reacting coal pile. Stott et al (5) provide a succinct statement of the thermal equation applicable in one dimension being:

\[ \frac{dq}{dt} = k \times \left( \frac{O_2 \%}{20.93} \right)^{N} \times e^{-\frac{E_{act}}{RT}} \]  \hspace{1cm} (5)

\[ \frac{dM}{dt} = \frac{dH}{dt} \]

\[ \frac{dT}{dt} = \frac{C_s \rho_s}{A} \left( \frac{d^2T}{dx^2} - C_s \rho_s \nu \frac{dT}{dx} \right) dx + \Delta H_{O_2} \frac{dN}{dt} + \Delta H_{H_2O} \frac{dM}{dt} \]

\[ \frac{dM}{dt} - V \frac{dT}{dx} \]

I = Rate of heat change in element dx causing the coal to self-heat,
II = Rate of conduction heat transfer into element dx,
III = Rate of convective heat transfer into element dx,
IV = Rate of heat generation by oxidation in element dx, and,
V = Rate of heat loss by evaporation in element dx,

and where:- subscript \text{s} = coal,
subscript \text{g} = gas,
A = cross-sectional area of element.

\[ \text{COAL98 Conference Wollongong 18 - 20 February 1998} \]
C = specific heat,
θ = density,
T = temperature,
t = time,
x = length of element,
K = thermal conductivity,
V = approach velocity of airflow,
dq/dt = rate of oxidation (as per Equation 5 above),
dM/dt = rate of evaporation of moisture,
ΔH_{ox} = heat of oxidation, and,
ΔH_{ev} = heat of evaporation or condensation.

The combined effects all of all these factors, the inherent reactivity of the coal, its particle size, oxygen concentration, accumulated oxidation, temperature, wetting and drying, convection and conduction, airflow and pile geometry, make the study of the nature of spontaneous combustion very difficult. Because of the vast range of conditions that can occur in a stockpile or underground, no single test can be used to assess the self-heating behavior with any degree of certainty or confidence. Even large scale self-heating tests involving many tonnes of coal can only provide a limited understanding of these complex interactions due to the limited number of tests that can be performed.

With a suitable numerical model it is possible to examine a very wide range of conditions, and the complex interactions involved in a heating. Further, it is possible to examine the production of other gases such as carbon monoxide and carbon dioxide in the numerical simulation, so that the nature of the off-gas from a heating can be assessed against its development. This would provide an opportunity to examine the use of indicators such as CO make and Graham's ratio against the severity of a heating.

**BASIS OF NUMERICAL MODELING**

Numerical modeling is the only way known to take into account the complex interplay between the factors discussed above. To this end a number of numerical models have been developed based on the same ideas. To simulate self-heating, a volume of coal is represented by a series of interconnected nodes. Each node is taken to represent a discrete volume and mass of coal through which air passes and in which oxidation, and therefore heat generation, take place. The mass is assumed to be concentrated at the nodes and all reactions, oxidation, wetting and drying, are assumed to take place at nodes. All heat transfer (convection and conduction) processes occur between nodes. The rate of oxidation at a node is described by Equation 4 and the heat transfer by Equation 5.

A single line of nodes represents a one-dimensional model with air passing from node to node (plug flow). Heat transfer is also from node to node as illustrated in Fig. 1, with no heat transfer perpendicular to the line of nodes. The only heat losses to the environment occur at either end of the model and therefore one-dimensional models are restricted in examining the effects of scale. This type of model simulates the behaviour of a column of coal in an infinitely wide slab of coal. A one-dimensional model has the benefits of relative computational simplicity but is limited in its application to realistic scenarios.
A two-dimensional model can be made to provide for more complex heat transfer processes as shown in Fig. 2. Airflow is still assumed to be homogeneous plug flow from one end of the node grid to the other, but conductive heat transfer can take place across a line of nodes. Convective heat losses can also occur at boundary surfaces other than the end surfaces. A two-dimensional model simulates a slice through a block of infinite width perpendicular to the plane of the nodal grid. Where an axis of symmetry exists across which there is no heat transfer (an adiabat), the nodal grid can be split to reduce the number of nodes in the simulation. The number of nodes required for a two-dimensional model is obviously far more than for a one-dimensional model, but the two-dimensional model is better suited to more complex geometries.
The complexity of modeling increases from one to two and then to three dimensions, but a method has been developed which allows a quasi-three-dimensional model to be developed from a two-dimensional model. This can be done by considering each node as representing a cylindrical shell as illustrated in Fig. 3, rather than a slice of constant thickness. In a homogeneous cylinder there is no heat flow tangentially, only axially and radially. By calculating the area used to determine conductive heat transfer between nodes based on this idea, the basic two-dimensional model can be made to simulate a cylinder of coal of definite dimensions and mass.

The original models developed were Quattro Pro spreadsheets utilizing macros to increment the time steps required to run a model. Each of the one, two and quasi-three dimensional models was a separate spreadsheets. Since the original paper describing these models was published (Humphreys, 1996), the models have been rewritten in Visual Basic for Windows. The separate one, two and quasi-three dimensional models have been combined into a single Visual Basic programme and as far as possible use common code. The main features of the Visual Basic models remain unchanged from the original spreadsheet models as described here, but their functionality and speed have been greatly enhanced. These numerical models are now being used to examine the self-heating of coal with a particular emphasis of understanding the nature of underground heatings.

**PRELIMINARY RESULTS OF THREE-DIMENSIONAL MODELLING**

Some preliminary results from the use of the quasi-three-dimensional model will help to illustrate the capability of the model and some of the areas that will be investigated more fully in time. It is hoped that this model will be able to be used to investigate a wide range of coal reactivities (inherent reactivity and particle size), airflow conditions (related to pressure differentials), mass of coal involved and moisture content of the coal.

The basic output from the model is a history of the temperature of the heating and the composition of the off-gas. An example is shown in Figs. 4 and 5 below. Fig. 4 shows the peak coal temperature as a model heating develops and Fig. 5 shows the basic composition of the off-gas from the same heating. The basic conditions modeled in this case were a cylinder of coal 6m long and 6m diameter, with an airflow flux of 25 l/min/m², and an oxidation reactivity approximately that of a Central Queensland sub-bituminous coal with a particle size of about 25mm. The airflow through the model is expressed as a flux representing the airflow passing through a cross-section of one square metre of the model. It is done in this fashion so that comparison between models of different cross-sectional area is easy and to indicate the levels of airflow required to sustain a heating.
Fig. 3 – Quasi-three dimensional heat transfer model.

Fig. 4 - Development of self-heating
By adjusting the input parameters of the model it is possible to examine their impacts on the self-heating process. For example Fig. 6 shows the effects on the self-heating characteristics of varying airflow through a heating of approximately 200 tonnes of coal in a cylinder 6m long and 6m diameter. The coal pile reactivity is the same as that used for the results illustrated in Figs. 4 and 5, and it can be seen that variations in the airflow can have a significant effect on the development of a heating. At low flowrates (1.25 l/min/m$^2$) the rate at which the heating develops is considerably reduced and the maximum temperature reached by the heating is about 95°C after about 4500 hours. As the airflow is increased the maximum heating temperature increases, thermal run-away can occur and very high temperatures can be achieved.

The reasons for this behaviour can be seen from an examination of the way the heating develops in the numerical model. At low airflows, the amount of oxygen entering the pile is limited and is consumed near the upstream surface. This leads to the formation of a hot spot close to the air entry point. Its maximum temperature is limited by conductive heat losses to the upstream surface and by the limited oxygen supply. As the airflow increases, the hot-spot forms deeper into the pile due to increased convective heat transfer and greater penetration of oxygen. The peak temperature is less restricted by heat losses and oxygen supply and very high temperatures can be achieved.

Variations in airflow

Variations in pile reactivity and size

Clearly one of the most important controlling factors in the development of self-heating of coal is the reactivity of the coal in the pile whether due to its inherent reactivity or its small size. The impact of variations in coal pile reactivity can be gauged from the results shown in Fig. 7. Again the amount of coal is about 200 tonnes. As the reactivity decreases the initial rate of self-heating is also reduced and the time required to reach thermal run-away increases. However, beyond a certain reactivity the maximum temperature rise achieved is restricted and no thermal run-away occurs. The reactivities shown in Fig. 7 are expressed as a proportion of the inherent reactivity of a Central Queensland bituminous coal. These would be equivalent to coal with a particle size of approximately 6.25, 12.5, 25 and 50 mm for 32, 16, 8 and 4% reactivity, respectively.
Although the least reactive pile shown in Fig. 7 did not heat above about 50°C for the model size simulated, it may well do so if the pile was larger. This effect illustrated in Fig. 8, which shows the self-heating curves for coal with a relative reactivity of 8%, at different pile sizes. For a pile 7m long and 7m diameter, the heating reaches thermal run-away at about 1500 hours (63 days). This is slightly increased as the pile size is reduced to a 6m cylinder, but not significantly. When the pile size is further reduced to 5m, the heating does not exceed 75°C, and no thermal run-away occurs. It seems that there is a certain critical mass associated with the development of a spontaneous heating, which is related to the reactivity of the coal involved. In the example shown it is somewhere between and 5 & 6 m cylinder. The critical mass required for a heating will depend greatly upon the reactivity of the coal involved which is determine from the inherent reactivity and particle size.

Off-gas composition and indicators of spontaneous combustion

The composition of the off-gas from self-heating is obtained as part of the results obtained by modeling. From this its possible to calculate various indicators of spontaneous combustion such as Graham’s ratio, carbon monoxide make or carbon monoxide to carbon dioxide ratio. It is also possible to estimate the production of other indicator gasses such as hydrogen and ethane. The composition of the off gases obtained for the 6m cylinder with a reactive reactivity of 8% and airflow of 25 lt/min/m² are shown in Fig. 5. The primary indicators of Graham’s ratio and CO make are shown in Fig. 9. Of most interest amongst these is the development of CO make during the modeled heating. Until the peak temperature exceeds 100°C, the CO make is very low and does not exceed 1 lt/min. This is followed by a period in which the peak temperature increases very rapidly to about 450°C and the CO make reaches about 10 lt/min. During this phase the heating hot spot becomes oxygen limited i.e. all the oxygen passing through the hot spot is consumed, and the hot spot begins to migrate upwind until it encounters the outer edge of the pile. At this stage the peak temperature increases again, and the CO make rises rapidly, in this case to exceed 50 lt/min. Eventually the heating reaches the surface of the pile and would certainly result in open fire. The development of Graham’s ratio follows a similar pattern and results in values well above those normally encountered in an underground heating. However, it must be remembered that the modeling results take no account of the effect of dilution with other airflows, which are inevitable in an underground mine.

Fig. 6 - Impact of variations in airflow

The CO makes observed from the modeling undertaken so far, are similar to those used as trigger points in the industry, but appear
to occur at temperatures which would give cause for concern. From these early results it is not possible to draw any firm conclusions on the relationship between CO make levels and the state of a heating, but this is an obvious area for further investigation.

Fig. 7 - Impact of coal pile reactivity

Fig. 8 - Impact of pile size
CONCLUSIONS

The examples of modeling results given here illustrate the complexity of the self-heating reaction in coal and the main factors affecting self-heating. It can be seen that apparently slight changes in conditions of, say airflow (pressure differential) or mass of coal, can be the difference between the development of a significant self-heating with very high temperatures and a low level, low temperature event. These differences may not be noticeable in an underground mine yet still have a significant effect on the development of a heating.

The main factors considered to contribute to the occurrence of spontaneous combustion in underground coal mines are the reactivity, mass and moisture content of the coal involved and the airflow through the heating zone. It is intended to continue the study of spontaneous combustion using the techniques described here to more fully examine these factors. With additional modeling the occurrence of spontaneous combustion can be described in terms of coal reactivity, mass, airflow and moisture. Additional work will be undertaken to relate airflow to pressure differential which will identify potential heating sites underground.

Finally, an analysis of the gasses produced by the complex mass/temperature distribution will be carried out to relate the traditional indicators of spontaneous combustion to the state of a heating.

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REFERENCES


Predicting Spontaneous Combustion in Spoil Piles from Open cut Coal Mines

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SUMMARY
Spoil piles are produced routinely in open cut coal mines. Spoil piles may contain waste coal and other carbonaceous horizons. Coal and carbonaceous materials react with oxygen in the atmosphere, producing heat. If the rate at which heat is generated is greater than the rate at which heat can be dissipated, the temperature of the spoil pile rises. If the heating remains unchecked, spontaneous combustion can occur. Spontaneous combustion poses significant safety, environmental and economic problems if it should become established in spoil piles. Using the basic features of self heating in spoil, the use of model is directed toward developing quantitative prediction of spontaneous combustion in open cut coal mine spoil piles.

INTRODUCTION
Self heating leading to spontaneous combustion is a moderately common occurrence in coal mining. Many mines have experience with the phenomenon. This can range from the nuisance value of small outbreaks in waste coal through to threats to the safety of mine personnel from spontaneous combustion in underground coal mines. Indeed sometimes in underground coal mines, spontaneous combustion has had disastrous consequences. Similarly in the transportation of coal, great care must be given to the likelihood of heating as fires aboard ships or barges can also prove to be disastrous. In open cut coal mining, the major emphasis has been on the environmental consequences of spontaneous combustion in spoil piles. Safety issues, however, also present themselves as combustion may erode large volumes of spoil at depth leading to subsidence, with or without added surface loads from people or vehicles (Kim and Chaiken, 1993).

While the main cause of spontaneous combustion is the exothermic reaction of coal with oxygen in the atmosphere and while many factors are known to affect the likelihood of spontaneous combustion (such as prevailing winds, cover layers, particle size distribution, amount of reactive material in the spoil) there is to date, no general method which allows quantitative predictions of spontaneous combustion. The work presented in this paper is aimed toward developing quantitative methods which will be able to improve markedly the ability to predict spontaneous combustion.

SPOIL PILES
Spoil piles are associated with open cut coal mining and are a necessary part of the mining process. Large quantities of material, removed to uncover the seam (or seams) to be mined are dumped to form spoil piles which, in general, are made up of broken sedimentary rocks including sandstone, mudstone, siltstone and carbonaceous shales. In addition thin coal seams, which are uneconomic to mine, as well as partings material which can contain appreciable amounts of carbonaceous material can also be dumped. Fig. 1 illustrates an idealised dragline spoil pile and shows a row of individual dragline dumps. The valleys between the spoil piles are often in-filled with carbonaceous material associated with the coal seams.

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Spoil piles are complex structures as they consist of a mixture of materials with differing reactivities to oxygen as well as a broad particle size distribution (Carras et al., 1994). In addition the material distribution in spoil piles depends on the nature of the strata before mining, as well as the method of mining and dumping with each mining method resulting in a different overall layering or mixing of the materials. For instance, a dragline will give rise to a different sequence of layers to a shovel with truck tipping over a face. Segregation of particle sizes also occurs as material rills down the spoil slope. In broad terms spoil piles may be pictured as structures of reactive material which allow the transport of gases and heat, with the gases of main concern in the initiation of self heating being oxygen and water vapour.

**SELF HEATING**

Self heating results when the rate at which heat is generated within a structure is greater than the rate at which the heat can be lost to the environment. In order to provide quantitative descriptions of self heating it is necessary to quantify the heat generation and loss processes. In spoil piles there are four major sources of heat. The most important is the direct oxidation of coal. Oxidation of other carbonaceous strata (eg shales, siltstones and mudstones) also occurs. The third source is the oxidation of reactive pyrite, commonly associated with coal seams. In addition condensation of water vapour within a porous structure (due to changes in relative humidity) can release considerable heat and aid self heating.

**Sources of heat**

1. **Coal oxidation**

The exothermic interaction of coal with oxygen has been studied for many years (Carras et al., 1994; Nordon, Young and Bainbridge, 1979; Nelson, 1989; Krishnaswamy et al., 1996; Chen and Stott, 1997; Khan, Everitt and Lui, 1990). While there is continuing debate as to the details of the reactions and the relative importance of different mechanisms, at −35°C the rate of initial oxidation for coals is ~ 0.1 to −8 x 10⁻⁴ (g O₂/g coal/s), for a particle size range of 0.3−1.7 mm, depending on the coal being considered (Carras and Young, 1994). Further, measurements based on isothermal calorimetry (Nordon et al, 1985) have shown values of the heat of reaction of −250 to 400 kJ/mol O₂. Such rates of reaction and heat generation give rise to thermal outputs of ~ 10⁻⁴ to 10⁻² W/g coal for coal oxidising without oxygen limitation at −35°C.
The rate of reaction has also been shown to depend, inter alia, on temperature, particle size, oxygen partial pressure, water content and extent of previous oxidation. The rate of reaction increases with temperature, Arrhenius fashion, but decreases with time of reaction. In other words, the coal 'ages or weathers'. These latter features of oxidation are shown in Fig. 2 where the rate of oxidation, rate of heat generation and heat of reaction are shown for an Australian coal at 55°C and for a particle size range of -3.3+1.7mm using the experimental methods employed by Nordon et al (1979 and 1985). The measurements cover a time span of 21 days and show the characteristic decline of rate with time. The heat of reaction is also shown and is considered to be approximately constant over the course of the experiment. The noise and drift in the values for the heat of reaction are because of the relatively low reaction rates and thermal powers being measured.

The origin of the decline in reactivity with time has been modelled by other workers as

1. shrinking core
2. an effectiveness factor
3. diffusion control, and
4. an Elovich process.

While these details are still being pursued, the overall features of the curves can be used to create an empirical rate equation for coal where the rate of heat generation can be written

\[ Q = H_0 R_0 f(r) C^n \exp(-E / RT) \exp(-\alpha q) \]

where \( Q \) is the rate of heat generated per unit mass of coal, \( H_0 \) is the heat of reaction, \( R_0 \) is the rate of reaction with oxygen (expressed as gO2/gcoal/s), \( A \) is a constant, \( f(r) \) is a function which depends on the coal particle size distribution, \( C \) is the oxygen concentration, \( n \) is an empirical exponent, \( E \) the apparent
activation energy, R the gas constant, T the temperature, $\alpha$ an empirical factor and $q$ the amount of oxygen that has reacted with the coal. In the above equation the first exponential describes the Arrhenius behaviour, the second the Elovich behaviour.

2. Carbonaceous material oxidation

The carbonaceous materials present in the spoil pile are generally less reactive than coal but there are normally vastly greater quantities of these materials than coal. Also, should combustion begin, these materials can provide a major fuel source for subsequent combustion as their specific energies can be very significant.

The oxidation rates of a variety of carbonaceous materials from open cut coal mines have been measured (Carras et al, 1994). The measurements show essentially the same features as coal oxidation, albeit at a reduced rate. Fig. 3 shows the rate of oxidation (determined at 14 days after the measurement began) for carbonaceous shales from some Hunter Valley mines.

![Graph showing rate of oxidation vs carbon content](image)

**Fig. 3 - Rate of oxidation after 14 days for some Hunter Valley spoil vs carbon content (db)**

The data include materials from both the Greta and Whittingham coal measures (Carras et al, 1994). The data in Fig. 3 show that the rate of reaction (and hence heat generation) is correlated well with the carbon content of the materials. The heat of reaction as determined by isothermal calorimetry has a range of $\sim 290$ to $360$ kJ/mol O, which is similar to the range measured for coals. The rate of reaction of carbonaceous shales has been shown to be able to be described by a similar equation to the one used above for coal.
3. Pyrite oxidation

The third heat source in spoil piles is the oxidation of pyrite. There has been considerable work on pyrite oxidation, particularly as it affects acid generation (Harries and Ritchie, 1981 and Jaynes et al, 1983). In some spoil piles, pyrite oxidation may be a significant contributing factor to self heating and further work is required for the relative importance of pyrite, coal and carbonaceous materials in spoil piles to be quantified from their self heating perspective. In coal stockpiles it is usually unimportant.

4. Water Vapour

Water vapour can play two roles in the heat balance. Changes in ambient relative humidity can lead to the condensation of water vapour in the coal stack. This can liberate significant heat. Similarly the evaporation of water can result in significant heat loss. In addition other workers have suggested that water may play a significant chemical role in the oxidation processes (Chen and Stott, 1993).

PREDICTING SELF-HEATING

The rate of heat generation described above for coal and carbonaceous shales can be used as the basis for the development of a method for predicting self heating. Briefly, the method is based on calculating the relative rates of heat generation and dissipation for a spoil pile. The equations which comprise the numerical model are summarised below (Nordon, 1979; Glasser and Bradshaw, 1990; Saghafi, Bainbridge and Carras, 1995).

1. \[
\sigma \frac{\partial T}{\partial t} + \rho_a C_a v \cdot \nabla T = k \nabla^2 T + Q_o
\]

2. \[
\varepsilon \frac{\partial B}{\partial t} + v \cdot \nabla B = D_b \nabla^2 B - M
\]

3. Momentum

\[
v = \frac{K}{\mu} (-\nabla P + \rho_a g)
\]

4. Continuity

\[
\nabla \cdot v = 0
\]

In the above equations T is temperature, \(\sigma\) is the composite thermal capacity of the medium, k is the composite thermal conductivity of the medium, \(\rho_a, C_a\) are the density and heat capacity of air, V is the velocity of air, \(\varepsilon\) is the pile voidage, \(Q_o\) is the total rate of heat generation and includes the heat of water condensation or evaporation as well as oxidation, B the concentration of oxygen or water vapour in the voids, \(D_b\) the respective effective diffusion coefficient, M the rate of depletion of oxygen through reaction, or the rate of depletion or production of water vapour which may condense or evaporate, P is the air pressure, K the specific permeability, \(\mu\) the dynamic viscosity of air and g the acceleration due to gravity.
In general the thermal conductivity depends on the materials, the packing of the spoil pile and the moisture content. The variation of gas density with temperature is only taken into account in the buoyancy term. The extent to which convection is important depends on the permeability of the spoil pile and on the external pressure distribution. In spoil piles it is the result of barometric pressure changes and the effect of winds. Numerical values for the permeability of spoil piles must at present be assumed.

Simplified versions of the general model outlined by the above equations have been applied to coal stockpiles, spoil piles and goafs. Spoil pile simulations to which the model has been applied include both drag line and truck tipping. The model for spoil allows for the influence of differing voidages through the depth of the pile as well as the presence of fractures which can act as conduits and provide access of oxygen deep within spoil piles (Carras et al 1994).

MODELLING SPOIL PILE SELF HEATING

In this section an example of the use of the CSIRO numerical model to predict spoil pile self heating is presented. Fig. 4 shows the spoil pile to be modelled.

![Fig. 4 - Section of the simulated spoil pile, reactive zone surrounded by unreactive materials](image)

The reactive zone of carbonaceous material is assumed to have been excavated by the dragline and to have formed a symmetrical cone around the initial sandstone overburden. Further inert material has then been dumped over the reactive material. The purpose of this illustrative calculation is to show how the spoil pile heating characteristics depend on the reactivity of the reactive zone. From a practical perspective this region can be considered to be a zone where inert material and reactive coal are mixed due to the mining process.

The spoil pile modelled had a height of 50 m and a base width of 128 m. The reactive zone was considered to extend over a depth of 20 m. The above geometry and materials distributions were used as inputs to the CSIRO numerical model along with values for the reactivity of a Hunter Valley coal and other spoil pile characteristics measured on Hunter Valley spoil piles. The numerical model was used to simulate heating over a 6 month period.

Fig. 5 shows contours of temperature for the case of the zone being made up solely of coal. The temperature maximum ($T_m = 31 \, ^\circ{\text{C}}$) is at the location where the coal can be accessed by oxygen from two sides. Similarly Fig. 6 shows the corresponding oxygen contours. Oxygen is rapidly depleted in the reactive zone.
Fig. 5 - Temperature contours of spoil pile section after 6 months

Fig. 6 - Oxygen contours of spoil pile section after 6 months

Fig. 7 shows the maximum temperature (after a simulation time of 6 months) as the proportion of coal in the reactive zone was varied from 0 to 100% coal.

While the above presentation showed the use of a numerical model to predict spontaneous combustion in spoil piles, methods to combat and control real self heatings are described in Carras et al, (1997) and Haneman and Roberts (1997) which the interested reader should consult for greater detail.
Fig. 7 - Maximum temperature in spoil pile vs carbon content of reactive waste

CONCLUSION

Self heating in spoil piles is due mainly to the interaction of coal and carbonaceous spoil materials with oxygen and water. Should the heating remain unchecked, spontaneous combustion can result with attendant safety, environmental and economic consequences. While considerable progress has been made in recent years on the understanding and modelling of spontaneous combustion the processes involved are sufficiently complex that greater work is required for methods to be developed which will allow accurate predictions of self heating and spontaneous combustion in spoil piles. It is hoped that the model presented will demonstrate what advances can be made in prediction of spontaneous combustion with presently available data.

REFERENCES


Chen, X. D. and Stott, J. B. (1993). The effect of moisture content on the oxidation rate of coal during near-equilibrium drying and wetting at 50°C. Fuel, 72, 787-792


Some Investigations into the Explosibility of Mine Dust Laden Atmospheres

A D S Gillies1 and S Jackson2

ABSTRACT

An investigation into different aspects of importance in the understanding of explosibility of hybrid mixtures of coal dust, air and gases potentially found within mine workings through a comprehensive laboratory program of explosibility tests was conducted. Plotting of test results revealed that conditions of potential explosibility could be described using two dimensional flammability limit surfaces for coal dust/oxygen; methane/oxygen; and coal dust/methane mixtures. From these plots, the three dimensional flammability envelopes defining the explosibility of the coal dust/methane/oxygen mixtures can be defined and illustrated for a coal sample. The surfaces of the three dimensional envelope describe limits which separate inert mixtures of coal dust and methane at varying oxygen levels from those concentrations which are ignitable under defined conditions. It is considered appropriate to generalise that the geometric shapes of these limit regions are applicable to all type classifications of coal dust. There are practical applications of these results to the underground environment.

The action of free radical initiators in the propagation of a methane gas explosion was examined for its applicability to the flammability of coal dust/gas/air mixtures. The oxides of nitrogen or NO radicals have influence upon the lean limits of flammability of hybrid mixtures and this is illustrated by use of three dimensional coordinate geometry. Gases can be introduced to the underground environment through the exhaust gases of diesel equipment. It is concluded that radical species can substantially increase the flammability of gas and dust flames and as a consequence raise risk of mine atmosphere explosibility.

INTRODUCTION

Explosions are a continuing threat to the safety of operations in underground coal mines. Although each incident differs in structural details from others, one can nevertheless determine a typical scenario from the evidence of post disaster investigations (Hertzberg, et al., 1982). The usual sequence of events is for an initial ignition in a methane/air mixture to raise and disperse clouds of coal dust from the mine floor and ribs, thereby creating an atmosphere capable of sustaining a rapid deflagration.

Many environmental and mechanical factors contribute to the explosion phenomenon. Modern mining methods and machinery are capable of high production rates from a single coal face resulting in the presence of increased levels of methane and coal dust which increase the risk of ignitions or explosions (Landman and Phillips, 1993). The production of coal dust quantities throughout the working sections of a mine is a byproduct of the cutting, loading and transporting operations conducted underground. Ventilation airflows carry the finer size fractions for some distances before being deposited onto floor, roof and ribs. In addition, high production rates in gassy mines can lead to the liberation of significant volumes of methane simultaneously with the formation of coal dust.

The lineal advances obtainable with modern mining equipment in high output workings in bituminous coal mines cannot always be fully utilised because of an inability to maintain methane concentrations below acceptable limits (usually 1.0 to 2.0 per cent by volume as enforced by mining regulations). Future development trends are towards greater face production output. This may lead to demands for changes in approaches to explosion suppression, improved ventilation and gas control measures.

From a safety point of view the question arises as to whether increasing the limiting gas concentration (assuming homogeneous mixing) involves an increase in the risk of explosion. This question cannot be answered simply, for an

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increased methane concentration may affect not only the development of an explosion but also the incendivity of the airborne dust (Reeh, 1980). Dust concentrations that are currently of no practical importance with regard to explosion potential may, in combination with significant methane levels, present a hazard. High energy sparks which may ignite methane at very low concentrations can lead to a propagating explosion. It is therefore of interest to determine how high energy ignition sources interact with coal dust/methane/air mixtures ("hybrid mixtures"). The primary objective of this study can broadly be defined as an investigation into the ranges of explosibility of a Queensland coal dust in mixture with methane under varying oxygen levels.

Diesel equipment finds wide use within the underground coal mining industry for the transport of personnel and equipment. As a result the composition of diesel exhaust gases has been widely studied in an effort to reduce the health hazards posed by toxic gases such as carbon monoxide and the various oxides of nitrogen. However recent work has established that some exhaust gases can ultimately increase the explosibility of mine atmospheres containing both coal dust and methane.

The action of free radical initiators in the propagation of a methane gas explosion has been examined for its applicability to the flammability of coal dust/gas/air mixtures. The influences of the oxides of nitrogen or NO\textsubscript{x} radicals upon the lean limits of flammability of hybrid mixtures have been investigated and some results are illustrated by use of three dimensional coordinate geometry.

**BACKGROUND**

The flammability of a system is describable as some form of limiting geometric surface that delineates a domain of flame propagation within from a region outside of that surface where flame propagation is not normally possible. That mathematical surface, the flammability limit surface, describes a discontinuity in the real combustion behaviour of any system. Its exact size and shape in space are of basic significance in evaluating the practical hazards involved in the use of fuels, refined substances, and synthetic chemicals.

In the case of the explosibility of coal dust, work has been conducted upon the two dimensional explosibility surface for coal dust with respect to oxygen concentration. European researchers (Deguingand and Galant, 1981; Krzystolik and Sliz, 1988; Bartknecht, 1989; Wolanski, 1992) have hypothesised that while the lean limit concentration of dust is not greatly influenced by a reduction in oxygen concentration the rich limit concentration is significantly affected (fig. 1).
The limits used to define the explosibility of a coal dust are the same as those defined for methane mixtures (fig. 2). Coward (1929) derived the methane/oxygen relationship from empirical data that depicted whether mixtures of methane and oxygen in mine air were potentially explosive under certain conditions. Coward's triangle indicates that methane gas is explosive between approximately 5 and 15 per cent by volume when mixed with air. Each of these diagrams represents a two dimensional mathematical surface of compatible dimension defining the explosibility of the respective reactive fuel in the presence of a variable oxygen concentration.

![Coward's triangle for methane](image)

**Fig - 2** Coward's triangle for methane (After Coward, 1929)

The methane/oxygen explosibility diagram is especially significant as methane is the most frequent constituent involved in mine explosions. However, major mine disasters caused by explosions invariably, in addition, involve coal dust. It therefore follows that these two diagrams could be utilised to draw a three dimensional envelope representing hybrid explosive mixtures. The dimensions for the envelope would be defined by the explosion limits of the coal dust, the methane and the limiting oxygen concentrations for both the dust and gas.

Foniok (1985) defined a hybrid dispersive mixture as flammable dust in air with a small addition of flammable gas. This mixture has lower lean limits of explosibility than either of the flammable constituents. Further, a reduction in the minimum ignition energy to ignite the mixture and a reduction in the most explosive concentration of dust is apparent.

Le Chatelier's law can predict the decrease in the lean flammability limit with methane admixture (McPherson, 1993). Le Chatelier proposed a linear relationship that weighs the lean limit for each of the two components (methane and coal dust) according to the percentage of each in the hybrid mixture.

The three dimensional geometric surface defining the explosibility of hybrid mixtures of coal dust and methane gas under varying oxygen concentrations will be defined according to the explosion limits of the fuel. The concept of the three dimensional geometric surface has been described by Gillies and Jensen (1994). The size and shape of the explosibility envelope is defined by the explosion limits of the respective coal samples (Lebecki, 1991) as methane can be considered to be a constant fuel source.

The oxides of nitrogen (NOx) are a common product found within diesel exhaust gases. Several oxides of nitrogen are usually found together and are collectively known as nitrous fumes (Le Roux, 1990). Nitrous fumes, being a mixture of NO, NO2, N2O4 and perhaps some N2O5 are within the mine environment present in the exhaust gases of diesel engines and are produced in small quantities by both oxyacetylene welding and arc welding. Of particular interest with regard to nitrous fumes is the characteristic that in the presence of high energy sources the fumes can act as a radical initiator. The result of this action is the formation of NO and NO2 radical ions. A radical can be defined as an atom or group of atoms with an
unpaired electron and includes such species as triphenylmethyl, chlorine atoms, sodium atoms and nitrogen dioxide (Nonhebel and Walton, 1974). Many free radical reactions are brought about by the addition of radical initiators to the reaction system. These initiators break down to form the free radicals.

A large body of work (Sosnovsky, 1964; Peters, 1991; Leffler, 1993) confirms that nitrogen dioxide free radicals will enhance the reactions of hydrocarbons through a nitration process. The ease with which gas-phase nitration occurs depends upon the nature of the hydrocarbon. The hydrocarbons that more readily yield radicals are usually the ones that are most easily nitrated. For example, methane is more difficult to nitrate than ethane as higher hydrocarbons can more easily disperse the methyl free radical.

There are only a small number of publications referring to the temperature depression effects of NOx gases on methane ignition and related aspects under circumstances found within the mining industry. Coward (1934) described "concentric tube" ignition experiments undertaken by Dixon. He found that retention of traces of NOx dramatically reduced the ignition temperature of many combustible gases with a maximum depression of ignition temperature for methane/air mixtures of 122°C in the presence of 7000 ppm NOx. The Twentieth Annual Report of the British Safety in Mines Research Board (1941) reported that the presence of 2000 ppm NOx in a gaseous explosive mixture of air and hydrocarbons of 16 per cent ethane in methane was found to have its ignition temperature depressed from 732 to 550 °C. Later Annual Reports (1942/43/44) from the same organisation reported that a powdered explosive (70 per cent ammonium nitrate and 10 per cent nitroglycerine) introduced into air as fine particles depressed ignition temperature of methane/air mixtures from 700 to 370°C.

Fairhall (1993) in a study at The University of Queensland confirmed the earlier findings of Dixon. He found that the introduction of chemically produced laboratory NOx gases into a methane/air mixture gave a 106°C ignition temperature depression at an introduced gas concentration of 1600 ppm.

Diesel engines are in widespread use in modern underground coal mines. To achieve safe operating conditions, regular engine maintenance is required and maximum power output may have to be reduced and engine derating undertaken when operating at altitude by reducing the fuel injector setting to account for decreased atmospheric pressure. Australian state mining regulations set maximum NOx exhaust limits of 1000 to 2000 ppm. Fairhall (1993) reported some diesel emission data from a survey of working equipment in an Australian coal mine. He found that raw exhaust gas tests of five diesel machines using "Draeger" gas analysis stain tubes reported NOx emissions generally within the range of 300 to 400 ppm. One machine produced readings of 800 and 1000 ppm over two tests. Exhaust oxygen levels were generally less than seven per cent and carbon dioxide in excess of ten per cent. Within the general body of mine air the maximum NOx concentration recorded was 2.5 ppm in an air split with more than one unit of equipment in use. Examination of raw exhaust readings from another mine with 20 units of diesel equipment in use revealed NOx emission levels across the range 20 to 600 ppm.

LABORATORY MEASUREMENTS

A series of experimental determinations of the combustion behaviour of both coal dust and gas mixtures was conducted within a standard 20 litre explosion chamber sphere. Siwek (1982) described the design and standard method of operation of this chamber for the laboratory determination of dust and gas explosibility. A homogeneous dust cloud can be formed within the chamber through evacuation of the chamber followed by pneumatic dispersion of the pulverised dust. Methane and any other gases are added with a syringe. An ignition source comprising two chemical igniters is used to provide ignition energy of 10 kJ. Controls placed upon the system ensured that all tests were conducted with standardisation for both atmospheric temperature and pressure. Undertaking three identical tests at each concentration and five tests at each limit concentration tested repeatability of results. Details of the experimental approach are described in Jackson, Gillies and Golledge (1997).

Although use of the 20 litre explosion test sphere has gained international acceptance for the laboratory evaluation of dust and gas ignitions, there exist two standard methods for defining an explosion. The principal difference between these is the ignition energy to be used. The standard set by the American Society for Testing and Materials (ASTM E1226-88, 1994; ASTM E1515-93, 1994) recommends the use of a low energy ignition system with energy level no higher than 2.5 kJ. On the other hand, the International Standards Organisation (ISO 6184/1,2,3, 1995) recommends the use of a 10 kJ ignition energy for the testing of combustible dusts and a 10 J ignition energy for the testing of combustible gases.
These recommendations also carry over to the testing of hybrid dust/gas flames where the ISO recommends use of the 10 kJ energy due to the presence of dust. However, by virtue of the reaction mechanism, gas flames and therefore hybrid flames are vulnerable to being overdriven by high ignition energy. As a result a discontinuity exists between the lean limit result for the high energy tested hybrid mixtures and the lean limit result for the low energy tested gas mixtures under ISO conditions. The ISO therefore recommends that when gas results are to be juxtaposed with hybrid results, the 10 kJ energy must be used (Siwek, 1982). As a result the lean limit values determined for methane using the ISO standard are significantly less than the accepted result of 5.0 per cent by volume predicted by the ASTM standard.

For this investigation, results have been obtained and evaluated using both standards. It can be concluded that the explosion limits predicted by the ISO standard encompass dust and gas mixtures that will ignite but not propagate unless the shock wave produced by the ignition can subsequently raise dust into the atmosphere thereby creating a dust concentration within the ASTM explosive region. Those mixtures defined as explosive under ASTM conditions have properties to propagate a flame so long as neither the airborne dust or gas concentrations decrease.

**COAL DUST EXPLOSIBILITY**

Proximate analysis for the test coal sample is shown in table 1.

<table>
<thead>
<tr>
<th>Proximate analysis result, %, for coal sample</th>
</tr>
</thead>
<tbody>
<tr>
<td>Moisture</td>
</tr>
<tr>
<td>Ash</td>
</tr>
<tr>
<td>Volatile Matter</td>
</tr>
<tr>
<td>Fixed Carbon</td>
</tr>
</tbody>
</table>

The lean and rich limit explosion data determined using the 20 litre testing chamber are shown in table 2. The testing of rich limits is inherently difficult due to the high dust concentrations involved, and as such, the rich limit data can only be taken as indicative of the real behaviour.

<table>
<thead>
<tr>
<th>Lean and rich limit data, g/m³, for the coal dust sample</th>
</tr>
</thead>
<tbody>
<tr>
<td>Standard</td>
</tr>
<tr>
<td>Lean Limit</td>
</tr>
<tr>
<td>Rich Limit</td>
</tr>
</tbody>
</table>

In addition to these results, the experimental series investigated the limiting oxygen concentration for the coal sample. In this case, the gaseous agent used to reduce the oxygen concentration within the vessel was nitrogen gas. It would be expected that if carbon dioxide were used as an inerting agent it would give results that vary slightly from those presented here. The results from the 20 litre chamber at reduced oxygen concentrations are shown in table 3.
Table 3 - Limiting oxygen concentration (LOC) data for coal sample

<table>
<thead>
<tr>
<th>LOC (20 l) %</th>
<th>Oxygen (vol %)</th>
<th>ISO</th>
<th>ASTM</th>
</tr>
</thead>
<tbody>
<tr>
<td>Lean Limits (20 l) g/m³</td>
<td>21</td>
<td>40</td>
<td>60</td>
</tr>
<tr>
<td></td>
<td>15</td>
<td>50</td>
<td>80</td>
</tr>
<tr>
<td></td>
<td>10</td>
<td>60</td>
<td>130</td>
</tr>
<tr>
<td></td>
<td>9</td>
<td>70</td>
<td>150</td>
</tr>
<tr>
<td>Rich Limits (20 l), at 8.5 vol % oxygen, g/m³</td>
<td>8.5</td>
<td>230</td>
<td>200</td>
</tr>
</tbody>
</table>

METHANE EXPLOSIBILITY

Coward's triangular shape for the limit surface of flammable gases has been widely applied to a variety of gases over an extensive range of ignition energies and explosion chamber types. The results obtained under ISO standards increased the range of explosive concentrations beyond those normally accepted. Siwek (1982) and Bartknecht (1989) have shown that increased ignition energies will reduce the lean limit value for methane ignitability and that high energy igniters (of the rating used) do not over drive the resulting explosion. The explosion limits for methane in air are illustrated in table 4.

Table 4 - Lean limits, vol%, for the flammability of methane gas in air

<table>
<thead>
<tr>
<th>Lean Limit</th>
<th>ISO</th>
<th>ASTM</th>
</tr>
</thead>
<tbody>
<tr>
<td>Rich Limit</td>
<td>27</td>
<td>16</td>
</tr>
<tr>
<td>Nose Limit</td>
<td>4</td>
<td>6</td>
</tr>
<tr>
<td>Methane</td>
<td>12</td>
<td>12</td>
</tr>
</tbody>
</table>

The limits resulting from the ASTM criteria mirror those results that are generally accepted for methane gas explosibility. The ISO data has expanded explosion limits results for all but the limiting oxygen concentration of 12 per cent oxygen. The explosion curves produced by the gas ignitions indicate that energies greater than 10 kJ have a significant effect upon the rate of gas ignition. This increase results in a subsequent increase in the explosion pressure produced by the gas ignition. It is unlikely that these increases caused by the high ignition energy will result in a propagating explosion. However, the possibility exists that the increased explosion pressure could raise mine dust into the atmosphere (Siwek, 1982). The ISO standard has been formulated to include these possibilities.

HYBRID MIXTURES EXPLOSIBILITY

Lean limit data at normal atmospheric oxygen concentrations for the coal dust sample in admixture with methane gas are illustrated in fig. 3 which presents limit data over the range of gas concentrations for which explosions could be initiated. The data has been plotted to present an explosion profile with the coal dust concentrations on the vertical axis and the gas concentrations on the horizontal axis. The values determined for the lean limits of the methane gas in table 6 are taken as the first point of each curve on the horizontal axis. The vertical axis intercepts are taken from the limit data for the coal dust samples. All points between these two represent the measured values for lean limit concentrations at various methane concentrations.

The resulting curves for each sample divide the graphs into three regions. Dust/gas mixtures with concentrations higher than the lean ASTM standard curve are capable of propagating a flame away from the ignition source without the necessity of added fuel. Concentrations falling within the region between the two standards represent those mixtures that cannot be guaranteed to propagate a flame. However these concentrations are capable of producing a shock wave of sufficient intensity to raise dust into the atmosphere, thereby creating a hybrid potentially explosive mixture. The lower region indicates mixtures that were found to be non-explosive under the ignition conditions within the explosion chamber. The
ISO curves follow the trends predicted by Le Chatelier's law with a generally direct linear relationship. However the ASTM curve shows a trend toward a curve where the adsorbed methane alters the behaviour of coal insofar as its explosibility is concerned, even increasing the magnitude of the explosion to a greater extent than the alterations caused by the presence of methane in the atmosphere together with coal dust. This effect has been found in previous work (Torrent and Arevalo, 1993).

![Graph](image)

**Fig. 3 - Lean limit data for the coal sample**

The results obtained for the rich limit curve cannot be considered as reliable as those for lean limit. The laboratory testing of rich limits is inherently difficult due to the size of the equipment and the volume of dust required. It therefore becomes difficult to maintain a stable homogeneous dust cloud at the required concentration. Nevertheless, rich limit concentrations were determined although due to the reduction in accuracy for concentrations above 1 kg/m³, testing increments of 200 g/m³ were used. The rich limit data are shown in fig. 4. The curves again portray three regions akin to the situation with lean limit curves. Both the ISO and ASTM curves follow the same trend with a near straight line relationship below 10 per cent by volume methane. However above 10 per cent methane, the coal dust rich limit concentration becomes more dependent on the methane concentration under ASTM conditions than under the ISO conditions.

The rich limit curve for the hybrid mixture is determined mainly by the limiting oxygen concentrations (LOC) of the dust and gas. The LOC for the hybrid mixture is taken to be the lowest value of either the LOC (dust) or LOC (gas). In most cases, the LOC of the dust is the lowest value and therefore becomes the LOC value for the hybrid. As shown in fig. 5, the LOC for the dust (and therefore the hybrid) occurs at dust concentrations between 50 and 100 g/m³. At this dust concentration the limiting effect from the displacement of oxygen by methane is at its most pronounced.
It is possible to speculate on the reasons for this behaviour with the ISO curve used as an example. The experimental series did not continue beyond 30 per cent by volume methane due to the inaccuracies in gas mixing at such high concentrations and therefore the rich limit curve could not be continued. However, by using the LOC for 30 g/m$^3$ of 11 percent by volume oxygen and assuming that methane has the same effect as nitrogen, a rich limit value of 47 percent by volume methane is obtained. The rich limit action of methane is not simply explained by inertisation of the atmosphere through oxygen depletion. Methane in this reaction undergoes a slow combustion under which more energy is consumed than is produced. While there may be sufficient oxygen to allow an initial ignition to occur, it does not progress as the released energy by combustion is less than the ignition energy required for propagation of the explosion within the surrounding atmosphere.

**LIMITING FLAMMABILITY SURFACE FOR THE HYBRID MIXTURE SYSTEM**

Data discussed to this point has been two dimensional in nature, presenting information on the behaviour of dust/oxygen, methane/oxygen, or dust/methane mixtures. Gillies and Jensen (1994) have shown that these two dimensional explosibility surfaces can be brought together to form a three dimensional mathematical surface describing the flammability limits of dust/methane/oxygen mixtures potentially found within the underground environment. This envelope, the flammability limit surface, has been defined for the coal sample under both ISO and ASTM conditions in fig. 5. As explained previously, the ASTM region includes mixtures that are capable of propagating a flame out from the ignition source. The
ISO region poses a marginally lower hazard, as the mixtures cannot propagate a flame on their own. These mixtures require the addition of more dust or gas, or a slightly higher oxygen concentration to allow an explosion to occur. However the power of the initial ignition produced by these mixtures is sufficient to raise dust into the general body of air. Such an event could lead to a propagating explosion.

It is therefore evident that the ignition source and the ignition energy will significantly alter the size and shape of the flammability envelope with respect to both the coal dust and the methane gas. For the methane gas in particular, both the ASTM and ISO standards produced lean and rich limit data at variance to generally accepted results. Under the ISO standard at high ignition energy the lean limit for the methane was found to be as low as 2.0 per cent by volume. The experiments indicated that increased ignition energy would expand the explosion limits of methane gas. Far from overdriving the gas flame within the laboratory chamber, the increased energy simply allowed the gas to burn at a faster rate. Bartknecht (1989) has determined that this laboratory simulated effect will extrapolate to the real environment.

A comprehensive experimental series was undertaken to determine the lean limits of flammability for dust, gas and hybrid mixtures in the presence of NOX gas. The tests were conducted at methane concentrations from 1.0 to 18.0 per cent by volume with addition of 2000 ppm of NOX. This test concentration level of NOX was selected as representative of the upper statutory limit allowable under some Australian mine regulations. Results indicated that the NOX gas can reduce the lean limit concentration for methane under ISO standards with an ignitable mixture of gas being formed from as little as 1.5 per cent methane by volume. Data has been plotted in fig. 6 under ISO standards and fig. 7 under ASTM standards. These curves clearly show the reduction in the lean limit values in the presence of the 2000 ppm NOX. The effect of the radical on the ISO standard is substantial. The effect on the ASTM standard curve is also significant with the methane lean limit reduced to 3.5 per cent by volume and the dust lean limit to 40 g/m³.

**THE INFLUENCE OF NOX ON EXPLOSIBILITY OF COAL DUST/METHANE GAS HYBRID MIXTURES**

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**Fig.5 - Absolute flammability limit surface for the coal sample**

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Figs. 6 and 7 curves have characteristic shapes dividing the region into three distinctive sections. Under both standards the maximum decrease in the lean limit values occurred at less than 1.5 per cent by volume methane.

The experimental series also investigated the influence of the radical initiator upon coal dust at lowered oxygen concentrations. This testing could not be expanded to include the testing of methane gas at lower oxygen concentrations due to the difficulties in adding the methane and NO\textsubscript{x} to the 20 litre test chamber in accurate quantities. When the NO/air mixtures were formed, the resulting oxygen/nitrogen ratio was no longer 21:78. The results from limited oxygen concentration testing on dust at a single concentration of NO\textsubscript{x} of 2000 ppm are shown in Table 5.
Table 5 - Limiting oxygen concentration (LOC) for dust in air under the influence of NO\textsubscript{3} free radicals

<table>
<thead>
<tr>
<th>NO\textsubscript{3} Concentration (vol. %)</th>
<th>0 ppm</th>
<th>2000 ppm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Oxygen Coal Dust (g/m\textsuperscript{3})</td>
<td>ISO</td>
<td>ASTM</td>
</tr>
<tr>
<td>21.0</td>
<td>40</td>
<td>60</td>
</tr>
<tr>
<td>15.0</td>
<td>50</td>
<td>60</td>
</tr>
<tr>
<td>10.0</td>
<td>60</td>
<td>120</td>
</tr>
<tr>
<td>9.0</td>
<td>LOC</td>
<td>LOC</td>
</tr>
<tr>
<td>8.5</td>
<td>70</td>
<td>LOC</td>
</tr>
<tr>
<td>7.0</td>
<td>LOC</td>
<td></td>
</tr>
<tr>
<td>6.0</td>
<td>LOC</td>
<td></td>
</tr>
</tbody>
</table>

At standard atmospheric oxygen concentrations the presence of the radical acts by reducing the influence of the rate determining reactions which form part of the methane gas flame system. However at lower oxygen concentrations, the presence of NO\textsubscript{3} acts to remove the dependence of the reaction on the oxidation process and replaces it with a nitration process. This effect is only limited owing to the small quantity of NO\textsubscript{3} present in the tests with the result that LOC values did not extend below 10 per cent by volume.

**HYBRID MIXTURES IN ADMIXTURE WITH OXIDES OF NITROGEN**

Testing was conducted upon concentrations of hybrid mixtures of methane gas and coal dust. A laboratory produced gas mixture of NO, NO\textsubscript{2} and air at a concentration of approximately 2000 ppm of NO\textsubscript{3} was added to these mixtures. The results indicated that the presence of NO\textsubscript{3} has a significant effect on the explosibility of coal dust/methane/air hybrid mixtures. Fig. 8 graphically sets down lean concentration findings under ASTM definition standards.

This three dimensional block represents the explosive envelope for the hybrid mixtures at varying oxygen concentrations. The lightest shaded region illustrates atmospheric concentrations under normal temperature and pressure conditions at which an underground coal mine may operate without the potential of ignitions occurring. If, however, 2000 ppm of NO\textsubscript{3} gas is present in the atmosphere, the dark potentially explosive region is expanded to include the marginal grey region to account for reductions in the lean limit values for dust/gas hybrid mixtures. The situation using the ISO standards interpretation is illustrated in fig. 9.

It can be seen that the oxides of nitrogen gas have a significant effect upon minimum oxygen concentration levels required for a potentially explosive mixture. This is due to the fact that the oxidation of the fuel has been replaced to some degree by the nitration of the methane gas. This allows the methane to produce the methyl free radical readily at low oxygen concentrations. However the reaction system requires some oxygen due to the low concentrations of nitrates and so a limiting oxygen concentration is observed. The effect of nitrates upon coal dust in the absence of methane is largely due to the volatile gases produced by the initial heating of the coal grains. The nitrate radical will attack these volatiles thereby reducing the lean limit concentration for dust.
Fig. 8 - Effect of presence of free radicals upon coal dust/methane/air hybrid explosibility under ASTM definition standards

Fig. 9 - Effect of presence of free radicals upon hybrid atmosphere explosibility under ISO standards
The influence of nitrous fumes acting as free radicals to increase the rate of reaction for combustible gases has been examined and the effect shown to be significant. Of particular significance is the effect of the radical upon lean limit concentrations necessary for an ignition leading to a propagating explosion to occur. However the probability of nitrous fumes from diesel exhausts building to sufficient quantities to constitute a hazard is low due to the high quantities of mine ventilating air that are found under modern mining method.

Coal mine regulations have been formulated based on generally understood combustion behaviour of gases tested in isolation or commonly occurring mixtures such as coal dust/methane atmospheres. It would appear that empirically based studies are required to confirm the behaviour of complex gas mixtures that may be found underground with increasing use of mechanisation and introduction of various "artificial" manmade materials.

The possibility exists that the action of the radicals can be used to increase the safety of underground operations. A methane flame will only propagate as long as enough radicals are produced by the chain branching reactions to maintain production of the methyl free radical. If an impurity in the form of a radical scavenger were to be introduced to the system, the scavenger could act to inert the radicals being produced by the branching reactions and therefore remove the ability of the methane to form the methyl radical. In such a situation, although the methane concentration may be within the explosive region, the flame would not propagate thereby causing flame extinction. It is therefore concluded that the action of appropriate radicals could be used as a method of atmospheric extinction within sealed panels.

CONCLUSIONS

A coal sample from Queensland’s Bowen Basin has been studied for explosibility behaviour under laboratory conditions using a 20 litre capacity testing chamber. The lean and rich limit concentrations for the dust sample in air were determined utilising ISO and ASTM standards. Further, the trends in the limit concentrations were examined while reducing the oxygen concentrations until the point of limiting oxygen concentration had been established. For the explosion testing of methane in oxygen, the results mirrored those of Coward (1929) with respect to the triangular shape of the flammability limit surface. The test apparatus indicated a lean limit of explosibility for methane of 4.5 per cent by volume. However methane gas will produce a high rate of pressure rise at 2.0 percent by volume under the influence of high ignition energy. This low concentration can therefore be considered to be flammable. The limiting oxygen concentration for the methane explosion test was determined to be 12.2 per cent by volume, a finding that agrees with published work (Bartknecht, 1989, Miniz, 1993).

Lean limit concentrations for the hybrid mixture of coal and methane generally followed Le Chatelier’s law. However the ISO standard results exhibit a phenomenon in which the lean limit of the hybrid is less than the sum of its components. This occurs due to the fact that not all of the coal mass is involved in the explosion when the lean limit for the coal is determined. The addition of methane enables more of the coal mass to ignite and thereby reduces the lean limit for the hybrid mixture.

The two dimensional flammability limit surfaces developed were used to construct a three dimensional flammability limit surface. This surface describes the explosion limits of the coal dust/methane/oxygen system and is most significantly influenced by the explosibility characteristics of coal dust with the limiting oxygen concentration for the system paralleling that of the coal. Methane presents a stronger influence over the explosion limits when low volatile content coals are considered.

An examination has been made of the action of free radical initiators and in particular the influence of the NO₃ radical on the lean limits of explosibility of mine atmospheres carrying coal dust/methane gas mixtures. Two and three dimensional geometry has been used to illustrate the effects. It is concluded that the presence of radical species can significantly change explosibility characteristics of methane gas, airborne coal dust and hybrid mixtures and substantially reduce flammability limits of the atmospheric mixtures.

ACKNOWLEDGMENTS

The authors acknowledge the generous assistance and advice given throughout this investigation by a number of parties and, in particular, the financial and technical expertise extended by the management and staff of the Queensland Government’s Safety in Mines Testing and Research Station, Redbank.
REFERENCES


Rapid Generation of Control Charts for Analysis of Complex Gas Mixes in Crisis Situations

I Porter and M Jacobs

ABSTRACT

After a methane explosion in a coal mine a pre-determined crisis management procedure is instigated. As the process rapidly evolves the inevitable question arises - are personnel trapped underground, are they alive and can they be rescued without putting the lives of the rescue crew at risk? This is a very emotive question with many sub-questions such as; could anybody have survived, is there the potential for a secondary explosion and is there a ‘window of opportunity’ before that secondary explosion occurs?

To determine if that window of opportunity exists two fundamental elements must be in place: a method of determining the atmosphere at key points in the mine and a method of predicting the course of change of that atmosphere with respect to time. The first element can be achieved if the tube bundle system is intact and drawing samples from known points, alternatively boreholes can be drilled from the surface. The second element can be realised by utilising a sequence of computer generated control charts which have a time axis, a percentage combustibles axis, the upper and lower explosive limits of the current atmosphere and a prediction option which allows the user to look at the potential changes in the atmosphere over a set period of time. These control charts would be part of the mines on-going crisis management system rather than a tool to be used after an explosion had occurred.

DETERMINING THE EXPLOSIBILITY OF AN ATMOSPHERE

In 1928 H.F. Coward published a paper (Coward, 1928) in the Transactions of The Institution of Mining Engineers detailing a method for determining the explosibility of atmospheres behind stoppings. This was the introduction of the Coward’s triangle for methane (figure 1), which graphically illustrated the existence of upper and lower limits of percentage methane which when mixed with air will form an explosive atmosphere. Also illustrated was how the upper and lower (marginally for lower) combustible gas percentage varies with percentage oxygen. By plotting a particular mix of methane and oxygen on the diagram it can be determined if that atmosphere is either explosive, explosive with the addition of methane, explosive with the addition of fresh air or cannot become explosive.

![Coward's Triangle for Methane](image-url)

Fig. 1 - Coward’s triangle for methane

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The basic Coward's diagram can be expanded to take account of the explosive potential of other gases such as CO and H₂, however, there are still a number of limitations regarding analysis during an emergency situation. The primary limitation is the difficulty in following the trend of the mix with respect to time particularly when the ratio of the various combustibles varies, hence changing the shape of the triangle. A number of solutions to these limitations have been proposed. Hughes and Raybould (1960) developed the idea of control charts which had time as the x-axis and percentage combustibles on the y-axis. For a given mixture (sampled at a known time) the upper and lower critical values can be determined from the explosive triangle and these can be plotted on the control chart against time. The state of the atmosphere is also plotted on the chart. A new triangle is developed for each new sample and the control chart is modified accordingly. After a number of samples have been analysed a trend can be established (figure 2). The problem with this is the time taken to develop the new triangle and update the control chart in a crisis situation.

Fig. 2 – Manually developed control chart illustrating method of construction
As can be seen from the diagram the trend of the mixture towards potentially explosive and explosive can be visualised. Limitations of this method are that the index value for potentially explosive and explosive situations can vary from location to location and require prior experience. This renders the method inappropriate for crisis situations where time for evaluation is at a premium.

To alleviate the problems associated with the previous methods, Ellicott (1981) modified the flammability triangle for complex mixes by including the additional extinctive effect of CO₂ and transforming the triangular diagram into an x-y plot (Fig. 4) with the nose point as the origin.

From the diagram it can be seen that as successive samples are plotted the trend of the atmosphere can be rapidly established. As with the original flammability triangles the limitation of the diagram is the lack of a time scale, hence requiring additional time consuming calculations to determine the trend of the sample with time.
To overcome all of the aforementioned limitations a computer program (EXGAS) was developed to automatically generate control charts which show a plot of combustibles against time. Also included on the plot are the upper and lower critical limits of the flammability triangle for that particular mix of flammables. Fig. 5 is an example of a control chart generated by the program.

Included in the algorithms used to generate the control charts are the greater extinctive effect of CO₂ over nitrogen and the effect of temperature on the shape of the flammability triangle for methane. If the greater extinctive effect of CO₂ were not included any errors so resulting will be on the safe side, however, table 1 shows that omitting the effect of temperature does not err on the safe side, particularly in a fire situation.

<table>
<thead>
<tr>
<th>Temperature (°C)</th>
<th>Lower Limit (CH₄%)</th>
<th>Upper Limit (CH₄%)</th>
<th>Nose Point (O₂%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>20</td>
<td>5</td>
<td>14</td>
<td>12.2</td>
</tr>
<tr>
<td>250</td>
<td>4</td>
<td>18.5</td>
<td>9.5</td>
</tr>
<tr>
<td>500</td>
<td>3</td>
<td>20</td>
<td>7.2*</td>
</tr>
</tbody>
</table>

*# extrapolated approximation*

After two data points are entered the program can then be set to prediction mode. The prediction mode can predict the trend of the given atmosphere and can also predict the future shape (upper, lower and nose) of the explosive triangle taking into account the various factors which effect the explosibility limits. Also, as the data points for the program are taken from the gas analysis system it was a simple matter to add an algorithm to determine the CO/O₂ deficiency ratio (Graham's Ratio) which would indicate the likelihood of a spontaneous combustion situation occurring; i.e. a potential source of ignition. This could also be added to the prediction mode. A significant potential of the programme is that with some minor software modifications and appropriate interfacing, the system can be connected directly to the gas analysis system and run in the background of any networked computer. If an event were unfolding and is detected automatically (say using an automated 24 hour prediction mode) it would generate a warning message which overrides the screen output and audio output of particular systems such as the main control. This system could not be overridden without certain actions being undertaken.
APPLICATION OF PROGRAM TO KNOWN DATA

The following data is taken from reports (Simtars 1995,) relating to the explosion in the area of 512 Panel at Moura No 2 Underground coal mine on 7 August 1994. As the only information taken into account is that from the gas monitoring system, the analysis does not profess to judge, draw conclusions on, or attribute blame to any person or operation associated with the explosion and post explosion events. It merely illustrates some of the potential advantages associated with the rapid production and analysis of control charts using the EXGAS program. The data analysed was obtained from the records of the gas monitoring system, a Maihak Unor tube bundle system. Data was available from 27 July, but was analysed only from 6 August when results became significant, 48 hours before the first explosion occurred. Fig. 6 is a plan of the workings around 512 Panel, with the gas monitoring points highlighted.

Sealing 512 panel

Work to seal 512 Panel started on Friday 5 August and was completed at around 1:15am on Sunday 7 August. In preparation for final sealing, at around 2:30pm on Saturday 6 August, monitoring point 5 was relocated to 20 metres inside the seal in No 3 heading, to monitor the atmosphere inside the sealed area. Point 16 was moved at 2:00am on Sunday 7 August to south return of 510 Panel. Tecrete seals, constructed of wire mesh baskets into which plaster is poured, were utilised. The plaster builds up strength over time to provide a seal which meets statutory requirements. The position of these seals is shown in Fig. 6. Prior to sealing, from around 17 June 1994, there had been several reports of a benzene or tarry smell in the area of 512 Panel, suggesting the possibility of spontaneous heating. The source of the smell proved elusive (SIMTARS, 1995).

Analysis of data from monitoring point 5

As stated previously data was available for all points from 27 July 1994, but prior to the explosion only monitoring point 5 provided significant data. This point, originally located in the north west return of 512 Panel, indicated relatively stable levels of CH₄ and CO until around 9:00 am on 6 August. After this time the methane level increased from around 0.30% to 0.85% at 2:30 pm on the same day, at which point it was transferred to it’s post sealing location 20 metres inbye the seal position of No 3 heading. Initial readings from this position were between 0.20% and 0.30% methane. Carbon monoxide levels at this time did not exceed 3 ppm and this was the case up to 4 hours before the final sealing. The CO level then increased steadily to over 10 ppm (a CO/O₂ deficiency ratio of 0.2) about an hour before sealing and continued to climb after sealing to reach 102 ppm (a CO/O₂ deficiency ratio of 0.66) at 2:57 pm on 7 August. It should be noted that sealing an area does not in itself produce an oxygen deficient atmosphere, it is the oxidation of the carbonaceous material within the sealed area that does so. The combustible content of the atmosphere at 2:57 pm was 3.4%. At this point, using the prediction option from EXGAS, the time to the lower limit was around 9.5 hours (12 midnight). By 6 pm that evening the deficiency ratio had risen to 0.7 and the methane level had reached 4%, and with an oxygen level of 18.7% the atmosphere was rapidly approaching the lower explosive limit. The prediction function indicates that the time to the lower limit was 5.5 hrs (11:30 pm). Depending on which particular sample is analysed, the time at which the atmosphere reaches the lower limit varies from 11 pm to midnight. This variance is not critical, at this stage, as a time range rather than an exactness is required. As the background level for the deficiency ratio was between 0.1 and 0.2 and had risen to 0.7 it suggests that there may be heating (source of ignition) in the vicinity of a potentially explosive atmosphere. At this point a warning message would be generated by EXGAS and a potential crisis situation indicated.
Continuing to use EXGAS to produce control charts and predict the future state of the atmosphere would show the following situation, Fig. 7, at 10 pm, one hour and forty minutes before the explosion.

---

**Explosibility Diagram of %Oxygen vs %Combustibles**

![Explosibility Diagram](image)

<table>
<thead>
<tr>
<th>Sample</th>
<th>%Combustibles</th>
<th>%Oxygen</th>
</tr>
</thead>
<tbody>
<tr>
<td>236</td>
<td>4.85</td>
<td>18.41</td>
</tr>
</tbody>
</table>

**Time to lower limit:** 1 hr 37 mins

**CO/02 deficiency:** 0.79

---

Fig. 7 - Output from EXGAS for data recorded at 10 pm, 7 August.
This shows that the deficiency ratio has risen to 0.79 and that the atmosphere will reach the explosive limit at 11:37 pm. Other samples taken around this time show similar deficiency ratios and a lower limit time of plus or minus a few minutes off 11:35 pm. Continuing to view the crisis, at 11:08 pm the deficiency ratio had risen to 0.82 and the lower limit was predicted for 11:40 pm. If all these factors could have been rapidly put together at the mine, there was still 30 minutes to get the miners in the vicinity of 512 Panel to relative safety. Fig. 8 shows the control chart for the 72 hours up to and after the incident.

![Control Chart](image)

Fig. 8 - Seventy two hour control chart from 6 August to 9 August 1994

**DISCUSSION**

It's easy to be wise in hindsight. The above analysis of data from gas monitoring point five, indicates that all the information required to advise the removal of all personnel from the area before the explosion was available. At the mine the information from analysis of the gas data, although used to predict the time the atmosphere would reach the lower limit, did not, for whatever reason, appear to warn of the potential source of ignition. To warn of the impending crisis, output from the gas analysis system would have to be viewed and analysed by a skilled person who had knowledge of the indicators of spontaneous combustion. That person would have to have been on-site that day (a Sunday) and have been available to monitor the trend in the atmosphere. In the current corporate climate of economies of scale in personnel and multi-tasking this is too great a risk to leave to chance. It would therefore be desirable to have access to a program such as EXGAS with the facility to take output directly from the gas analyser and have this linked to a crisis management program (currently under development at the University of Wollongong) which would give an audible and visual alarm if a crisis situation was predicted. The crisis management program would then provide a step by step procedure that any mine personnel could follow.

An important point to note is that monitoring point 5 is the only sampling point in the sealed area. Is the atmosphere sampled at this point indicative of the atmosphere in the rest of the area and is it acceptable to use these values in a crisis situation? The answer to both these questions is open to conjecture, but probably to the negative. It would be correct to say that a number of points situated at key locations would be desirable. Determining the optimum number and location of monitoring points should be the subject of future research.

**CONCLUSION**

As only methane is taken into consideration in the case study the full potential of the control charts may not be obvious. In fact analysis of the explosibility data could have been, and in fact was prior to the first explosion, done manually. If, however, temperature and other combustible gas variations were taken into account, manual calculation of the control charts and prediction of trends with respect to time would be laborious and overly time consuming. The program at it's current stage of development goes some way to solving this problem.
So where was that window of opportunity - the 36 hours between the first and second explosions? Unlikely, as analysis by gas chromatograph of samples taken from boreholes drilled into 512 panel after the first explosion showed that methane levels were high and the potential for a second explosion existed during the full 36 hours.

The window of opportunity did in fact exist in the 12 hours before the first explosion. If surface personnel had access to appropriate computer hardware and software, the crisis situation which was unfolding may not have been missed and evacuation of underground personnel could have taken place.

REFERENCES


A Review of Fatal Outburst Incidents in the Bulli Seam

C R Harvey¹, R N Singh²

ABSTRACT

The Bulli Coal Seam, located in the Illawarra Coalfield of New South Wales, has a long and varied history of sudden outbursts. From available information this problem has resulted in twelve fatalities over the last one hundred and one years, with over five hundred separate outburst incidents being identified. These incidents have varied in severity and intensity from the discharge of 1 to 2 tonne of coal, with a slight increase in gas emission, to the discharge of 200 to 400 tonne of coal with some 6,000 m³ of gas being liberated and large items of mining equipment being pushed 1 to 2 metres down the roadway.

Geological features associated with these outbursts can be mapped on a regional and mine by mine basis, to provide some indication or warning. Similarly changes in gas content and gas composition can also be determined on a regional and mine by mine basis. However sudden geological changes (such as a 6.5 metre seam displacement within 15 metres) variations in gas content (such as 6 m³/tonne to 15 m³/tonne) and changes in gas composition (from 95% CH₄ to 90% CO₂) all within one mine does make the prediction or forecasting of outburst incidents exceedingly difficult.

A review of the fatal outburst incidents in the Bulli seam can give valuable insight into how efficient coal mining, within the Bulli coal seam is now linked with the effective use of gas drainage techniques and the management of the outburst risk. The number of variables and "unknowns" associated with outbursts have required the development of fully documented procedures and systems, so that at all stages during the mining operation the risk of injury to mine workers is minimised. The main emphasis for these management systems is the prevention of outbursts by relieving gas pressures using drainage techniques to achieve specified threshold gas level to permit safe mining conditions.

INTRODUCTION

The Bulli coal seam as a primary source of coking coal is a major economic resource for New South Wales and Australia. Geological conditions associated with this seam have required the development of specialised mining techniques and the adaptation of mining equipment, for coal mines to remain competitive. For many years outbursts have been regarded as a mining phenomena one of the many quirks of nature which make underground coal mining inherently dangerous. Today this type of approach is unacceptable.

The challenge to modern day mine management is to recognise the problem; research and understand it; then develop mining and management systems so that as much coal as possible can be mined as safely as possible. A detailed review of the fatal outburst incidents for the Bulli seam, will show how the level of knowledge and understanding of outbursts have evolved and, how plans and systems have been developed to effectively manage the outburst risk.

BULLI SEAM OUTBURSTS

Whilst outbursts have been a phenomena associated with mining the Bulli coal seam for the last one hundred and one years, the mechanism behind outbursts was not fully understood and hence the various outburst parameters were not

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² Professor of Mining Engineering Wollongong University
clearly defined. This lack of definition and background information was due mainly to each mine being responsible for its own operations, as well as an inability to admit that outbursts were a problem for all Bulli seam mines. As a consequence, a lack of technology and information transfer between mines existed for some time.

This situation changed drastically after the South Bulli Outburst, 25 July 1991 when three miners were killed. Various working groups and task groups were established to identify the mechanisms causing outbursts, the most suitable means of managing the outburst risk, develop standardised data collection and information interchange within the industry, review outburst prediction techniques and recommend areas of future research. The general characteristics of outbursts as they relate to Bulli seam mines are given in Table 1. The importance of structures, seam gas composition and seam gas concentration are apparent. Similarly gas drainage was identified as currently the only effective mechanism for preventing outbursts.

Table 1 – Bulli seam outbursts

<table>
<thead>
<tr>
<th>Colliery</th>
<th>No. of Outbursts</th>
<th>Size in tonnes</th>
<th>Gas</th>
<th>Geological Structure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Appin</td>
<td>22</td>
<td>2 - 88</td>
<td>mainly CH₄ &amp; CO₂ on dykes.</td>
<td>Predominantly strike slip faults; mylonite zones.</td>
</tr>
<tr>
<td>Brimstone</td>
<td>2</td>
<td>30</td>
<td>CO₂</td>
<td>Mainly dyke related structures with strike slip movement.</td>
</tr>
<tr>
<td>Corrimal (closed)</td>
<td>4</td>
<td>12</td>
<td>CH₄ &amp; CO₂</td>
<td>Shear zone associated with minor faulting &amp; dykes.</td>
</tr>
<tr>
<td>Kemira (closed)</td>
<td>2</td>
<td>60 - 100</td>
<td>CO₂</td>
<td>normal fault with mylonite.</td>
</tr>
<tr>
<td>Metropolitan</td>
<td>37</td>
<td>1 - 150</td>
<td>mainly CO₂ with minor amounts of CH₄</td>
<td>Predominantly with dykes &amp; faults that exhibit slicken sides &amp; mylonite.</td>
</tr>
<tr>
<td>South Bulli</td>
<td>7</td>
<td>1 - 300</td>
<td>mainly CO₂</td>
<td>Strike slip faults with mylonite; dyke zones &amp; thrust faults.</td>
</tr>
<tr>
<td>Tahmoor</td>
<td>88</td>
<td>5 - 400</td>
<td>mainly CO₂</td>
<td>Mainly strike slip faults; with dykes (110° - 135°) &amp; thrust faults: mylonite usually present.</td>
</tr>
<tr>
<td>Tower</td>
<td>19</td>
<td>1 - 80</td>
<td>mainly CH₄</td>
<td>Mainly strike slip faults with dykes.</td>
</tr>
<tr>
<td>West Cliff</td>
<td>250</td>
<td>4 - 350</td>
<td>mainly CH₄ with CO₂ to the NE development</td>
<td>Predominantly strike slip faults (100° - 110°) with slicken sides &amp; mylonite; dykes and thrust faults have been associated with outbursts.</td>
</tr>
</tbody>
</table>

FATAL OUTBURST INCIDENTS

Up to December 1997 there have been twelve fatalities attributed to outbursts in the Bulli seam. These are summarised in Table 2 and discussed in detail below.
Table 2 - Fatal outbursts in the Bulli coal seam

<table>
<thead>
<tr>
<th>COLLIERY</th>
<th>DATE</th>
<th>No. KILLED</th>
<th>SIZE (tonnes)</th>
<th>GAS</th>
<th>STRUCTURE</th>
</tr>
</thead>
<tbody>
<tr>
<td>Metropolitan</td>
<td>10 June 1896</td>
<td>3</td>
<td>Unknown</td>
<td>CH₄ (firedamp)</td>
<td>Dyke and soft fault zone</td>
</tr>
<tr>
<td>Metropolitan</td>
<td>27 July 1926</td>
<td>2</td>
<td>140</td>
<td>CO₂</td>
<td>Fault with 5m throw</td>
</tr>
<tr>
<td>Metropolitan</td>
<td>2 December 1954</td>
<td>2</td>
<td>90</td>
<td>CO₂</td>
<td>Normal fault with 0.3m throw</td>
</tr>
<tr>
<td>Tahmoor</td>
<td>24 June 1985</td>
<td>1</td>
<td>400</td>
<td>CO₂</td>
<td>Behind a dyke associated with strike slip movement</td>
</tr>
<tr>
<td>South Bulli</td>
<td>25 July 1991</td>
<td>3</td>
<td>300</td>
<td>CO₂ &amp; CH₄</td>
<td>Thrust fault with 35 cm of mylonitic coal; very high gas pressure.</td>
</tr>
<tr>
<td>West Cliff</td>
<td>25 January 1994</td>
<td>1</td>
<td>350</td>
<td>CO₂</td>
<td>Intersection of 2 strike slip structures; 30 cm of mylonitic coal.</td>
</tr>
</tbody>
</table>

The reports and documentation on the earlier incidents are sketchy and limited to the reports prepared and available at the time. Records from the Department of Mineral Resources have proven to be the most informative.

10 June 1896

On 10 June 1896, three men were killed by an outburst of coal and gas at Metropolitan Colliery (Enoch Pugh; James Borton and H. Shipton). The men were suffocated by the gas (claimed to be fire damp) from the outburst in the No. 7 West Heading. An inquest was held on 11th June 1896 at the Metropolitan Colliery Office by the District Coroner, C.C. Russell, Esq. The jury returned a verdict of accidental death.

27 July 1925

On 27th July 1925, two men were killed by an outburst of natural gas and coal at the Metropolitan Colliery (George West and Fredrick Green). The men were suffocated by the gas emitted (identified in 2 samples as being between 50% and 62% CO₂) from the outburst site and was associated with an upthrow fault immediately adjacent to a dyke in the 70 yard heading of the Western area section of the mine (see Figs 1 and 2). Approximately 220 tones of coal was ejected with the gas concentration being sufficient to kill a horse approximately 115 yards away from the outburst site.

Miners working the area had commented that the coal immediately prior to the outburst was harder than usual. It had been identified in previous outburst incidents that these events were related to geological structures, particularly dykes where the characteristics of the coal had been significantly altered, and fortunately these incidents had not been fatal. The concept of one district or part of the mine having outburst prone zones was recognised.

Hence the comments of the Inspector reporting on the fatality and the mechanisms used to alleviate outburst problems are quite telling: "Since the outburst, boreholes have been kept ahead in all places in the near vicinity of the fault and these have given off a little gas which has proved to be practically pure carbon dioxide".

Similarly the mechanism underlying the principles behind outbursts in the Bulli Seam were alluded to when the same inspector commented "The last four feet or so of coal worked by the miners in this place was harder than usual, and no gas had been given off..." "This hard section of coal may have acted as a dam and retarded the escape of the gas which is given off freely from the coal faces in this section."
Fig. 1 - Scene of fatal accident to F Green and G West, outburst of gas Metropolitan colliery

Fig. 2 - Outburst of Gas – Metropolitan Colliery
2 December 1954

On 2 December 1954; two men killed by an outburst of the Metropolitan Colliery. (names not given in Dept. Annual report): the men were asphyxiated by the gas (deduced to be CO₂) and a boring machine operator rendered unconscious.

The outburst was in an area of known faulting with the site being associated with two small down throw faults with several defined vertical joint planes in the shale roof and a well defined “slicken sides” in the seam, coal in the fault area was soft; and being worked under fully mechanised mining methods. The use of boreholes to identify outburst structures was raised in the Chief Inspector of Coal Mines report, stating: “In my opinion, the recent happenings clearly indicate the necessity for maintaining the boreholes well in advance of all solid places and, in addition to the centre hole, flank holes are being bored in each rib and the holes are being maintained at least twenty five feet in advance of the faces”.

24 June 1985

On 24 June 1985, one man (Michael Joseph Penny) was killed whilst operating a continuous mining machine in an outburst prone area (C heading of 204 panel) of Tahmoor Colliery. The outburst resulted in approximately 330 tonnes of coal and roof shale material and an estimated 3500 cubic metres of gas, comprised predominantly of carbon dioxide.

The outburst was associated with a known dyke structure which had been intersected in three previous panels with increasing thickness from 20 cm to 1m. The thickness of the outburst site could not be determined due to roof falls. In 204 panel, about four metres before the dyke an off shoot of an igneous intrusion (about 30m thick) was identified along with a shear zone and severe jointing being evident in the roof.

Due to previous outburst events and the proximity of the dyke the area to be mined was considered to be outburst prone. Hence outburst precautions were to be taken. The continuous miner was set up to cut out the right hand side of the heading with the head of the machine being sumped in at the roof.

At the time of the incident the shuttle car was behind the continuous miner and the driver sustained superficial facial injuries from small particles of coal and other material ejected by the outburst. Material ejected in the outburst covered the continuous mine the shuttle car, with the only access to the operators cabin on the continuous miner being via the rear of the canopy. The autopsy report indicated that the operator of the continuous mining machine died from asphyxiation. The other four men (including the shuttle car driver), recovered after making their way to fresh air.

An inquest into the event was held on 4 November 1985 before Coroner Donna Maria Delaney in the Campbelltown Coroner’s Court. The finding was “died of asphyxiation due to the inhalation of coal gas from an outburst”.

Consequent to this incident the following recommendations were given to enhance safety whilst mining in outburst prone areas of Tahmoor mine.

1. Upgrade the miner driver’s cockpit to give the driver better protection as well as having an independent air supply.

2. Gas drainage is to be carried out to the satisfaction of the District Inspector of Coal Mines.

3. It is intended to require the manager to use pulsed infusion shot firing similar to a recent practice of Metropolitan Colliery, if the gas drainage results do not prove satisfactory.

4. Modified precautionary measures will be put into practice whilst mining through outburst prone areas.

24 July 1991

On 24 July 1991, three men (Craig John Broughton, Robert Kelvin Coltman and Leigh Ronald Pearce) were killed by the outburst of coal and gas at the South Bulli Colliery.
The outburst in “B” heading of W12 Panel ejected an estimated 300 tonnes of coal and 6000 m$^3$ of gas (predominantly CO$_2$) into the working area. This occurred with sufficient force to dislodge the ventilation ducting, losing the auxiliary fan ventilation, slew the shuttle car sideways and had sufficient force to blow open the outbye ventilation doors causing a short circuit in the ventilation. The continuous miner driver was buried to his neck with outburst material and it is believed he died instantaneously from the effects of carbon dioxide. The shuttle car was being driven away from the continuous miner at the time of the outburst and from the injuries sustained by the car driver it would appear he was thrown out of the driver’s compartment by the force of the outburst. It would appear that the third miner killed, died attempting to assist the car driver and, was overcome by the gas.

Although the outburst had not been predicted, the investigation revealed that there were significant changes in face conditions with such factors as ingress of water, changes in stress direction, roof jointing, roof guttering, poor conditions, fluctuations in gas concentrations and gas composition. The presence of greasy backs, (slicken sides), white clayey material in the roof and softening of the coal were also observed. The inability to recognise and understand the significance of these changes ultimately lead to the death of these men.

25 January 1994

On 25 January 1994, one man was killed by an outburst of coal and carbon dioxide gas at West Cliff Colliery (Malcolm Leslie Butt). The outburst occurred in “B” heading of 486 Panel and ejected 260 tonnes of coal, with a large but unquantifiable amount of carbon dioxide from the right hand rib side.

On the previous shift, mining activity in the panel were operating under "normal" mining procedures (not outburst mining procedures). A number of changes in mining conditions were noted, particularly the hardness of the coal at the face, deterioration in roof conditions, the presence of a “greasy back” otherwise known as slicken sides in the roof trending longitudinally down the heading and, an increase in carbon dioxide being emitted during mining operations. These changes in mining conditions were deemed to be of significance, causing mining to continue only under outburst mining procedures. It is believed that this decision prevented other people from being killed, injured or affected, as outburst mining procedures limited the number of people working in the vicinity of the face.

The continuous mining machine in use was equipped with a purpose designed and built outburst protection canopy, including a supply of filtered air for breathing, via a half face mask. However, even with this protection, the miner driver was killed in the outburst. A post mortem revealed that the miner driver had died of anoxia and had sustained a small linear fracture to the rear of the skull, believed to have been caused by direct impact with the filtered air supply regulator gauge, at the time of the outburst.

The coal ejected from the outburst entirely covered the continuous mining machine and back to a distance of thirty metres from the face. The carbon dioxide gas given off with the outburst was in sufficient quantity to entirely fill the face area, displace all oxygen and proceed to migrate back down the panel, filling number seven cut through and affecting the adjacent heading.

As shown in Figs 3, 4 and 5, the outburst was associated with a combination of two strike slip fault zones (one trending 350°, the other 280° magnetic). The intersection of these two fault structures created a zone of intense shearing/jointing resulting in lower coal permeability and increased stress in the coal and associated roof strata. This in combination with the presence of Mylonite and gouge material was believed to account for the volume of gas released.

The dominant 350° fault zone appeared to have a similar alignment with strike slip fault structures identified in previous mining, associated with development headings for panels 484 and 485. Drilling immediately in advance of mining, in “B” heading of 486 panel did not accurately identify the location of this structure. Gas drainage holes drilled from 485 panel failed to reach across the longwall block and effectively drain the development heading of 486 panel. Also whilst holes had been drilled in advance of mining in 486 panel, they had not penetrated the structure. These holes had deviated to the left (south) and were identified in the left hand rib and face.
Fig. 3 - West Cliff Colliery 486 right panel outburst and fatal accident geological mapping

Fig. 4 - West Cliff Colliery 486 right panel outburst and fatal accident site geology plan
350° STRIKE SLIP FAULT ZONE

3mm GOUGE

2-3mm GOUGE

40-60mm GOUGE

5mm GOUGE

1-2mm GOUGE

2-3mm GOUGE

LIMT OF OBSERVATIONS DUE TO BURST COAL

OBSERVATIONS
1 88° to 264° MAGNETIC STRIKE SLIP
2 STRIKE SLIP
3 DIP SLIP

Fig. 5 - West Cliff Colliery 286 right panel outburst and fatal accident section through strike slip fault “b” heading southern rib

WHERE TO NOW?

Since 1994 an order under Section 63 of the Coal Mines Regulation Act (1982) has been imposed on all mines operating in the Bulli Coal Seam. This order sets rigorous levels for insitu gas concentrations (being 9 m³/t for CH₄ and 5 m³/t for CO₂) for mining to be permitted.

Consequently outburst mining procedures are now regarded as an ultimate barrier which should not have to be utilised. The concept of sealing a continuous miner driver in a special canopy while more then two hundred tonnes of coal is ejected at him, in an irrespirable atmosphere, no longer has a great deal of appeal. Hence gas drainage now has a greater level of significance for all Bulli seam mines.
The refinement of gas drainage techniques have seen the development of specific drainage profiles for different gas concentrations, focusing upon the spacing of drainage holes and the amount of time necessary to achieve the threshold values, thereby permitting safe mining operations. In conjunction with these advances in gas drainage methods there have been significant improvements in drilling techniques and technologies, focusing upon the need for directional drilling control and surveying of the holes.

The overall management of all the components and information needed to verify that it is “safe to mine” has been encapsulated in outburst management plans. In this respect an outburst management plan (OMP) is a control document which specifies practices, resources, activities and responsibilities relevant to the effective management of the outburst risk. To achieve this and effectively manage the outburst risk the management plan should have the following elements:-

- The OMP must be based on an assessment of the outburst risk to be managed at each particular mine site. Due regard must be paid to the different levels and types of risk presented by predominantly carbon dioxide as opposed to methane seam gas environments.
- The OMP must be fully and effectively documented to ensure that processes and standards at the mine are suitable for the management of the outburst risk and proceed in accordance with the plan.
- The OMP must contain a policy statement signed by the most senior officer of the mining company.
- The policy statement should contain an expression of the broad objectives of the plan and the corporation’s commitment to the attainment of these objectives.
- The OMP must ensure that all documents and records are maintained to indicate that all appropriate actions have been taken.
- The OMP must be in a form which allows effective transfer of information and responsibilities, which is able to be effectively updated. And
- The OMP must be adequately resourced in respect of plan development, implementation and ongoing maintenance. This will include the appropriate involvement of employees.

To facilitate the preparation of these plans the Department of Mineral Resources has prepared an Outburst Mining Guideline; MDG No: 1004, outlining the essential elements and components for an effective Outburst Management Plan.

CONCLUSIONS

The level of knowledge and understanding in managing the risk of outbursts for Bulli seam mines has definitely improved. One hundred and one years ago outbursts were regarded as an inevitable risk of mining and to some extent were treated as an “act of god”. Currently gas drainage is utilised as the primary mechanism for taking the energy out of an outburst structure and thereby making it safe to mine. The Outburst Management Plan is the mechanism whereby assurance is given that all precautions are in place, gas drainage has achieved the appropriate threshold levels and it is in fact safe to mine.

The success in managing the outburst risk has the potential to generate complacency and for this reason attention must be directed to auditing the outburst management plans and improving gas drainage techniques. It is possible that guidance for the future may come from past observations particularly the comments of the Inspector reporting on the 1925 fatality; “Since the outburst, boreholes have been kept ahead in all places in the near vicinity of the fault, and these have given off a little gas which has proven on analysis to be practically pure carbon dioxide.”

Also the comments from the Chief Inspector of Coal Mines reporting on the 1954 fatality; “In my opinion, the recent happenings clearly indicate the necessity for maintaining the boreholes well in advance of all solid places and, in addition
to the centre hole, flank holes are being bored in each rib and the holes are being maintained at least twenty five feet in advance of the faces". Gas Drainage would appear to be part of the answer there may be the limitation to this technology

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ABSTRACT

In November 1996 the NSW Minister for Mineral Resources, Hon Bob Martin MP, commissioned ACIL Economics and Policy Pty Ltd to conduct a wide ranging review of safety within the New South Wales mining industry.

The review was a response to a disturbing number of fatalities which had occurred in the industry during the Minister's term of office together with a number of alarming near misses where serious failures in mining systems had occurred. Shortly after the review was commissioned, four mine workers died at Gretley Colliery near Newcastle. An inquiry is currently being conducted into the incident.

ACIL were given the following terms of reference for the MSR (MSR):

1. Identify key issues which need to be addressed before a significant and measurable improvement in mine safety performance, and an observable reduction in the potential for serious safety incidents, can be expected.
2. Explore options for addressing these key issues, through consultation with a wide range of stakeholders.
3. Consider how the findings of the Warden's Court Inquiry into the 1994 Moura Mine disaster should be applied in New South Wales.
4. Evaluate the role, activities, structure, employment conditions, and resourcing of the State's Mines Inspectorates in light of the identified key issues.
5. Evaluate existing legislative provisions in the light of the identified key issues.
6. Provide Government with recommendations on how mine safety in New South Wales could be enhanced, with particular regard to the potential contribution of the Inspectorates.

The ACIL report titled "Review of Mine Safety in NSW" was tabled in Parliament by the Minister in April 1997. The review considered industry safety performance and its measurement through considering available statistics, drawing on submissions and, predominantly seeking the views of a broad cross section of industry and Government personnel. This included mine managers, inspectors, union officials and the workforce.

In its report, the MSR makes numerous observations concerning issues of safety and the mining industry. Arising from those observations the review made forty four (44) recommendations (See Appendix A) for either consideration or action to address the issues identified.
THE RECOMMENDATIONS

The recommendations covered everything from safety measurement, safety incentives, communicating commitment, workforce involvement, training, contractors, risk management, the Inspectorates, legislation and regulation, and implementation of the Moura inquiry recommendations.

The Minister has expressed support for all 44 recommendations and conveyed an expectation that they be addressed.

MANAGEMENT STRUCTURE

A two level tripartite structure was set up to address the recommendations - a ‘Steering Group’, comprising a high level mix of representatives from mining companies, unions and government, including the Minister for Mineral Resources; and an ‘Implementation Group’, chaired by Professor Dennis Else, Chairman of the National Occupational and Safety Commission.

Four Task Groups were set up by the Implementation Group to carry out the work required to address the recommendations.

The management structure for the implementation of the MSR’s recommendations therefore is shown in Fig.

![Management structure of recommendations](image)

Carriage of a number of issues raised by the MSR recommendations have been given to tripartite Task Groups, the NSW Minerals Council or individual mine sites for implementation:

- Remote control equipment (recommendation 15) - this is being dealt with by Task Group 1 (chaired by John Waudby, Department of Mineral Resources);

- Performance measures and information sharing (recommendations 1,2,3,5,8,18) - these are being dealt with by Task Group 2 (chaired by Garnett Halliday, Cadia)

- Principles governing risk assessment, a two-tiered regulatory approach and mine safety management plans (recommendations 16,39,41) are being dealt with by Task Group 3 (chaired by Gordon Galt, Cumnock Colliery). This Task Group also has partial responsibility for contractor issues (recommendation 14).
Inspectorate and Moura issues (recommendations 23-38, 40, 42-44) are being addressed by Task Group 4 (chaired by Bruce McKensey, Department of Mineral Resources)

- The NSW Minerals Council has carriage of the development of guidelines for site safety performance measurement, involvement of site personnel in the formation of safety targets, the impact of production bonuses, the improvement of safety information exchange through CEOs forums, and the promotion of guidelines for contractor safety management (recommendations 4, 5, 6, 9 and 13)

Individual mine sites are responsible for implementing a wide range of issues including, review of safety incentive schemes, promotion of middle management commitment to safety initiatives, worker involvement in assessment and management of core risks, and training for managers and workers (recommendations 5, 7, 8, 10-12, 14, 17, 19-22, 38, and 41).

The Implementation Group plays an oversight role to ensure mines implement their recommendations and has a similar role in regard to ‘training’ issues (recommendations 6, 19-21 & 43).

**PROGRESS**

*Task Group 1 - introduction and use of remote controlled equipment underground*

Task Group 1 has completed an issues paper and an options and recommendations paper on the remote control of mining equipment. Draft documents entitled, ‘Guidelines for the Use of Remote Controlled Mining Equipment’, and ‘Guidelines for the Design of Remote Control Systems for Mining Equipment’ have been circulated widely to the mining industry for comment and input.

The ‘use’ guidelines will be finalised and published in early 1998 and take into account a fatality involving remote controlled equipment at United Colliery in November 1997.

The ‘design’ guidelines are also expected to be finalised and published in early 1998.

The Task Group will be notifying mines of the existence of both guidelines and will be running a seminar in March 1998. The Group will also be developing a self-auditing process, in addition to its recommendation that the Department of Mineral Resources conducts audits to monitor the use of the document. The Group has also recommended that there be a tripartite review of the guidelines and audit results.

*Task Group 2 - performance measures and information sharing.*

A discussion paper ‘Mix of Measures for Industry OHS Performance’ has been drafted to canvas views on the proposed industry mix of measures and has been circulated to industry. This document highlighted the MSR’s criticisms of industry’s reliance on lost time injury frequency rates as the major performance indicator.

The paper offered ways in which industry could take the lead and explore alternative methods to more effectively monitor industry performance.

For example, indicators which cover culture, systems and outcomes have been developed to provide a spread of measures across all areas related to the current and future performance of the industry. Task Group Two recommended that these indicators should be reviewed at least every two years with a view to change and improve.

Task Group Two has also drafted guidelines ‘Mining Enterprise Measurement of OHS Performance for Mining Enterprises’ along with an accompanying ‘how to’ document ‘Moving to the New Way of Measuring of Measuring OHS Performance’ prepared by the NSW Minerals Council with the assistance of their OHS Committee and Andrea Shaw. Both documents were distributed widely for comments.

*COAL98 Conference Wollongong 18 - 20 February 1998*
Final recommendations will be submitted to the Implementation Group by late January and an implementation plan will be completed in February. A final report will be prepared for the Implementation Group meeting on 2 April 1998.

An 'Incident Reporting' form has also been designed and the Group has sought feedback/comments from industry along with e-mail addresses/fax numbers in preparation of a high potential incident database.

Task Group 3 - risk assessment, a two-tiered regulatory approach, Mine Safety Management Plans, and Contractor Issues

A draft discussion paper on the design of a new mine safety regulatory system has been completed on behalf of the NSW Minerals Council by SYSTEC P/L. The Group has reviewed the paper and will be consulting stakeholders during January and February. A final report (including an implementation plan) on the recommended principles for a new regulatory system and is being prepared for April 1998.

In brief, the ‘implementation plan’ will consist of:

**Step 1: Current phase (complete by February 1998)** Seek Stakeholder and Ministerial in-principle support for the core features of the recommended regulatory model in the SYSTEC paper. Determine if open cut sector is 'in' or 'out' of this process.

**Step 2: Design (Timeframe 3-6 months)** Design an action plan to implement the model, including consultative arrangements and an action plan detailing how legislative, policy and administrative change would be achieved.

**Step 3: Implementation (Timeframe 3-4 years, including a transition period)** Implement the actions contained in the action plan. A transitional arrangement to allow companies to move to compliance would be necessary.

**Step 4: Review (Timeframe ongoing)** Regular reviews of the implementation and effectiveness of the regulatory framework.

The Task Group has also reviewed and assessed Mine Safety Management Plan (MSMP) models and will prepare a work book to assist mine sites by February 1998.

The issue of contractor OH&S is also being investigated, in particular the development of a register of contractors.

Task Group 4 - inspectorate and Moura related issues

Task Group Four has appointed project managers to supervise and monitor the implementation of the twenty MSR recommendations it has responsibility over.

It has also prepared a status report that looks beyond addressing the recommendations - the MSR itself states that the individual recommendations provide a 'starting point response' and that efforts will need to go further.

The report describes the changes which will be made to the Department’s Mine Safety and Environment Division including a new structure, resource requirements, and changes to work priorities and patterns. The target date for restructuring is 1 July 1998.

This structure is based on a regionalised and integrated Safety Operations Unit, where the State has been divided into five new areas and a single manager in each area will be responsible for both coal and non-coal operations. The Sydney Office (Head Office), will have a Technical Services Group, Information Group, Investigation Group and an Environmental Group.

An implementation program is also described which sets out four issues of high priority - major hazard management, emergency readiness, a small mines program and developing and implementing the Mine Inspection Act General Rule.
NSW Minerals Council - guidelines for measuring safety performance, on site involvement re:safety targets, exchange of information via CEO forum, promotion of contractor OHS guidelines

Before considering a detailed study on the safety impact of production bonuses (and of possible measures available to address any negative effects), the NSW Minerals Council (NSWMC) has conducted a survey to gather basic information on the extent and nature of incentive schemes operating within the industry. The results are being collated and options/recommendations being offered to the Implementation Group in early 1998.

The NSWMC has also conducted a CEO forum in August 1997 ‘to allow a greater exchange of information on safety approaches’. A second forum is scheduled for early 1998.

Its contractor OH&S guidelines have also been finalised and ‘promotional’ workshops have commenced. A second workshop was held in Cobar in November 1997 and additional workshops are planned for Broken Hill and the Hunter Valley in the new year.

Mine Sites - review of safety incentive schemes, promotion of middle management commitment to safety initiatives, worker involvement in assessment and management of core risks, training for managers and workers

New South Wales mine sites were informed of their responsibilities to implement 14 of the recommendations in a letter from the NSW Minister for Mineral Resources in July 1997. A newsletter circulated in August 1997 and in a letter from the Chairman of the Implementation Group sent in September 1997 were also distributed.

The Implementation Group then distributed a questionnaire to all mine sites (with more than four employees) in early October 1997 to determine the level of implementation of the 14 recommendations. Responses to the questionnaire are currently being collated with future action to be determined.

CONCLUSION

The MSR’s Steering Group meets on 9 April 1998- exactly one year after the Review was tabled in Parliament by NSW Minister for Mineral Resources, Bob Martin, MP.

Much of the work required to implement the 44 recommendations from the MSR are expected to be completed or near completion at this time. However, to continue to improve industry safety performance, industry participants must be constantly alert to their everyday actions and decisions.

The MSR itself states that the individual recommendations provide a ‘starting point response’ and that although actioning the recommendations will have some impact on the industry safety culture, efforts will need to go far beyond this. ‘Fundamentally, industry participants must be constantly alert to the safety incentives and disincentives produced by their everyday actions and decisions. It is only through this vigilance that we can expect the improvements that are achieved to be sustained over the longer term’. (Review of Mine Safety in NSW by ACIL Economics and Policy Pty Ltd., 1997).

REFERENCE


APPENDIX A - MSR RECOMMENDATIONS

Measuring safety

1. NSW mining industry safety performance be measured on a mix of indicators. This mix might include LTIFR, FIFR, Disabling injury and progress in managing core risks.

2. The exact mix of measures be determined on a tripartite basis as a matter of urgency

3. The NSW DMR adopt this mix of measures and use it in the targeting of the Inspectorates' safety related activities.

4. The NSWMC develop guidelines for use by mine operators in determining how individual site safety performance is to be measured.

Safety aims

5. Companies give greater attention to involving those on site in the formation of safety targets.

Safety incentives

6. The industry commission a more detailed study of the safety impact of production bonuses and of possible measures available to address any negative effects.

7. Companies re-evaluate their existing safety incentive schemes with a view to establishing their actual safety impact as distinct from their effect on LTIFR.

Roles played by key individuals

8. Company boards take a more active role in requiring reporting on a mix of safety indicators which more accurately reflect site safety performance.

9. The NSWMC convene a CEO level safety forum to allow greater exchange of information on safety approaches.

10. Mine operators give high priority to promoting middle management commitment to and ownership of safety initiatives through the effective involvement of middle managers in the development and implementation of all such initiatives.

11. Mine operators provide training and support to enable middle managers to effectively carry out their role in communicating safety requirements to work groups under their control and ensuring compliance with safe operating procedures.

Workforce involvement

12. Companies re-evaluate their approaches to involving workers in safety management with a view to achieving greater worker participation particularly in terms of the assessment and management of core risks.

Contractor safety involvement

13. The NSWMC take an active role in promoting the use of the Guidelines for Contractor Occupational Health and Safety Management by its members.
14. Companies devote greater effort to the safety aspects of contractor selection and management, given that contractor safety performance in the broad remains a problem area.

**Engineering and equipment**

15. There be a tripartite examination of safety issues associated with the introduction of remote controlled equipment underground.

**Risk management**

16. The NSWMC and Inspectorates continue to promote risk assessment and management approaches as a valuable safety management tool.

7. Companies review their approaches to core risk assessment and management in the light of the identified concerns.

**Collation, analysis and use of accident information**

18. A tripartite group be asked to develop proposals for stakeholder consideration on how information sharing on accident cause can be improved. The group should in particular focus on the following areas: provision of information on serious incidents, and accidents across the industry, (that is between operators); and more effective means of communicating this information to mine middle managers and the mine workforce.

**Training**

19. Companies introduce structured safety and communications related training for Mine Managers, and mining professionals.

20. The levels of hazard awareness training provided to mineworkers in both the coal and metalliferous sectors be increased.

21. Each operation review its emergency procedures training.

22. Test evacuations of all or parts of sites should be an integral aspect of operations' approaches to emergency preparedness.

**The inspectorates**

23. The Department of Mineral Resources devolve environmental responsibilities to other officers with specific environmental expertise, and require the Inspectorate to focus wholly on matters related to minesite safety and health.

24. The Department move to create support positions of Mines Safety Officer with the detailed job description for such officers to be determined within the Department.

25. The Department give consideration to the introduction of cross-inspection as a mechanism for maximising the best use of Inspectorate resources.

26. Inspectorate policies and procedures on investigation and enforcement be developed and published.

27. The creation of a discrete Accident Investigation and Analysis Unit within the Inspectorate.

28. The Department of Mineral Resources determine the number of additional Inspectors required in the light of approaches taken to the redistribution of environmental responsibilities and the creation of Mine Safety Officer positions.
29. The Department introduce a more systematic approach to the prioritisation of Inspectorate activities.

30. Physical examinations of site operations continue to be a major aspect of the role played by Inspectors in both the coal and metalliferous sectors.

31. All Inspectors conduct both pre-announced and unannounced minesite visits, and that there be a requirement for sufficient unannounced visits to create a perception of a significant likelihood of an unannounced visit at any time.

32. The Department act without delay to resolve outstanding salary issues by bringing Inspectorate salaries into closer parity with those paid by other Inspectorates and by industry.

33. In the event that a significant increase in Inspectors remuneration levels is proposed, all affected positions be declared vacant and advertised.

Legislation and regulation

34. The Inspectorate adopt a more active approach to enforcement of the metalliferous General Rule.

35. A database on the status of implementation of requirements under the General Rule be developed and maintained by the Inspectorate.

36. The Department act immediately to establish the status of the implementation of the General Rule among smaller operators with a view to determining what particular assistance may be required.

37. Timeframes for the implementation of provisions under the General Rule be established and promulgated.

38. Companies, unions, and Government, devote further effort to informing mineworkers about the General Rule and its implications.

39. There be an immediate tripartite re-examination of legislative options for the coal sector, particularly as regards the practicality, and likely impact of, a two tiered regulatory approach.

40. Further consideration be given to the priority presently being given to the development of a single piece of mining legislation in NSW.

Moura inquiry implementation

41. NSW coal operators be required to prepare Mine Safety Management Plans to identify and manage all core risks.

42. As a first step, the Metalliferous Inspectorate be required to report to the Minister in detail on the possible application of MSMPS, and of other Moura Inquiry recommendations, to the metalliferous sector.

43. The Moura Inquiry training and communications recommendations be implemented by NSW coal industry stakeholders including the Inspectorate.

44. The DMR chair a NSW stakeholders group charged with determining the applicability of the Moura Taskgroup recommendations in NSW.
Implementation of the Moura Recommendations - A Managers Perspective

D Reece

INTRODUCTION

The tragedy of the Moura No. 2 incident has lost much of its emotional mileage for those organisations that thrive on loss and suffering. However, for the families, workers, mine owners and operators in the Queensland underground coalfields, the challenge of delivering a positive outcome, though well underway, is in its infancy. The pressure created by interested parties is necessary for overcoming the inertia involved in change; but they cannot deliver the change. The responsibility ultimately resides at each individual minesite with the people who work there each day.

Moura No. 2 has progressed through a number of stages. The Warden’s Inquiry generated a wide range of recommendations for the industry to address. Representatives from Government organisations, mine management, union bodies and the general workforce were then organised into taskgroups and delegated the responsibility of expanding the recommendations so that they could be practically integrated into mining operations. A number of sub-committees were also initiated in order to adequately cover particular, often individual issues in detail. The results of the taskgroup and sub-committee efforts has generally fallen into the areas of legislative changes, operational changes, or research activities.

At the time of delivering the Moura No. 2 recommendations, the then Queensland Labor Government unequivocally committed to instigating each and every one. This was also accepted by the following Liberal Government. This has been fairly controversial for the Queensland industry at the time because it has brought wide sweeping changes very quickly and will continue to do so for an extended period of time.

The focus of this paper is on the practical application and integration of the suggested or required changes into an underground coal mine in Queensland. It is not exhaustive but concentrates on those recommendations that have already provided safety, operational or financial benefits. It is specifically related to Central Colliery but will have a general similarity to other underground coal mines in Queensland due to the overriding mechanism for change, that is legislation.

CHANGE MECHANISMS AND PROCESSES

The system for driving the change at Central (and the Capricorn Coal Venture) is via the process mapping and re-engineering discipline. This provides the framework, logic and rationale with which each objective of the Moura recommendations is addressed and locked into present and future procedures. The topics to be covered fall into three broad categories:

- Equipment - oxygen self rescuers and means for self escape, monitoring and analytical systems, seals, inertisation apparatus and refuge chambers;
- Management Systems - training, sealing and monitoring procedures and management plans; and
- Legislation - which is linked, in principle at least, to all the topics that are discussed.

Capricorn Coal Management Pty Ltd Central Colliery
SELF ESCAPE

The emphasis for escape is on self escape rather than immediate reliance on mines rescue. This is covered with a range of options: personal oxygen self rescuers (30min duration); emergency breathing apparatus (60min duration) at each crib area, longwall face end and caches every fifteen pillars in the main headings; the refuge chambers previously described; life lines or emergency lighting/sound guidance systems and mobile refuge chambers in each panel.

Segregated intakes are standard throughout Queensland mines as per legislation and so provide a fresh air second escapeway from most areas of the mine.

Various scales of evacuation are mandatory at each site on an annual basis. These are designed to audit and retrain in the full range of protocols; management plans, trigger points, alarms and responses and personal preparedness.

Whilst the issue of escape is focused on after the event, the discipline of planning and auditing has provided the mine management and workforce with a greater degree of confidence and knowledge in both the ongoing control of the mine and the minimisation of loss if something were to fail.

GOAF SEALING

The requirement for sealing goaves with seals rated to withstand an overpressure of 20 psi initially created problems with cost, a backlog of sealing and labour requirements. However, these have been offset by the advantages gained. Following extensive testing a number of suppliers entered the market for supplying and/or installing these seals. This competition has led to more economic alternatives whereby a seal can be installed in about half the time and cost of those previously installed by mine personnel. Quality is also superior due to the degree of specialisation provided by the supplier.

Trials were conducted, at the mine, of the various types of seals available. This investigation resulted in a selection based on robust design, resistance to convergence, sealing ability, speed of installation and cost competitiveness. The same type of seal is installed in each cut-through of the longwall, on retreat. This has reduced the previous problem of methane contamination from the adjacent goaf into the existing tailgate.

Thirty seals and two bleeder ventilation shafts were installed at Central in order to effectively seal the old goaves and reduce the number of bleeder roads. This has greatly reduced the extent of roads that are required to be inspected and maintained.

Permanent refuge chambers are installed adjacent to the shafts because the bleeders are essentially single entries. Emergency winding facilities are available in each shaft. Hence, at the mine extremities emergency refuge or escape facilities are readily available. The additional seals along the main returns have provided greater control of the extensive goaf area and a reduction in the methane concentration in the returns.

A recent spin-off to this extensive sealing regime has come in the area of gas utilisation. Utilisation in the form of power generation or direct gas usage has been considered for some time. The main delays for this occurring have been:

- the ability to guarantee supply during longwall relocations;
- the ability to generate sufficient electricity to be of benefit to the extended site; marginal rate of return in the current economic climate.
- The benefits gained from the extended and improved integrity of the goaf area mean that there is now good potential to overcome the first two issues.
MONITORING

The full gas monitoring system comprises a range of software and hardware. Tube bundle and telemetric systems are both used for environmental sampling. Tubes are predominantly connected to each goaf area (nominally each longwall panel has at least one sample tube), as well as duplicating the telemetric system in the critical areas. Telemetric points (covering CO, CH₄, velocity and pressure, as appropriate) are provided at belt driveheads, return splits, shafts and electrical installations. These are both linked to a PC based analysis system. The first provision of the system is for detailed information on the environment in the mine. The system also provides immediate warning when predetermined alarm points are exceeded. In accordance with recent legislative changes this must be acknowledged, logged and corrected in accordance with formal procedures. This system has demonstrated its reliability to the extent that the response is measured in minutes - for a diesel vehicle in a panel, to rib crush at a seal allowing oxygen ingress or, a collapsed stopping that leads to a pressure change at the main fans.

The monitoring system is coupled to analytical software that provides information on gas explosibility via Coward or Ellicott diagrams as well as a range of ratios. This means that an accurate assessment of the changing environment in the mine or in a sealed area can be rapidly determined.

Legislation requires mine evacuation during and after sealing operations and close monitoring of the atmosphere to ensure the safety of personnel. Where previously this has meant outages as long as six days for the mine, the combination of improved sealing techniques, responsive monitoring and a greater level of understanding of spontaneous combustion and the sealing process throughout the workforce, now result in one and a half days outage (i.e. a weekend), with the further target of less than twenty four hours being achievable. This will become less of an issue with continuing experience and the utilisation of various inertisation methods as part of regular procedures.

The tube bundle system can also be coupled to a gas chromatograph for more accurate gas analysis and for the detection of products of combustion. Though this capability has been available and required in Queensland mines for a number of years the chromatographs that are now available provide results in about ninety seconds, to an accuracy of ten ppm, compared to the previous time of one and a half hours at fifty ppm.

This range of technology remains predominantly academic without the integration of the equipment into the mining system. This is done via a series of management plans that require the equipment, analysis and communication constantly being cycled and assessed. The required education and communication processes that support this system means that all mining officials and a large proportion of the workforce are familiar with and understand the analysis of explosive gas mixtures and their interpretation.

MANAGEMENT PLANS

Legislation requires the assessment of the principle hazards at each mine and the development of management plans for each one by a cross section of the mine workforce. This has not been carried out “piece meal” but rather as an integrated process of risk assessment to determine all hazards followed by the development of all necessary plans and training to specific deadlines. Management plans typically cover, Spontaneous Combustion, Gas, Ventilation, Strata Control, Evacuation, Outburst and Water.

Whilst this is clearly a huge task, again a number of benefits were realised. There has been a certain amount of co-operation between mines in order that the result would be achieved; cross checking with consultants, suppliers, contractors and other operators occurred as part of this process; the requirement to include a range of representatives yielded a more “friendly” document and also commenced the process of training and communication.

The training aspect cannot be understated in this process. It has provided not only an avenue for communication but also confidence through increased knowledge. The requirement for Quality Assurance type document control; whilst a principle for good business; is even more applicable to Queensland where the workforce is typically transient that information is naturally harder to control.
Though there has also been a huge workload for the Inspectorate, this process will ultimately simplify their systems by having benchmarks and standards across the industry and in turn lead to more professionally run operations.

**LEGISLATION**

Much of the required change to the operating structures was brought about by legislative changes. These have been far reaching - touching on areas of equipment acquisition, operating and management standards and procedures, statutory positions and competencies through to mandatory audits and reviews. Though initially seen as being dictatorial in nature, these directives served to catalyse owners, operators, unions, inspectors, suppliers and contractors very quickly into the same direction for mutual benefit. Each organisation responded to the challenge, not with ‘can it be done?’, but rather as “how is it best achieved?”, e.g. seal types, oxygen self rescuers and self escape systems, monitoring systems, refuge chambers, etc. This clear direction actually assisted mine operators to achieve the task set. This was because suppliers had the opportunity to make products available within a known time frame for a clearly defined market to an expected standard. Not only has a healthy level of competition been evident between suppliers (and is still evident), but there has also been healthy cooperation with mines and previous competitors in order to achieve the goals to everyone’s benefit.

**CONCLUSION**

The recommendations and required actions from the Moura No. 2 Inquiry and following Task Groups have been onerous and extensive in the accountabilities and responsibilities placed on the operators of Queensland’s underground coal mines. The immediate response is to resist the imposed change and look for an easy way out.

However, the acceptance of this change as being necessary. It has seen energy, otherwise used in resistance, channeled into education, training, investigation and management and has resulted in demonstrable benefits in cost control, and significantly improve levels of workforce knowledge and commitment.

There is very little that is new in the work that is underway in the Queensland mines when compared to a more global focus. However, the comprehensive range of issues being grappled with in a short time span is demonstrating just how much can be achieved when necessity dictates.

There are far more changes occurring due to the Moura recommendations than has been presented in this discussion. The points described have been those that have delivered benefits to the operation and should be considered for other operations not respondent to the Queensland requirements. They are cost effective business solutions providing step change improvements to an industry needing an overhaul in the areas of safety, cost structure and commitment.

**REFERENCES:**

Emergency Management Systems and Plans

M Bird

ABSTRACT

Despite improving technology, better training and the use of modern monitoring and communication systems there is a disproportionate rate of diameter occurrences in the mining industry. The use of appropriate plans and a practical approach and managing risk can reduce the number of occurrences and ensure better control of emerging situations when they do occur.

INTRODUCTION

From the time coal was first mined there have been difficulties. How to support the roof, getting fresh air to the face, mining economically, transporting the coal, seam gases needed dilution, fires occurred, coal dust could become explosive, outbursts, spontaneous combustion and rock bursts.

Usually, the solution to these difficulties relied on the mine management who would respond to each problem as it occurred. This could work well, depending on the knowledge and experience of the management personnel.

As a result of continuing disasters occurring in the coal mining industry and subsequent inquiries there has been a drive towards management plans being developed. The development of these plans originate with a risk analysis conducted by a cross section of management and the work force. They are proactive in their nature, have limits or warning levels incorporated and it is desired that they have ownership of all mine personnel through involvement and training.

Most of the technical areas are well developed and understood. Roof support plans, ventilation plans and gas management plans have been around for years, although the standard may and does vary between operations. Each of the plans must have provision for an escalating event which eventually will evacuate the mine and/or activate the emergency plan.

Generally, through history it has been found that the mine management and employees are well equipped and trained to manage the daily technical matters of mining coal. When a new problem occurs for the first time or when a major emergency occurs the industry in general is not prepared with the appropriate infrastructure, knowledge or experience. Although this is natural, in that we do not expect the unexpected to occur, we do not plan to have a major emergency and we all believe it will not happen to us.

Subsequently, coal miners from the manager to the face worker have been criticised in a number of Inquiries for their lack of systems and planning during an escalating situation and/or major emergency.

The purpose of this paper is to give some background and an update on the direction that the legislation is taking in regards to management plans, emergency procedure plans and the resources and training that are being developed.

EMERGENCY PREPAREDNESS MODEL

Fig. 1 shows the Emergency Preparedness Model that has been developed to assist operators in evolving their emergency plans. It is designed to be used as a aid ensuring that all aspects have been taken into account. The stages of development are as follows:
- Organisational Intent and Commitment - this includes the overall objective of the plan, how it is to be developed and who is involved.

- Risk Management - this process is to determine the risks that are inherent at each operation and to rank them. It is expected that an operation would have a number of different risks and as such would develop customised plans for each one.

**Fig. 1 – Emergency Preparedness**

- Emergency Measures - each emergency situation starts from a state of normality which deteriorates with time. This may occur quickly or over several years. The critical element here is to determine measures and limits that indicate early and/or critical points in the development of the different emergencies.

- Emergency Organisation - this is the details on the personnel who are responsible and will respond to the emergency occurrence as it develops.

- Facilities, Equipment and Personnel - this stage lays out the information on underground and surface layout, allocation of resources, what equipment is to be made available and who is capable of performing each functions or using these resources and equipment.
• Training and Assessment - this lays out the system of developing and maintaining each persons skills as they are required under the plan.

• Trial and Simulation - this test the system and the personnel which results in the areas of improvement being identified.

• Auditing, Risk and Capability Assessment - this completes the chain with reviews and updates being carried out to determine current and future suitability of each of the plans. At this stage you also check that the plans still cover all of your probable risks.

QUEENSLAND REQUIREMENTS FOR EMERGENCY PREPAREDNESS PLANS ARISING FROM MOURA NO. 2

The Moura No2 Wardens Inquiry made a number of recommendations which are being progressed by both the Moura Task Groups and the Queensland Inspectorate who are working in conjunction.

Queensland Legislative Changes

Legislation for the Queensland Coal Mining Industry now requires operations to have a 'Safety Management Plan'. These plans are to cover all aspects of the mining operation and shall include a ventilation plan, gas management plan, spontaneous combustion plan and emergency response plan. Each plan is to have its own limits and indicators which eventually may escalate to a major emergency.

Each underground coal mine in Queensland is required to have a contract of agreement with the Queensland Mines Rescue Service as a result of the new Mines Rescue Act which came into force on 1st January, 1998. This agreement covers the numbers of personnel to be trained in mines rescue, topics to be trained in, equipment required, maintenance and the call out / activation system.

Both the 'Safety Management Plan' and the 'Mines Rescue Contract' are to be developed in conjunction with each other so that a complete system is developed.

2 task groups have been set up to study aspects arriving from the Moura disaster

Moura Task Group 4 - Mines rescue strategy development

Task Group 4 was divided into five (5) sub-groups to progress some of the Moura No2 Warden Inquiry recommendations. These sub-groups covered :- the following matters

1. Self escape and aided rescue management plans and training

2. Communications - alerting personnel, systems and personnel monitoring

3. Gas Management - tube bundle, telemetric systems, chromatography and boreholes

4. Aided Rescue - mine exploration, rescue vehicles, monitors and communications for rescue teams, refuge chambers and large diameter boreholes.

5. Incident management, ventilation modelling, emergency preparedness guidelines and knowledge based incident management systems.

These Task Group 4 sub-groups are to be dealt with in detail by Jakeman, (1998).
Synopsis

Both the Queensland Inspectorate and the Moura Task Group 4 are developing detailed guidelines for operators utilising the latest technology which will assist them in developing their own emergency preparedness plans.

These developments will cover the following recommendations from the Moura No2 Warden Inquiries:

- spontaneous combustion management;
- mine safety management plans;
- training and communications;
- self-rescue breathing apparatus;
- emergency escape facilities;
- gas monitoring system protocols;
- sealing - design and procedure;
- withdrawal of persons; and
- inertisation.

NSW CMRA 1997 DRAFT REGULATIONS - EMERGENCY MANAGEMENT SYSTEMS

Background

The NSW Joint Safety Review Committee (JSRC) has completed the draft CMRA Regulations, distributed them for Industry comment, received submissions, reviewed these submissions, made the appropriate alterations and has them with the legal draft-persons for finalisation.

Draft regulations

Under the proposed new regulations the Manager is required to have Management Systems for:

1. Surface fire prevention and control - (General regulation).
2. First Aid - (General regulations).
3. Emergency provisions - (Underground regulation).
4. Explosion suppression - (Underground regulation).
5. Inspection of all places in a mine - (Underground and Open Cut regulations).
6. Use of explosives - (Underground and Open Cut regulations).
7. Mechanical and Electrical staff (alternative) - (U/G and O/C regulations).
8. Control of ventilation - (Underground regulation) including
• spontaneous combustion management system.
• gas management system.
• design, monitoring and control system.
• other hazards management system.

Format for Developing the Management System

The manager must develop and document a system wherein minimum standards are prescribed for which such that

1. employee representatives with appropriate skills and experience must be consulted;
2. training must be provided;
3. periodic audit and review is required; and
4. the District Inspector and Check Inspector must be notified within 7 days.

Management system for emergency provisions (underground) which shall include:-

• The Mine manager is required to develop and implement for the underground parts of the mine a

6. Mine manager must identify emergencies that may occur
7. Must cover such occurrences as fire, fall, pollution of mine air or inundation
8. System must address but is not limited to :-

• actions to be taken by a person finding a mine fire;
• the escape of persons affected by an emergency;
• treatment and transport of injured persons; and
• procedures to be adopted when external emergency service required.

8. An emergency system must include, as a minimum, provision for :-

• at least 2 means of egress
• effective communications to all personnel of egress from each part of mine
• a means of making persons familiar with egress paths
• marking of egresses such that they can be easily followed in poor visibility
• sufficient type and number of transport or alternate means and escape equipment to allow the safe evacuation

• a competent person on duty on the surface with effective communications

• the rapid and effective sealing of the mine - allowing for re-opening

5. Regard of any relevant guidance material from Chief Inspector

6. Must be documented and at mine

The need for detailed plans and information on current technological for NSW mine operators is the same as the requirements for Queensland. The Moura Task Groups have joint Queensland and New South Wales membership and are the current vehicle being used by both States. This should lead to a common approach between the States although it is unlikely that a common Coal Mining Act and Regulations will occur.

EMERGENCY PREPAREDNESS AND MINES RESCUE GUIDELINES

Background

In 1994 the NSW Mines Rescue Act was rewritten and the previous regulations revoked. The basic guidelines of the previous regulations continued to be adhered to with variations made as Mines Rescue Board NSW policy.

On the 28th June 1995, there was an explosion at Endeavour colliery. All mine personnel were safely accounted for but due to insufficient environmental data from the effected panel it became difficult to re-enter the mine. During the subsequent rescue operations there was some debate over procedures to be used and limits to be established.

These two events lead to the development of the Emergency Preparedness and Mines Rescue Guidelines by the Mines Rescue Board NSW (1997).

The draft document has been circulated throughout the mining industry for comment and discussion and was endorsed in principle by the Mines Rescue Board NSW in May 1997.

The document is to be reviewed every six months to allow for changes in technology and the evolution of mines rescue practices. Queensland Mines Rescue Service have reviewed and evaluated the Guidelines with a number of variations having been incorporated. There are still some alterations recommended by Queensland Mines Rescue which will be part of the first review in March 1998. It is still the objective of both States to have a common set of Guidelines.

The guidelines cover:

1. Glossary and abbreviations.

2. Intent.

3. Functions and Responsibilities of the Mines Rescue Board.

4. Functions and responsibilities of the Mines Rescue Service.

5. Roles and Responsibilities of the Mines Rescue Brigade and others.


8. References

- nature and intensity of incident.
- risk categories.
- explosibility.
- overseas criteria for entry of mines rescue brigadesmen.
- toxicity.
- oxygen deficiency.
- heat and humidity.
- Visibility.

9. Procedures for the deployment of Rescue Brigades

- response by less than 5 persons.
- brigades of 5 or more persons.
- establishment of the Fresh Air Base (FAB).
- the standby team.
- coupling-up inbye the FAB.
- return to FAB.
- operational times.
- succession plans.

The Emergency Preparedness and Mines Rescue Guidelines detail the roles, functions, structure and responsibilities of personnel involved in an emergency situation. The reference section gives current technical views on critical environmental issues. The procedures section gives detailed rules on how rescue personnel can operate.

These guidelines will continue to evolve over time and as technology advances. They are also meant to be used by coal mine operators to assist them in establishing emergency limits / triggers and in developing emergency plans.
CMQB EMERGENCY PREPAREDNESS AND MINES RESCUE PROGRAM

Background

In 1996/97 the Mines Rescue Service NSW (MRS) developed its Coal Mine Emergency Response and Rescue Standard for the training of its employees and Brigadesmen (see Appendix 1). The competencies for the Underground Rescue and Response were developed in line with the National Training Board requirements and registered with the National Mining ITAB.

The MRS is well advanced in the development of the Surface Rescue and Response Standards. In addition, there has been another level developed which is for the First Responder.

In 1996 the Coal Mining Qualifications Board (CMQB) NSW approached MRS to provide a course for persons wishing to sit for the Managers and Undermanager Certificate of Competency. This was to bring the certificates prerequisites in line with Queensland, where a candidate must have held a Mines Rescue Certificate of Competency.

The NSW CMQB reviewed the competencies required for Level 4 - Incident Management and Emergency Preparedness under the Underground Rescue and Response Standards and deemed them appropriate for Manager and Undermanager candidates. This course is currently being reviewed by the Queensland Board of Examiners.

Course structure

The course is now registered with the National Mining ITAB, comprises eight training modules and is conducted on a full time bases over seven days and includes the following modules which can be conducted as stand alone units of competency, shown in Table 1.

Table 1 – Coal emergency preparedness and mine rescue

<table>
<thead>
<tr>
<th>MODULE</th>
<th>DESCRIPTION</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.0 Basic Mines Rescue And Fire Fighting</td>
<td>To provide participants with a working knowledge of mines rescue and fire fighting procedures, and the physical experiences of these in a simulated mine situation.</td>
</tr>
<tr>
<td>2.0 Gas detection and interpretation in an Emergency</td>
<td>To provide participants with adequate information on gases present in the underground environment to enable effective and safe incident control.</td>
</tr>
<tr>
<td>3.0 Risk Management and Assessment</td>
<td>To provide participants with an understanding of the principles of risk management in a coal mine context, and to apply a risk management model to identify hazards and risks, and implement appropriate controls.</td>
</tr>
<tr>
<td>4.0 Emergency Preparedness and First Response</td>
<td>To provide participants with an understanding of the necessity for emergency preparedness planning, and the skills to develop an emergency preparedness plan within a mine's organisational structure.</td>
</tr>
<tr>
<td>5.0 Ongoing Incident Management</td>
<td>To provide the skills and knowledge to plan for the ongoing management of an incident after an initial response.</td>
</tr>
<tr>
<td>6.0 Escape and Intervention Strategies</td>
<td>To equip participants with a working knowledge of escape and intervention systems.</td>
</tr>
<tr>
<td>7.0 Post-Incident Management</td>
<td>To provide an understanding of the considerations and procedures for long term, post-incident procedures.</td>
</tr>
</tbody>
</table>
Recognition of Prior Learning (RPL) will be granted for Module 1: Basic Mines Rescue and Fire Fighting to those applicants who have completed the Mines Rescue Brigadesmen's Certificate with the Mines Rescue Service NSW, or Queensland Mines Rescue Brigade equivalent.

People applying for RPL for any of the individual Modules 2-7 will be expected to show evidence of achievement to the Outcomes and Assessment Criteria of the particular module for which RPL is requested. If evidence for this is not available or incomplete applicants will be assessed against the assessment criteria for the module.

People applying for RPL for the Course, comprising Modules 2-7, will be expected to show evidence of achievement to the Competency Standard for the Course. If evidence is not available or incomplete Module 8 will be used for assessment of the applicant’s competence.

Assessment Method: The training course is competency-based. The Assessment Criteria therefore reflect the Performance Criteria of the competency standards.

Formative assessment is based on a hypothetical minesite and an incident which takes place on that minesite. Participants will progressively apply what they have learned in each training module to demonstrate their ability against the assessment criteria of the module.

Summative assessment is facilitated through Module 8: Assessment Simulation. In this module the participant is provided with the full information about a mine incident and is asked to plan the management of the incident, show the way in which he/she would implement the plan, and the post-incident procedures which he/she would institute. This is assessed against the assessment criteria of the preceding modules.

To date, some thirty persons have completed the course with a total of twenty two persons being deemed competent. Three courses have been conducted for the following groups:-

- May 1997 - Manager and Undermanages candidates;
- September 1997 - Internal to NSW Mines Rescue; and
- September 1997 - Manager and Undermanager candidates.

The course has had extremely positive feedback from participants who have gained a detailed knowledge on how to develop an emergency plan. In achieving this result they also have a detailed appreciation of the current technologies strengths and limitations, rules that mines rescue personnel can be deployed and most importantly some of the lessons we have learnt from previous mining disasters.

CONCLUSION

Coal mining emergencies and disasters have been occurring at a rate which is disproportionate to the technological changes that have been occurring in the industry. We have a resourceful management group, a trained and intelligent workforce, improved monitoring and communications and yet we still seem to be reacting to occurrences instead of managing them.

Is this because we are just reactive to occurrences, not learning from the experiences and knowledge of others or is technology changing that fast that we are introducing new equipment and methods in areas that there is no knowledge and experience?

The direction in legislation, both coal mining and OHandS is for proactive planning and training with documented plans which are updated, tested, reviewed regularly and have ownership of the whole workforce. This new legislation is less prescriptive but it means that the mine operator must develop detailed plans which are appropriate for their operation.

As has been indicated in this paper, there are a number of specialised resources being developed to assist operators in the development of these plans. This ranges from plan structure, to research and development, to guidelines developed by
specialists. It is hoped that these will be used and that a proactive system which is strengthened from experiences from all coal mines can be implemented throughout the coal industry.

ACKNOWLEDGMENT

The assistance of Mr Paul McKenzie-Wood, Manager Coal Mines Technical Services (Mines Rescue Service NSW) in the critique of this paper is gratefully acknowledged.

REFERENCES


Mines Rescue Board NSW 1997, *Emergency preparedness and mines review guidelines*, Revision 0
APPENDIX 1

Coal mine emergency response and rescue standard

The competency standards for an Underground and Surface Coal Mine Emergency Response and Rescue organisation can be said to cover five functions to allow it to operate effectively. These functions are:

- Underground Response and Rescue;
- Surface Response and Rescue;
- Maintenance;
- Research; and
- Training.

These functions have four levels of responsibility and skill. These functions and their levels are shown on the matrix below:

<table>
<thead>
<tr>
<th>LEVEL</th>
<th>INCIDENT AND EMERGENCY PREPAREDNESS</th>
<th>MAINTENANCE MANAGEMENT</th>
<th>RESEARCH MANAGEMENT</th>
<th>TRAINING MANAGEMENT</th>
</tr>
</thead>
<tbody>
<tr>
<td>4</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>3</td>
<td>Fresh Air Base Control</td>
<td>Incident Control</td>
<td>Maintenance Planning</td>
<td>Training Coordination</td>
</tr>
<tr>
<td></td>
<td>Surface Control</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>Team Leadership</td>
<td>Team Leadership</td>
<td>Technical Maintenance</td>
<td>General Research</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Training Development</td>
</tr>
<tr>
<td>1</td>
<td>Team Membership</td>
<td>Team Membership</td>
<td>Routine and Service Maintenance</td>
<td>Training Delivery</td>
</tr>
</tbody>
</table>

FUNCTION

- UNDERGROUND RESCUE and RESPONSE
- SURFACE RESCUE and RESPONSE
- MAINTENANCE
- RESEARCH
- TRAINING
Emergency Escape Systems

R Bancroft

ABSTRACT

The Inquiry into the 1994 accident at Moura No. 2 Colliery included in it's findings that mine escape and rescue options for persons in underground coal mines were in need of review. The Inquiry recommended the establishment of industry working groups to report to the Queensland Chief Inspector of Coal Mines on matters including escape strategy and life support for escape from mines.

One of these task groups examined issues relevant to self-escape from Australian underground coal mines and formulated recommendations and guidelines in response to the terms of reference and scope.

A similar working group was also established in NSW by the Chief Inspector of Coal Mines to provide guidance notes for underground emergency escape systems.

This paper covers the major points arising from the work of both the Queensland and New South Wales working groups and the results of a study of overseas escape strategies that was undertaken by the NSW working group.

ESCAPE STRATEGY

Evaluation of the various factors involved identified a number of major elements that need to be addressed in the development of an emergency escape system to enable persons to escape safely from a mine following a mine fire or explosion.

Some of these elements are:

- Early Warning
- Self Rescue Apparatus
- Communications
- Guidance Systems / Lifelines
- Escapeways / Transport
- Refuge Chambers / Changeover Stations
- Training of personnel and
- Safety Management Plans for Evacuation

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Senior Inspector of Mines Queensland Department of Mines & Energy
The escape of persons underground will be enhanced by the use of a planned strategy that has been developed giving due consideration of these elements and recognition of the potentially difficult circumstances a person could encounter following an incident. Importantly the strategy will include the realisation that the mobilisation of rescue personnel could take time or may not be feasible due to the presence of potentially explosive or toxic atmospheres. The initial reactions of persons underground to an incident situation are a significant determinant on their survival. Planning, preparation and training for such emergencies are essential components required to improving the likelihood of survival.

**EARLY WARNING**

The role of an early warning system is to sense the first signs of fire or explosion and communicate an alarm so that evacuation of the mine or part of the mine can take place. Control measures taken at the earliest possible time would allow egress through reasonably smoke free escapeways and maximise effective escape.

Carbon Monoxide sensors and increasingly, smoke sensing systems, offer considerable potential for early and more reliable fire detection than do other available systems.

A control system must be established to receive and analyse data on the underground environment. The system must include decision making protocols and enable control to be maintained and action coordinated during an emergency.

Consideration should be given to the incorporation of a communication system throughout the mine that can be used to immediately notify underground employees in all areas of the mine of the need to evacuate. The system should have the ability to provide employees with incident details. Principal systems include telephone, traditional two-way radio, ground induction and leaky feeder.

Western Australian, Canadian and Mount Isa Mines metalliferous mines have introduced systems to release stench gas to the ventilation system to initiate emergency procedures.

Computer generated emergency alert systems are available where recorded messages can be transmitted to localities on an at risk basis. The Revmaux (HBL) mine in France utilises such a system with a maximum of 10 localities alerted at one time. An alert immediately triggers the escape procedure. Similar types of system are being used in Australian collieries.

The Personal Emergency device, referred to as “PED” through-the-earth system is capable of sending radio messages from the surface to wearers of receiving units on a mine-wide basis after an incident, but it’s utility is limited by an inability to return signals from the wearer to the surface. Medium frequency partially inductive systems (e.g. Rimtech, Taiheiyo) provide increased potential for survival after an incident because of the robust nature of the wave carriers used (pipes, cables etc.). Prototype units for locating trapped miners have been developed overseas but their application is limited to short range, direct line of sight and restrictive circumstances.

**SELF RESCUERS**

Filter type self rescuers were introduced into the coal mining industry in the 1960’s in response to many fatalities that had arisen due to conveyor belt fires (e.g. Creswell Colliery, UK, 1950 - 80 fatalities). They are only effective where sufficient oxygen is present in the atmosphere to support life.

The introduction of fire resistant anti-static conveyor belts, fire resistant oils, lesser use of timber supports and improved environmental monitoring technology has reduced the risk of major mine fires and hence the principal reason for the introduction of filter type self-rescuers.

An explosion occurring in the vicinity of a working face is now the principal hazard that may require the use of a self-rescuer.
In many explosions reviewed, due to the reduced oxygen content of parts of the mine atmosphere following explosions, the use of a filter type rescuer would not have enabled persons to escape.

For this reason, it is considered essential that all persons underground be equipped with a self-contained self-rescuer (SCSR), i.e. a self-rescuer that provides the wearer with respirable air.

There are many brands and types of SCSR's currently available.

These are mainly manufactured either in Europe, the USA or South Africa with each country having different testing and approval criteria. The only international standard currently available for the testing of chemical type (KO2) oxygen self-rescuers is EN 401 (BS/1993). The testing of compressed oxygen self-rescuers in Australia is covered by AS/NZ 1716 (DMR, 1996).

Because of the differing test criteria used, and the confusion that this creates when evaluating different brands, EN 401 has been recommended as the standard for testing of chemical oxygen SCSR's until an Australian standard, equivalent to EN 401, is developed.

Immediately following an underground mine explosion, visibility can be significantly reduced due to presence of smoke and dust, whilst the smoke and toxic gases present may also irritate the eyes. Disorientation may result from this lack of visibility, and combined with the lack of communication, serious limitations are place on the ability to effect escape.

South African research and experience with chemical oxygen SCSR's has shown that poor visibility and disorientation can reduce the distance travelled to 60% of that expected under normal conditions.

Many cases have been cited where persons have not been able to find their self-contained self-rescuer immediately adjacent to them (DMR, 1996).

Due to this disorientation and lack of visibility, it is essential that all people' underground carry an SCSR with them at all times.

Another factor that can play a major part in the escape of persons using self-rescuers is body mass. This subject is dealt with comprehensively by Paul Mackenzie Wood in his paper "Deployment of Self-Contained Self-Rescuers in Coal Mines".

There is a requirement in all Australian underground coalmines for the use of approved self rescuers. The minimum requirement is currently for filter type self rescuers in NSW and in Queensland self-contained self rescuers have been required since 1 January 1998.

COMMUNICATIONS

There is a need for a communications system that would survive an incident and provide ongoing two-way communications between escaping or trapped miners and rescue personnel on the surface. The system must be compatible with the type of self-rescue apparatus to be used and the likely escape or refuge options available to survivors. As power to the mine is likely to be interrupted during an incident, self-contained battery powered backup should be integral to the system. Whilst voice is the highest priority for transfer, systems which can also transmit data and video signals should be encouraged to assist the rescue process.

The minimum coverage requirement is for a communication system to be established along escape routes.

The location and tracking of all persons (and most vehicles) in underground mines should also be considered in any escape system. Effective two-way voice communication will contribute to this requirement but more efficient electronic systems should be pursued.
Formal management plans to support efficient use of communication system should be developed. This control plan should attempt to identify the range of possible incidents, early warning trigger levels and incident response actions. It should also determine who is to be communicated with, who communicates messages and what information or instructions are to be given.

**ESCAPEWAYS**

Rescue response following an incident involves a period of time that, in most circumstances, requires persons underground to attempt an organised escape, rather than await rescue. In Australian collieries, the distance from the working face to the surface can be considerable, and in many cases the seam grade can be quite steep. These escape route difficulties, allied with the expected problems of disorientation and poor visibility, give rise to a requirement for a roadway to be established in each mine that meets the criteria of good trafficability.

This roadway should, as far as practicable, be capable of maintaining a respirable atmosphere that is free from fumes and airborne dust, after an explosion or fire. To achieve this, the escapeway should be an intake airway, protected from damage by being segregated from other roadways, with stoppings capable of withstanding low intensity explosions.

Use of the term “second means of egress” is commonly applied to return airways, with little thought being given to which is the most desirable escape route. In emergency exercises involving different scenarios, employees invariably attempted to escape via the returns, even when this may have been the most inappropriate route. The concept of “second means of egress” as the primary escape route should be replaced by the concept of an “escapeway”.

Mine management should carefully consider which airway would make the most suitable escapeway. Because of the need to maintain a respirable atmosphere, the risk of fire in this roadway should be reduced to a minimum. This could be achieved by restricting the use of equipment in this roadway to those items that are either fitted with fire suppression devices, or which incorporate a fail safe system to prevent the outbreak of fire.

Vehicular escape would, in most circumstances, afford the best chance of persons making a rapid escape from the mine, and escapeways should be designed to maximise the likelihood of facilitating vehicular escape, without precluding or endangering passage by foot.

**GUIDANCE SYSTEMS**

To assist in gaining access to escapeways, and in guiding persons along escapeways in conditions of low visibility, clear guidance systems that will survive an incident are required. Knotted ropes with directional cones fitted (lifelines) have been developed for this purpose. More recently, battery-powered guidance systems, such as the “MOSES” system used in South Africa incorporating directionally discriminating audible pitches and flashing LED’s have been developed to provide clearer guidance.

**CHANGE-OVER STATIONS**

Dependant upon the distance of the working areas from the surface and the duration of any self contained self-rescuers (SCSRs) to be carried, the provision of underground caches of SCSR must be considered to facilitate the escape of persons to the surface. The number and separation distance between caches should be based on the assumption that the mine atmosphere is irrespirable all of the way to the surface, and that visibility throughout the mine will be very poor.

Caches installed throughout a mine should be constructed so that they are protected from the effects of low intensity explosions. Persons exchanging SCSR’s should be able to do so in a safe manner. This could be accomplished by being able to exchange SCSR’s in irrespirable atmospheres with minimal risk to persons or by the provision of changeover
stations equipped with respirable air. Consideration should also be given to equipping changeover stations with communication facilities, capable of surviving an incident, to facilitate escape co-ordination.

In addition to designated caches located at strategic locations in the mine, consideration should be given to the provision of either a cache of SCSR’s or some other system of respirable air, on board personnel vehicles. Compressed air systems are now available, comprising a storage cylinders and a number of face masks connected to a common supply regulator, that could meet this need.

Refuge chambers

Refuge chambers have an accepted place in rescue strategies in South African coal mines where workers are instructed to make their way to the section refuge chamber. This is mainly due to the large area extent of the mine workings, the generally shallow depth of workings (enabling borehole recovery in the event of a disaster) and the differing cultural backgrounds and experience of the mine workers.

The majority of opinions sought on the use of refuge chambers in Australian coal mines indicates that Australian coal miners, in the absence of incident information or training, would attempt to reach the mine surface rather than stay underground in a Refuge Chamber.

In the first instance, escape systems should be provided to enable persons to escape to the surface of the mine or alternative place of safety. Operators should, however, examine their own circumstances and possible scenarios to ascertain whether or not there is a place for refuge chambers in their Self Escape Management Plan.

Current thinking indicates that it is very unlikely that rescue teams will be sent into a mine where explosive or toxic concentrations of gas are present. Miners will generally need to effect their own rescue.

For this reason it is believed that regardless of whether a Refuge Chamber or a Change Over Station is used, the system should be mainly designed so that miners have a safe place to assemble.

The Refuge Chamber or Change Over Station should preferably be supplied with a respirable atmosphere and means of communication to the surface so that people can plan their escape and change from one self rescuer to another in safety.

When designing the system it should also be recognised that some persons may not be able to escape due to injuries sustained, or health and fitness factors. Consequently, consideration should be given for the provision of arrangements to enable these persons to be safely refuged whilst facilitating the timely escape of uninjured and fit individuals.

Training

Provision of oxygen self rescuers, early warning systems and escapeways will be of limited value unless the people attempting escape can make the appropriate decisions when confronted with an emergency situation. It is essential that all mineworkers be given adequate and regular training in all aspects of the mine escape system.

Training exercises should entail more than just travelling through the second means of egress or escapeway.

A feature of both USA and South African mineworker training is participation in regular evacuation exercises, often under simulated conditions of disorientation or low visibility.
EVACUATION MANAGEMENT PLAN

Consideration of all the various aspects of the mine when examined in the light of the previously enumerated factors should be incorporated into a mine evacuation or Self Escape Management Plan.

The plan should be developed using the criteria established in guidelines for Queensland Safety Management Plans or the New South Wales Risk Management handbook for the Mining Industry MDG 1010.

When properly implemented this would provide all persons underground with the capability to reach a place of safety, recognising the difficult environmental conditions likely to be encountered following an underground incident.

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Deployment of Self-Contained Self Rescuers in Coal Mines

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ABSTRACT

Field trials at three New South Wales and one Queensland coal mines were carried out to gather data on oxygen “run out” times of Self-Contained Self Rescuers (SCSRs), time taken to escape from the mine, distances travelled and the average heart rate of subjects wearing SCSR. The study has led to a method of predicting the duration of oxygen supply from a SCSR as a function of the wearer’s body weight, physical fitness and the prevailing environmental conditions. Escapeway design, planning for emergencies, familiarity with SCSR and experiential escape training are critical to control panic and maximise the likelihood of survival of a person attempting to escape in an emergency involving fire or explosion.

INTRODUCTION

The filter self rescuers (FSRs) currently approved for underground use in Australia are not designed to function in an oxygen deficient environment and accordingly cannot be relied upon to save lives, particularly in situations where there has been a fire or explosion in a mine. The function of FSRs is to remove low concentrations of carbon monoxide (up to 1.5%) from the inhaled air (Strang and Mackenzie-Wood, 1990). FSRs do not produce any oxygen. In order to escape from an irrespirable atmosphere to a respirable zone, a miner must be provided continuously with sufficient oxygen.

In the light of the recent fatalities experienced in the Australian coal mining industry, such as the 1994 Moura No.2 disaster in which 11 lives were lost underground, there was a general consensus for the need to legislate SCSR to replace FSR. The recommendations of the Moura No.2 inquiry included the following extract:

“….. development and introduction of oxygen based escape systems from underground coal mines, a means to maximise the likelihood of survival, in the event of fires or explosion”.

In Australia, SCSR are used for special purposes, for example when working in gas outburst prone mining conditions, and in recent years SCSR have replaced the FSR under these conditions. In the State of New South Wales, an exemption is required if a colliery wishes to replace the FSR as compulsory equipment to be worn by each person who wishes to proceed underground. The exemption will only be granted if the mine manager can demonstrate that each person is provided with full protection through the changed arrangements. The deployment of SCSR in Queensland as approved standard units came into effect in January 1998 (The Moura Task Group 4, 1996).

This paper discusses a study, which was conducted to develop procedures for an introduction of the SCSR into Australian coal mines (Mackenzie-Wood et al, 1997). The other study objectives were:

1. To evaluate the influence of personal factors such as age, weight, physical fitness on individual’s oxygen requirements to escape from and underground place of work to a place of safety.

2. To evaluate the capability and comfort of SCSR.

3. To derive a method of predicting an individual’s oxygen requirements to escape from and underground place of work to a place of safety.

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SELF-CONTAINED SELF RESCUERS (SCSR)

There are two types of SCSRs available commercially, compressed and chemically produced oxygen. The compressed oxygen type supplies oxygen to the wearer on demand from a cylinder of high pressure oxygen, the carbon dioxide in the exhaled breath is removed by a soda lime or alkali canister. The chemically produced oxygen type uses a chemical, potassium superoxide (K\(\text{O}_2\)) that reacts with the moisture in the wearer's breath to produce oxygen and potassium hydroxide. The potassium hydroxide then chemically removes (scrubs) carbon dioxide. Both reactions are exothermic, causing the canister to become hot. Heat exchangers made of wire mesh are usually fitted in the breathing tube to reduce the inhalation temperature.

Coal mining industries in NSW and Queensland require that SCSRs comply with Australian Standard (AS) criteria where applicable. Compressed oxygen SCSRs must meet the requirements of AS 1716 - 1994 and the duration is determined when the criteria for inhalation temperature, breathing resistance, inhaled CO\(_2\) and oxygen cylinder pressure is exceeded.

Currently there is no Australian Standard for chemically produced oxygen SCSRs, the standard used for assessment is the British and European Standard BS/EN 401:1993. The criteria for determining duration in this standard is:

- Maximum inhalation temperature \(\leq 55^\circ\text{C}\)
- Exhalation/inhalation resistance \(\leq 1.3\text{kPa}\)
- Inhaled CO\(_2\) \(\leq 1.5\%\)

Manufacturers currently favour the chemical oxygen type. Chemically based SCSRs require little maintenance due to the absence of gauges, pressure reducers, lung demand valves and other moving parts. However, powdering of the K\(\text{O}_2\) granules may occur with vibration caused by carrying or transporting the SCSR. This may cause a path of low resistance through the chemical bed leading to a premature breakthrough and build up of CO\(_2\) in the wearer's breathing circuit.

The Portal-Pack\textsuperscript{TM} Self-Contained Self Rescuer was selected for the project because it was the only unit of 60 minutes duration that had met the requirements of BS/EN 401:1993 (based on the NSW Department of Mineral Resources Tests) and had been approved for use in NSW underground coal mines. The Portal-Pack is a single-use, self-contained closed-circuit breathing apparatus (Fig. 1).

![Fig.1 - Portal-pack\textsuperscript{TM} self-contained self rescuer](image-url)
Its operation is completely independent of the surrounding atmosphere. Once properly donned, the SCSR can assist a miner to escape from an area containing smoke, toxic gases or an oxygen deficient atmosphere. Its operating life during escape depends on the demands of the user. Two chemical sources within the unit release the life sustaining oxygen.

The initial source of oxygen is from a cylinder containing the chemical Sodium Chlorate \((\text{NaClO}_3)\), commonly known as the “chlorate candle”. Its function is to provide an immediate source of oxygen to fill the system including the breathing bag. When the breathing tube is pulled at the time of donning the apparatus, a primer cap initiates the chemical reaction which “burns” the chlorate to produce oxygen according to Equation 1:

\[
2\text{NaClO}_3 \rightarrow 2\text{NaCl} + 3\text{O}_2 \tag{1}
\]

Over the first two to three minutes, the chlorate candle produces roughly 10 litres of oxygen. The released oxygen partially fills the breathing bag. Once the individual commences to breathe into the unit the chemical reaction of the wearer’s breath and the potassium superoxide will produce more oxygen continuously. If the “Chlorate Candle” fails to initiate, the unit can still be used and the breathing bag became fully inflated by the end of seven minutes.

The second source of oxygen is a canister containing potassium superoxide \((\text{K}_2\text{O}_2)\). This chemical consists of coarse granules held in place by baffles contained in the canister. The chemical reaction between moisture in the exhaled breath and the \(\text{K}_2\text{O}_2\) liberates the oxygen. The Portal-Pack contains about 600 gm of \(\text{K}_2\text{O}_2\), which produces approximately 140 litres of oxygen.

The chemical reaction is given in Equation 2:

\[
4\text{K}_2\text{O}_2 + 2\text{H}_2\text{O} \rightarrow 4\text{KOH} + 3\text{O}_2 \tag{2}
\]

In addition to this, a second reaction takes place between the potassium hydroxide and the carbon dioxide in the external breath to combine and retain \(\text{CO}_2\) according to the following Equation 3.

\[
2\text{KOH} + \text{CO}_2 \rightarrow \text{K}_2\text{CO}_3 + \text{H}_2\text{O} \tag{3}
\]

The above reactions are self-regulating. The harder the wearer works the more oxygen is generated and the more \(\text{CO}_2\) is removed. The basic operation of the unit is depicted in Fig. 2.

![Fig. 2 - Operation of SCSR](692)
The duration of a SCSR is the time taken for the oxygen supply to run out and is indicated by the complete collapse of the breathing bag. The duration of a SCSR was expressed in Equation 4.

\[
\text{SCSR Duration (minutes)} = \frac{\text{Useable Oxygen (litres)}}{\text{Oxygen Consumption (litres/minute)}} \times 100 = \frac{T.Rr.}{V_{O_2}}
\]  

(4)

Initially, the wearer is likely to experience increased breathing resistance. As the chemical becomes exhausted, carbon dioxide (CO₂) builds up within the circuit and the wearer may develop a headache or light headedness.

**MINE SIMULATED ESCAPE TRIALS**

The objectives of the underground investigations were to gather in-mine data on escape times, distances travelled and average heart rates and to develop a technique to predict how much oxygen was actually needed for an average miner to escape from an underground mine. Underground simulated escape trials were conducted at South Bulli, Elouera and Myuna Collieries located in NSW and Crinum Colliery in Queensland. The mines were selected to represent the variety of conditions that are normally encountered in the actual escapeways of Australian coal mines. The escape routes were selected by the mine and based on the following requirements:

1. a distance which would require the volunteers to walk for a minimum of one hour, and
2. an established escapeway to simulate typical underground escape route conditions.

Attempts were made to ensure that the profiles of the volunteers represented those of the current workforce in Australian underground coal mines. Prior to participating in the field trials, all the volunteers were assessed by an occupational physician to identify those with significant medical conditions or those who may have problems walking out of the mine because of significant musculoskeletal problems. None of the volunteers were considered "unfit" to participate in the study.

The age distribution of the 37 subjects from South Bulli, Elouera, Myuna and Crinum Collieries is shown in Fig. 3, where 81% of all the subjects were between 30 and 49 years old. The minimum and maximum ages of the subjects was 27 and 57 years old with an overall average age of 40.7 years.

![Fig. 3 - Age Distribution of the 37 Subjects](image)

The weight distribution of the 37 subjects is illustrated in Fig. 4, where 87% of the subjects weighed between 70 and 99 kg. The minimum and maximum weight of all the subjects are 66 and 130 kg with an average of 85.5 kg. The weight and age
distributions of the selected volunteers represented the profiles of the current workforce employed in NSW and Queensland.

![Weight Distribution Chart]

**Fig.4 - Weight distribution of the 37 subjects**

**FIELD TRIALS**

Prior to wearing the SCSR each volunteer participated in a training program. At the conclusion of training all volunteers had to correctly don and wear training SCSR. Training was conducted by demonstration, lecture and video.

The content of the training program included information and instruction on the following:

1. components and principles of operation of the SCSR;
2. donning procedures;
3. expectations while wearing;
4. recognition of oxygen "run out"; and
5. symptoms of carbon dioxide retention and oxygen deficiency.

During the simulated escape trials, the 37 volunteers walked along the escape route at their respective mines on Day 1 carrying the SCSRs on their belts. The walk was repeated on Day 2 with the same subjects wearing the SCSRs. As the route in each mine had been identified by the mine as an established escapeway, the physical conditions were realistic with uneven ground, roadway water, overcasts to climb over, low roof conditions, inclines, pit furniture to negotiate and warm conditions. Each mine escapeway was different. The walking pace on each day was kept at a constant rate of about 2 km/hr. The heart rates of each subject was recorded by a Polar Vantage NV™. The heart rates for one of the subjects is reproduced in Fig. 5.

At the end of Day 2 each SCSR’s breathing bag was monitored by an observer to enable the oxygen “run out” time to be determined. Normally, the end of the trial could be defined for each individual as the point of complete collapse of the breathing bag, which may be accompanied by increased breathing resistance. If carbon dioxide builds up in the breathing circuit headache and light headedness may also occur. This is most likely to be caused by inhaled concentrations of CO₂ of 2-3%. The duration of the SCSR was taken from the time the individual started to breathe oxygen via the SCSR, ending with the complete collapse of the breathing bag.
DISCUSSIONS OF RESULTS

Performance and comfort of the SCSRs

All the 37 volunteers were asked to complete a questionnaire designed to assess the performance and comfort of the SCSRs.

The following conclusions could be drawn from the questionnaire.

(i) All the subjects felt they could don and wear the SCSRs in an emergency;
(ii) 95% felt the SCSRs would protect them in oxygen deficient or toxic atmospheres;
(iii) 46% felt the SCSRs were uncomfortable to wear around the head and neck;
(iv) 84% would not like to carry the SCSR for a complete normal working shift;
(v) 89% found the nose clips uncomfortable but appreciated the need for a tight nasal seal to prevent the ingress of irrespirable atmospheres;
(vi) 38% felt the temperature of the inhaled air was comfortable to breathe and 54% found the air hot but tolerable; and
(vii) 19% found breathing comfortable and 56% found breathing resistance noticeable but tolerable.

All volunteers were advised to carry gloves or rags to place between the canister and the chest wall, despite this precaution a small number of subjects had observable evidence of superficial skin scalding.

Oxygen “run out” time

The oxygen “run out” time distributions for the 37 subjects are shown in Fig. 6. In 60% of individuals, the duration of the SCSR equalled or exceeded the 60 minute nominal duration time of the SCSR. Of all the subjects, 11% has duration greater than 70 minutes and 8% less than 50 minutes. One individual ran out of oxygen in 45 minutes.
Predicting oxygen consumption

Various studies in USA (Bernard, Kamon and Stein, 1979; Berry, et al., 1983; Buskirk, Nicholas and Hodgson., 1975) have linked the oxygen consumption ($VO_2$) to average heart rate (HR) and the body weight (W) as per the following equations.

**PSU Model**

$$VO_2 = \frac{HR - 66}{36}$$

**Foster Model**

$$VO_2 = 0.024HR - 1.54$$

**NIOSH Model**

$$VO_2 = \frac{W(HR - 61.25)}{3230}$$

Using the above three equations, Day 1 average heart rates and the body weights of the 28 subjects, from the South Bulli, Elouera and Myuna trials, the average oxygen consumption rate and hence oxygen “run out” time for each subject was estimated. The results predicted by the three models as well as the observed are presented in Fig. 7. The three models generally underestimated the average $VO_2$ and hence overestimated the oxygen “run out” time for average heart rates under 120 bpm (fitter individuals). For heart rates above 120 bpm the predictions were inconsistent. There is a strong possibility that the above models were developed under conditions which were different from the Australian simulated field trials in this study.

In order to achieve better prediction capability for oxygen consumption, pre-trial and field data were statistically analysed. The data was examined to determine if there were any significant relationships between $VO_2$ and average heart rate, age, weight, smoking habits, drinking habits, physical fitness estimates based upon bicycle ergometer tests, exercise habits, previous breathing apparatus experience and other factors.
Fig. 8 shows that there is a tendency for lower average heart rates and body weights to be associated with lower values for \( \dot{V}O_2 \). Higher exercise rating, greater than a score of 5, was generally associated with lower \( \dot{V}O_2 \). The coefficient of correlation between observed \( \dot{V}O_2 \) and weight is 0.78. This suggests a strong linear relationship between observed \( \dot{V}O_2 \) and weight, in other words weight is a good predictor of \( \dot{V}O_2 \). In practice, a correlation coefficient over 0.7 shows a strong linear relationship between the variables, 0.3-0.7 is moderate and less than 0.3 is considered as a weak association.

The associations between \( \dot{V}O_2 \) and other measured variables could be described in statistical terms as follows:

(a) There was a strong association between \( \dot{V}O_2 \) with body weight;

(b) Exercise rating (habits) was moderately associated with \( \dot{V}O_2 \);

(c) A weak association existed between \( \dot{V}O_2 \) and both cigarette use and alcohol consumption; and

(d) The association between \( \dot{V}O_2 \) and age was very weak.

It was observed that the average heart rates for each of the 37 subjects were not the same on Day 1 and Day 2. In fact the average heart rate was slightly higher on Day 2 in 60% of the subjects. The correlation coefficient of average heart rate on Day 1 and Day 2 was found to be 0.79.

Both the compiled medical data and the simulated escape field data were statistically analysed. Based on the mine simulated escape trials from the first three mines, the Equation (4) was developed as

\[
\dot{V}O_2 = \frac{7.5W + HR}{500} + 0.043
\]  

(4)

A comparison of prediction of \( \dot{V}O_2 \) on the fourth mines' (Crinum) trials using the previous three models as well as Equation (4) is depicted in Fig. 10, indicating that the Equation (4) gives the best predictive values. All the four mine data were finally used to produce the Equation (5) referred to as the "University Of Wollongong (UOW) model".
The above predictive UOW model relates oxygen consumption ($V\text{O}_2$) with average heart rate (HR) and body weight (W) and is based upon a representative group of Australian male underground coal miners. This model appears to be of better
predictive value than the previous models, therefore a better estimate of oxygen consumption and hence predicted oxygen “run out” time. The above predictive model is recommended for use by Australian collieries. The model is simple and requires minimal technology. It requires a device to measure average heart rate during a simulated escape in an established escapeway, an accurate set of bathroom scales and a calculator.

Equation (6) relates oxygen consumption (\( \dot{V}O_2 \)) with average heart rate (HR), body weight (W) and exercise rating (ER) of an individual.

\[
\dot{V}O_2 = \frac{5.5W + HR - 115ER}{500} + 0.549
\]  

(6)

Equation (6) requires standardisation with an inter- and intra-observer reliability study before its utility can be established. The method would require the assistance of a health professional experienced in assessing the quantity and quality of an individual’s exercise history. The method is based upon the subject’s weekly exercise habits, intensity of exercise and scored on a scale of 1 to 10.

South African work on smoke has found that dense smoke reduces the average speed of travel and hence the breathing rate of the SCSR wearer. However, one effect does not off-set the other and distances covered in a dense smoke were found to be a low as 60% of those achieved in field trials in clear smokeless conditions (Keilblock, 1997).

CONCLUDING REMARKS

Using results from field simulated escape trials, a linear model has been developed to predict oxygen “run out” times of SCSR. While oxygen consumption is related to the amount of work performed, this study has demonstrated that personal factors can influence the amount of oxygen consumed while wearing a SCSR. Considerable physiological research has established that fitter and younger persons tend to utilise oxygen more efficiently in the body’s microscopic tissues. This results in less oxygen being consumed per unit of work. The following factors were apparent following analysis of the relationships between oxygen “run out” time and individual related factors:

1. Body weight has a major influence on oxygen consumption with heavier individuals likely to consume the SCSR’s oxygen more rapidly than smaller individuals, therefore it has the strongest predictive value for oxygen “run out” time.

2. Average heart rate is of moderate predictive value in estimating oxygen “run out” time. Average heart rate can reflect the terrain, burdens to be carried, roof height, obstacles to be negotiated (climbing), speed of travel, individual efficiency, physical fitness, heat and humidity.

3. Exercise habits are weakly predictive of oxygen “run out” time. However the utility of this factor is of uncertain value at this stage due to the need for interpretation of the exercise history and a scoring system which is subjective.

4. Better aerobic physical fitness significantly reduces oxygen consumption at the tissue level. However paradoxically, better physical fitness has little influence on oxygen “run out” time for SCSRs. As the SCSR is likely to produce oxygen in excess to the needs of the fit individual, the surplus oxygen is discharged to atmosphere via the pressure relief valve. Therefore the efficiencies in oxygen utilisation by the tissues associated with physical fitness are offset by losses to atmosphere. Overall the balance resulted in a minimal effect on oxygen “run out” time.

5. Age had a slight influence on oxygen consumption, however better physical fitness and lower body mass modified the amount consumed. Therefore age is of dubious predictive value.

6. Previous experience in wearing breathing apparatus appeared to have no significant value in predicting oxygen “run out” time.

7. Habitual consumption of alcohol and cigarettes may slightly increase \( \dot{V}O_2 \) and this would be modified by both physical fitness and body mass, therefore has negligible predictive value.
In life threatening underground incidents, as occurs following an explosion or during a fire, individuals are likely to walk quickly or run as a consequence of panic. As a result oxygen may be consumed more quickly or the chemical used less efficiently. Consequently there is a risk the duration of the SCSR will be reduced. It is this factor which will reduce oxygen "run out" time more significantly. Therefore if the rate of travel is controlled anxiety is unlikely to have a significant effect upon oxygen consumption. Anxiety may result in the release of hormones into the circulation (the fight and fright reaction) and subsequently increase the individual's heart rate. However unless large muscles, such as those used in walking and running are working or exercising there is unlikely to be a parallel increase in oxygen consumption in the tissues.

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Recent Developments in Coal Mine Inertisation in Australia

S Bell, D Cliff, P Harrison and C Hester

ABSTRACT
During 1997 two different types of coal mine inertisation equipment were trialed in separate underground coal mines in Queensland.

In April 1997 tests of the Polish developed GAG-3A inert gas generator were carried out at the Collinsville No 2 Coal Mine. These demonstrations indicated that the GAG-3A device was applicable in underground coal mines and that with respect to inertisation rate the device outperformed all other currently available techniques. The unit was assembled rapidly and operated safely during surface and underground trials and no mechanical problems were encountered with the device. The GAG-3A is not suitable for every fire scenario particularly if large volumes of fresh air are being drawn into the mine (where multiple units may be required), but with respect to the selection criteria developed by the Moura Implementation Process Task Group 5 criteria, the unit performed up to the specifications required.

In May 1997 tests were carried out at the Cook and Laleham Collieries using a Tomlinson diesel boiler. These demonstrations indicated that a low volume generator, such as the boiler, can proactively inert a large goaf area effectively. At Cook a volume of approximately 700 000 m$^3$ was inerted over 10 days and at Laleham an area of approximately 70 000 m$^3$ was inerted over 2 days. The boiler is not suitable for emergency use, due to the low flow rate of inert gas generated (approximately 0.5 m$^3$/s). There was no significant seam gas present during either trial.

Both test programs were financially supported by ACARP and the Department of Mines and Energy. BHP, Shell, and MIM also contributed to the GAG-3A project and Cook Resources Mining contributed to the Tomlinson boiler project.

INTRODUCTION
Any coal mine can be beset by problems associated with fires and explosions. Many techniques have been used to control these events with varying degrees of success. Inertisation in various forms has been used for many years and was last used in Queensland to some effect in 1986 when the NSW based Mineshield device vaporised 600 tonnes of liquid nitrogen, which was progressively injected into the post explosion atmospheres of the Moura No 4 coal mine.

Two projects were designed to test devices which either produce large volumes of inert (low O$_2$ concentration) gas from a commonly available fuel (Avtur) - the GAG-3A jet engine system or, low volumes of inert gas from a diesel fuel - the Tomlinson boiler. It should be understood that inertisation can be divided into two categories.

Category 1 - Low flow devices, typically 0.5 m$^3$/sec. This equipment includes:

- Pressure Swing Absorption (PSA) - Which uses chromatographic techniques to separate nitrogen and oxygen from normal air.
- Membrane technology - uses molecular diffusion through a membrane to separate nitrogen and oxygen from normal air.
- Distillation - Where air is liquefied and then fractionally distilled to produce oxygen and nitrogen.

1 Safety in Mines Testing and Research Station (SIMTARS)
• Tomlinson Boiler - boiler flue gas is produced by burning diesel fuel in a modified hot water boiler. The exhaust gas of the boiler is adjusted through combustion controls, to reduce the oxygen concentration to less than 1 %, with about 15 % carbon dioxide, 5 % water and the residue, nitrogen. A trace of carbon monoxide ( < 20 ppm) and unburnt hydrocarbons are also produced.

• Mineshield - Where liquid nitrogen is vaporised and injected into the mine. There are significant problems with respect to the use of liquid nitrogen in Queensland because of the comparatively low production capacity of the current nitrogen generating units and the logistics associated with transportation. This device can generate up to 5 m³/s of inert gas.

• Carbon Dioxide - liquid and solid - similar problems to the Mineshield operation in that the gas has to be produced by vaporisation and then heated to prevent density problems (CO₂ density is 1.5 than of air at the same temperature). Dry ice - solid carbon dioxide is particularly stable as it forms an ice skin than prevents volatilisation.

Category 2 - High flow devices, typically 20 - 30 m³/sec.

The GAG-3A jet engine system is currently the only operational device on the market, which will produce inert gas at these higher flow rates. Further research on very high flow devices (80 - 100 m³/sec) using a Pratt-Whitney TF30 jet is yet to be carried out and the significant size of this type of device would make its application in coal mines difficult. Fig 1 shows the assembled GAG-3A located in one of the portals of the trial mine and Fig 2 shows a schematic illustrating the main features of the unit. The jet engine burns the fuel consuming the oxygen and producing carbon dioxide and water, similar to the exhaust of the boiler. The jet requires an afterburner to complete the combustion process to the desired low oxygen exhaust concentration and water cooling to reduce the exhaust temperature and remove the thrust developed within the engine.

Fig. 1 - The GAG - 3A inert gas generator located at the portal to Collinsville No. 2 Mine
THE GAG-3A INERT GAS TRIALS

The Collinsville project was jointly funded by ACARP, the Queensland Government, BHP, and Shell. MIM made their recently closed Collinsville No 2 mine available for use by the research team. The project was organised to provide four demonstration days for Industry members to witness the use of the GAG-3A device. One day was assigned for the underground demonstration where the jet was located underground and operated to show the inertisation of a panel or small localised area. The other three days where allocated to surface demonstrations where the jet was connected to a portal seal and a large area of the mine. The objective of these latter tests was to rapidly inertise a small coal fire in a steel sled.

Mine layout

The mine was reopened by members of the Queensland Mines Rescue Brigade and was extensively instrumented by SIMTARS to enable real time remote determination of gas concentrations, air velocity, air temperature and relative humidity. Fibre optically linked television cameras were used to provide real time pictures of the movement of inert gas through the mine and also to view the fire sled during the coal burning phase of the tests.

Gas analysis data was provided by SIMTARS Mobile Gas Analysis Facility and the MIM Maihak tube bundle monitoring system. A high speed MTI gas chromatograph was available for more detailed analysis. Data loggers were installed underground to collect information from the sensor arrays and this data was displayed real-time on surface computers.

Figs 3 and 4 show the location of the monitoring systems used during the tests.
Fig 3. -Location of monitoring systems at Collinsville No. 2 Mine - first set of surface trials

OFRCEQ FAN NO. 1 (NOT IN USE, LOUVRES OPENED AS INTAKE)

IONITORINGPOINT GAS, BIPERATURE, 1.€1..0C/7Y AND HIPID/7Y

GASZA

FAN NO. 1 (NOT IN USE, LOUVRES OPENED AS INTAKE)

MONITORING ZONE

MAIN MINE FAN (ON ELECTRIC DRIVE)

MONITORING POINT
- GAS, TEMPERATURE, VELOCITY AND HUMIDITY
Fig 4. - Location of monitoring systems at Collinsville No. 2 Mine - underground trials
Fig 5.- Location of monitoring systems at Collinsville No. 2 Mine - second set of surface trials
The CSIRO mine exploration device, Numbat, was also used during the whole mine inertisation phase and useful corroborative visual data was obtained from this device.

Mine ventilation system

To facilitate the conduct of the demonstration at Collinsville No 2 mine, certain aspects of the mine ventilation system had to be altered for each phase of the trials.

Originally the mine had been ventilated by two axial flow fans located on the eastern and western sides of the main development headings. At closure, the mine was temporarily sealed by constructing Tecrete stoppings in 3 & 5 Headings (see Fig 3), by infilling 4 Heading portal, and Tecreting over the louvres on the two fans. When the mine was re-opened only 5 Heading was fully opened, although an access door was constructed in the stopping in 3 Heading portal and No 3 Fan on the eastern side of the mine was restarted. This provided sufficient ventilation to clear the mine of the predominantly carbon dioxide seam gas and later for the purposes of re-entering and working within the mine.

This however was not a satisfactory arrangement for the inertisation trials with the GAG-3A operating on the surface. For these trials it was intended to locate the GAG-3A in 3 Heading portal and to inert a large part of the mine between 2 and 5 Headings and down to 18 Level. To be able to do this it was necessary to seal 5 Heading at the surface and provide an alternative surface intake if the fan was to be kept running. This was achieved by opening the control louvres on No 1 fan (western side of the mine) without starting the fan. This allowed fresh air into the western returns which travelled to the mine fan across the inertisation zone via the overcasts at 6 & 7 Levels (see Fig 3).

For the first set of surface trials on 7th and 8th April 1997, No 3 Fan was run on its electric drive and measurements were taken underground to ensure the air quantity was sufficient to avoid stalling the fan. Generally the fan flows were about 90 m$^3$/s. A regulator in 3 Heading was used to admit a controlled quantity of fresh air past the GAG-3A as part of these trials.

For the underground demonstrations on 11 April 1997, care had to be taken in establishing a ventilation system that was safe and would prevent the potentially lethal GAG-3A exhaust gases from contaminating outbye occupied areas of the mine. For these trials the GAG-3A was located in 5 Heading, and the exhaust was ducted through a stopping between 14 and 15 Levels (see Fig 4). To provide airflow over the 2000 litre underground fuel pod in 13 Level, the stopping between 5 and 6 Headings was breached and airflow was controlled by a brattice regulator. Airflow over the GAG-3A engine unit was controlled by a sliding door at 14 Level, 5 - 6 Headings. To prevent the possibility of GAG-3A exhaust gas moving above 15 Level, brattice stoppings were constructed in 3 and 4 Headings, and the GAG-3A exhaust directed into the eastern returns by breaching the seal at 16 Level, 5 - 6 Headings. For these trials, No 3 Fan was operating on electrical drive, 5 Heading was fully open and the regulator in 3 Heading was fully open. As a further precaution, sentries provided by Mines Rescue and equipped with gas monitoring equipment and self contained breathing apparatus, maintained a watch on the brattice stoppings in 3 and 4 Headings to ensure no exhaust gas moved up dip against the ventilation. At no stage during the trials was there any indication of this occurring.

For the second set of surface trials conducted between 14 and 18 April 1997, the same basic configuration was used as in the first set of surface trials (see Fig 5). However, to reduce undesirable leakage of fresh air into the inertisation zone, it was decided to run No 3 Fan on its diesel emergency drive and to short-circuit the fan at the surface by opening the airlock doors. The regulator in 3 Heading was also sealed up with boarding and cement to reduce leakage. The combined effect of these measures was to reduce the airflow underground to about 30 m$^3$/s and to greatly reduce the potential for leakage into the inertisation zone.

As part of these trials a controlled underground coal fire in a steel fire sled was established. The inertisation trials would involve attempting to extinguish this contained coal fire. The location of the fire was in 5 Heading between 14 and 15 Levels, the same as the GAG-3A location for the underground trials (see Fig 5). The Tecrete stoppings built around the GAG-3A site at 5 Heading, 14-15 Level, and at 14 Level, 4 - 5 Heading, were both demolished so that the inert gases were directed over the fire site. The brattice stoppings in 3 and 4 Headings, 14 - 15 Level were left in place, and the breached stoppings at 13 Level and 16 Level were closed off.
Fig 6 (a - b) The spread of inert gas fan on
For the surface trial of 15 April 1997, it was decided to attempt to run the GAG-3A with the fan switched off. The ventilation configuration was otherwise identical to the other surface trials of that week.

After each of the surface trials, the inertisation zone was cleared by opening the 5 Heading stopping, the access door and regulator in 3 Heading, and closing the louvres on No 1 Fan. This caused the maximum volume of fresh air to enter the mine and effectively cleared the inertisation zone in time for the next trial. The mine was extensively monitored, using the original fixed Maihak system and SIMTARS Mobile Minewatch gas analysis laboratory, to ensure that all areas of the mine were safe for personnel entry. No personnel were allowed underground after a trial until the area was comprehensively inspected by a deputy.

**Experimental results**

Figs 6a - 6c depict the spread of inert gas with the fan operating through the sections of the mine covered by the demonstration. In a period of 360 minutes the bulk of the area of interest had the oxygen level reduced to below 12%. A similar exercise in 1986 at Moura No 4, in a smaller area, took 600 tonnes of vaporised liquid nitrogen and five days to achieve a comparable result.

Figs 7a - 7b depict a similar though slower outcome with the mine fan off. There were some problems with seal leakage and loss of inert gas at the surface, but the jet was still effective.
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Fig. 7 (a - b) The spread of inert gas fan off
Demonstration outcomes

The selection criteria for the trial developed by the Moura Implementation Process Task Group 5 Committee were met with the exception that output flow rates were slightly below the levels predicted (19 m$^3$/s against an expected 20-25 m$^3$/s). This diminution in flow rates was attributed to higher ambient air and water temperatures than expected from operation in more temperate climates.

<table>
<thead>
<tr>
<th>The Task Group 5 Criteria for the Demonstration were as follows:</th>
</tr>
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<tbody>
<tr>
<td>1 Unit can continuously deliver 20 m$^3$/s of oxygen deficient atmosphere.</td>
</tr>
<tr>
<td>2 When operating normally the unit is to operate without any evidence of external flame.</td>
</tr>
<tr>
<td>3 The unit is to deliver oxygen deficient atmospheres containing less than 5% oxygen.</td>
</tr>
<tr>
<td>4 Automatic shutdown of equipment following failure of the water supply system.</td>
</tr>
<tr>
<td>5 The unit is to demonstrate the ability to extinguish flames from an underground coal fire within specific operating conditions as determined from the trial.</td>
</tr>
</tbody>
</table>

The unit was assembled rapidly and operated safely during all aspects of the trial and no mechanical problems were encountered.

Over 100 industry stakeholders visited the demonstration and feedback questionnaires were generally positive. The demonstration supported the view that the device was suitable for coal mine use.

No external flame was visible on the device.

The unit produced noise levels in excess of 124 dB(A) (when measured 1 m from the jet) in both surface and underground operations.

Environmental noise levels measured 2.3 km from the GAG-3A were not influenced by the operation of the unit.

The simulated fire in the mine was extinguished by the inert gas from the GAG-3A. This was evidenced by temperature profiles and actual video footage.

The limited gas stratification experiment conducted indicated that the gas produced by the GAG-3A tended to move closer to the roof than the floor.

The trial demonstrated that the GAG-3A device has applications in underground coal mines and that it outperformed all other available technologies with respect to volume of inert gas produced.

It is clear that the GAG-3A produces lower oxygen levels over a wider range of excess air conditions and therefore has a much wider range of applicability than other available technology.

Conclusion

The GAG-3A jet inertisation device demonstrated its capability with respect to the inertion of relatively large underground volumes in a short time span. No other available inertisation technology has flow rates comparable to the GAG-3A unit. The unit operated without problems and the underground demonstration particularly showed the non-intimidatory nature of the device.

The GAG-3A does not have universal applicability and high fresh air fan intakes would tend to swamp the inert gas output of the jet. However innovative ventilation solutions are needed to get around this problem because in the final analysis the GAG-3A (or multiples thereof) is simply a tool to produce large volumes of inert gas. This generator must be utilised by
competent ventilation engineers to maximise the benefit from the device. Furthermore this device has been used successfully for decades in coal mines in Eastern Europe where the use of the jet has been professionally supported by ventilation engineers working in mines with complicated ventilation layouts.

THE TOMLINSON BOILER TRIALS

Introduction

Cook Colliery in association with Mr John Brady of Statutory Management Services successfully tendered to ACARP to carry out an inertisation trial at Cook Colliery using the low flow inertisation generated by a Tomlinson diesel fuel boiler. The experimental phase of the project was completed on 1 June 1997.

SIMTARS was contracted to carry out part of the gas monitoring to augment the tube bundle system already installed at the Colliery.

Methodology for analysis and testing

Background:

Full details of the project are contained within the formal ACARP final report (Brady, 1997). The aim of the project was to inject the inert exhaust gas from the boiler down a borehole and displace the air within a large open goaf area rendering the goaf area inert. The progress of the inertion was carefully monitored both through fixed gas monitoring, and by inspections, by rescue teams in breathing apparatus, to investigate the effectiveness of the inertisation and identify any layering or stratification of the gases and unusual air flow patterns.

Briefly an area of Cook Colliery consisting of sections: 9 West, 10 West, 7 South, 12 South, 6 South, and 5 North, was inerted using the exhaust gas from the Tomlinson boiler (see Fig 9). Exhaust gas (typical analyses as in table 1) was pumped via compressor through a borehole at the intersection of 9 west and 5 north into this area. From the supplied mine plans the volume of goaf to be inerted was estimated to be approximately 417 300 m$^3$ - i.e. the volume of the roadways and other voids within the goaf. The volume of the goaf was also estimated from the tonnage of coal extracted which gave a higher estimate of 670 000 m$^3$. Fig 8 shows the boiler in operation.

![Fig. 8 - The Tomlinson boiler installation at Cook Colliery](image-url)
The inert gas was supplied at a rate of approximately 0.5 m³/s at 1 atmosphere pressure and less than 40 °C. The exhaust temperature was controlled to be less than 20 ° above ambient.

### Table 1 - Typical exhaust concentrations

<table>
<thead>
<tr>
<th>Component</th>
<th>Exhaust Concentration</th>
<th>Notes</th>
</tr>
</thead>
<tbody>
<tr>
<td>Oxygen</td>
<td>&lt; 2 %</td>
<td>values as low as 0.1 % were achieved but at this level CO was in excess of 1000 ppm</td>
</tr>
<tr>
<td>Carbon monoxide</td>
<td>&lt; 20 ppm</td>
<td>Start up and very low O₂ emissions (&lt; 1.0%) caused significantly higher levels of CO to be generated</td>
</tr>
<tr>
<td>Carbon dioxide</td>
<td>13.5 %</td>
<td></td>
</tr>
<tr>
<td>Nitrogen</td>
<td>balance (&gt; 84 %)</td>
<td>traces of unburnt hydrocarbon fuel were present as well as ppm levels of NOₓ and SO₂</td>
</tr>
</tbody>
</table>

**Sample collection**

The principal method for collection of samples was through a series of tubes run underground to specific locations, although additional samples were collected from gas bags filled underground by rescue teams during monitoring of the...
inertisation process. In addition check samples were taken on a regular basis from the Cook Colliery tube bundle system for correlation.

A total of eight locations were established underground.

Sample analysis

Two methods of analysis were employed. The eight tubes were plumbed directly into a manifold and then to a bank of analysers, infrared for CO, CO$_2$, and CH$_4$, paramagnetic for O$_2$. In addition bags were filled and could be analysed on the MTI portable gas chromatograph (GC). It was able to analyse for O$_2$, CO$_2$, N$_2$, CH$_4$, H$_2$ and higher hydrocarbons but not for CO. Samples collected from the Cook Colliery tube bundle were analysed this way. Cook analysed the bags on their CAMGAS GC system also as an additional comparison. Results were logged in a record book and into the SPLUS for Windows computer analysis and interpretation program.

The Cook Colliery tube bundle system monitored for CO and CH4 and was reported separately in the ACARP report.

Results And Discussion

Operation of the boiler

Inertion efficiency

The boiler initially suffered a number of breakdowns before settling into reliable production. The estimated total operation time 219.25 hours which generated 394,650 m$^3$ of exhaust gas over an elapsed period of 266.33 hours.

Fig 10 shows the cross site oxygen concentrations versus time. Essentially the oxygen concentration in the goaf when the boiler began operation was 19 %. At the conclusion of the trial the oxygen concentrations at the monitoring points varied between 9.7 and 11.5 %, averaging about 10.5 %.
The graphs of oxygen concentration versus time show that the points closest to the boiler were reduced in oxygen first. This is consistent with the gentle seepage of a cloud of gas low in oxygen mixing with the goaf gas. In addition up until mid morning on 26 May there was a brattice curtain across the top of E and F headings in 5 North and across what would be F heading of 9 West between 21 and 22 cut through to isolate 5 North from the rest of the goaf area. This would have the effect of speeding up the inertisation of this area over the rest of the goaf.

If the goaf gas was displaced by the boiler gas then the expected average oxygen concentration, using the smaller estimated goaf volume, throughout the goaf would be:

\[
O_2 (\%) = \left( 394650 \times 1.3 + (417\ 300 - 394\ 650) \times 19 \right) \over (417\ 300)
\]
where the average oxygen concentration in the exhaust gas was 1.3 % and the goaf gas was 19%. This gives an average oxygen concentration of 2.3 % or 8.6 % using the larger goaf volume. Clearly the goaf gas was not simply displaced by the exhaust gas. The graphs indicate good mixing of the gases. Thus a better model would be calculated using the recursive formula:

\[
O_{t} = \frac{417300 - O_{t-1} + \Delta t \times 1.3}{417300 - 0.5 \times \Delta t}
\]

where \( t \) is the difference between time \( t \) and \( t-1 \). As \( t \) tends to 0 this tends to 8.2 % for an inertisation time of 219.25 hours. When the larger goaf volume estimate is used the average oxygen concentration is predicted to be 11.2 %. There were various approximations made in setting up the calculation and in the estimations inherent in the calculation including flow rates and temperatures, and the lack of monitoring in some sections of the area to be inerted. Thus the calculation at best indicates that the mechanism of dispersal was consistent with a well mixed gas cloud diffusing/seeping around the area. This seepage must have been assisted by air currents within the goaf region, perhaps set up by the inertion process based on temperature and density differentials.

Crudely speaking this indicates a very high (70 to 100 %) inertion efficiency ( i.e. change in oxygen predicted with perfect mixing etc to that obtained). This is a very simple calculation and ignores the unknown mixing efficiency occurring in 12 South and 7 South. For the purposes of this calculation natural oxygen loss and seam gas infiltration have been ignored. Calculations by John Brady indicate that they will only perturb the calculations in a very minor way (approximately 10%).

Other effects

Other effects due to the inertisation process were observed by the rescue teams during their visits to the goaf area. These included:

- An absence of layering, despite the gas being 20 °C hotter than the goaf gas and more dense than air if at the same temperature. This was probably due to the mode of injection through a narrow vertical borehole which facilitated good mixing with the goaf atmosphere.

- A suppression of the methane seam gas make. This was also evident in the Laleham study - reported separately (Brady, 1997).

The expected methane make is 3900 m³ per day, based on methane content of return air from panels when ventilated (Brady, 1997). Over the period of inertion this would equate to 46,656 m³ of methane being released. Assuming full mixing and similar losses to above this would equate to an end concentration of methane of approximately 5 % by volume. In fact the methane level measured was less than 1 % , starting at 0.4 %. A rise of approximately 10 % of that expected. Thus if the projected methane emission rate is accurate inertisation actively suppresses seam gas emissions.

CONCLUSIONS

- Inertion of a large open goaf area using a low flow inertion device as a Tomlinson boiler was successfully achieved.

- The goaf tested had a small seam gas influx which was suppressed by the action of the boiler.

- The inertion gas showed no evidence of layering and mixed well with the goaf gas and distributed well throughout the goaf through diffusion rather than plug flow.
ACKNOWLEDGEMENTS

The contribution to this research by many others is acknowledged especially David Humphreys and John Brady.

REFERENCES

Developments in Self Escape and Aided Rescue Arising from the Moura No. 2 Wardens Inquiry

A SPECIAL REPORT

by

Joint Coal Industry Committee from Queensland and News South Wales

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COAL98 Conference Wollongong 18 - 20 February 1998
The MSHA seismic location system

MICROSEISMIC APPLICATION FOR AUSTRALIA

Large diameter drilling
Rescue emergency vehicle
Introduction incident management
Although the coal industry in Australia continues to expand and chase improvements in technology, equipment, training and safety, the industry still suffers from underground incidents which have tragic consequences for individuals and families. The responsibility is for all stakeholders, employers, employees and support agencies to remove causes and behaviors which create such incidents. This will make underground coal mining safer for all personnel.

The fundamental mind set to safety performance has changed dramatically over the last 15 years. During the early 1980's, the recognition was made that the industry safety performance was very poor. At the end of that decade and by the early 1990's many large companies, together with government and the unions were supporting safety training programs to reduce Lost Time Injuries - the high level measure of safety. These programs produced varying levels of success, but developed a sceptism amongst some parts of the industry where the results were managed, not the risks.

A few incidents occurred in the early to mid 1990's which showed that the rate of change and the approaches to the management of safety were not good enough. The industry started to look at Risk Management and the need to formally quantify the risks, controls and protection to improve safety performance.

The Wardens Inquiry into the accident at Moura No. 2 in 1994 highlighted the need to review rescue operations for persons underground.

A coal industry committee, already set up to review the fundamentals of coal mines rescue, was given the task to review escape and rescue options by the Queensland Chief Inspector of Coal Mines. The committee consisted of a broad cross section of major stakeholders of the NSW and Queensland coal industry. The process of review was fundamental, exhaustive and widespread in its scope. It was clearly appreciated by all the committee members that escape and rescue options for mine personnel could be substantially updated and improved. The mind set previously held, That mines rescue was the calvary charging over the will to rescue people, needed to be changed in response to recent major incidents. This change would encompass the techniques, equipment, design of mines and the role of the rescue service.

The work has taken 2.5 years to get to this stage and will require the stakeholders in this industry to implement the recommendations, complete R and D projects and participate in the information sharing that will be necessary to sustain these improvements. I would like to thank all those who have contributed including the sub committee participants. The active involvement, participation and dedication of everyone, has recognised only the need to improve safety without the barriers of state, political or industrial interference. It is encouraging to know that we can work together for the common good when the need is greatest.

Mitch Jakeman
Chairman of Task Group 4
Developments in Self Escape and Aided Rescue

COAL98 Conference Wollongong 18 - 20 February 1998
EXECUTIVE SUMMARY AND RECOMMENDATIONS

The inquiry into the 1994 accident at Moura No.2 Colliery included in its findings that mine escape and rescue options for persons in underground coal mines were in need of review. It recommended the establishment of industry working groups to report to the Chief Inspector of Coal Mines (Qld) on matters including escape strategy and life support for escape from mines.

Task Group No. 4, comprising industry stakeholders from Qld and NSW, has examined issues relevant to escape and rescue from Australian underground coal mines and formulated recommendations and guidelines in response to the set objectives and scope.

With regard to existing practice it was determined that:

- for persons underground at the time of a major incident, escape options are limited and there is no consistent strategy in place for rescue of mineworkers across the industry
- filter self rescuers have a limited application in mine emergencies
- knowledge of conditions underground after an accident is insufficient for accurate assessment of the mine environment
- rescue strategies require upgrading

The diversity of mine configurations and potential emergency scenarios led the Task Group to determine six (6) critical issues:

Self escape

There is a primary need to enhance the capabilities of underground persons to effect their own rescue - ie "self escape". This is to be achieved by the provision of facilities in mines, training of mineworkers and management and the development of generic and mine-specific escape strategies.

Escape routes, alternative routes and facilities are to be planned, developed and equipped as part of Self Escape Management Plans. An oxygen-based escape system is required with the following attributes:

- all persons to wear self contained self rescuers;
- replacement SCSR caches provided at suitably located changeover stations; and
- use of refuge chambers where appropriate.

Communication introduction

A generic strategy is required to establish the location and condition of persons underground after an incident and to maintain communication during aided rescue. The key areas requiring development are:

- guidance systems for self rescue;
- communications post incident; and
- determination of status of all personnel underground after an incident.

These will require technology transfer and some research. An industry expert committee has been formed to champion these initiatives.
Gas management

Gas management guidelines are established for effective and safe incident control. This should look at the design and location of monitoring systems, the integrity and interpretation of information, mines rescue requirements for both underground and to the surface and the training/competency of people and systems.

Aided rescue

There is also scope for intervention, assistance and rescue of underground persons by surface personnel, i.e. - "aided rescue". However, significant changes to rescue strategies are required in order to make use of modern technology to increase the chances of successful aided rescue. The determination of the status of conditions and personnel underground are key areas requiring improvement. Rescue options and post incident control strategies need to be appropriate for the incident.

Areas that need to be researched and developed are a new mines rescue vehicle, consideration of large diameter boreholes for rescue where appropriate, and the upgrade of environmental monitoring and communications equipment needed for rescue teams.

Aided Rescue Management Plans are required at mines to ensure that a coordinated and effectively-resourced rescue response can be mounted in order to maximise the chances of saving persons trapped in a mine. Plans should address:

- means to establish and monitor the status of persons underground;
- means to establish, monitor and assess conditions in the mine;
- establishing post-incident controls;
- rescue options; and
- training for aided rescue

There is a clear need for the mines rescue services to revise their operating strategies and infrastructure to accommodate the new ways of aiding rescue proposed by this report.

Incident management

Provision of new escape and rescue systems will be of limited value unless the people in danger or participating in rescue can make the appropriate decisions when confronted with an emergency situation. Planning, preparation and training for such emergencies is essential to improving their chances of survival.

Incident Management allows those managing the incident to follow decision criteria to minimise risk and conduct a successful emergency.

Training developments

Every underground mine should develop a Self Escape Management Plan as part of a Safety Management Plan to provide all persons underground with the capability to reach a place of safety, recognising the difficult environmental conditions following an incident. Training initiatives in the form of new generic training resources and training guidelines should be developed to support the use of these plans.

Specific recommendations aimed at improving capabilities in these six critical areas have been made. These recommendations promote the Task Group's proposed vision for response to incidents at mines in the future:
All persons underground at the time of an incident shall be trained, equipped and able to make an escape to the surface, or place of safety, if physically capable. Monitoring, communications systems and other new rescue technology will provide surface personnel with the capability to safety deploy aided rescue measures to rescue those unable to self-escape.

Further development, prioritisation, management and funding of initiatives in this area require the continued involvement of industry stakeholders. At the end of 1996 there were several urgent initiatives that could be quickly implemented by industry to bring about improvements in emergency escape and rescue.

The specific recommendations of the Moura No. 2 Warden's Report Implementation, Task Group 4 Report (1996) are listed below.

**Recommendation 1**

Every underground mine should develop a Self Escape Management Plan and an Aided Rescue Management Plan. The interrelationship between the Escape and Rescue Plans must be examined and incorporated into the Mine Safety Management Plan.

**Recommendation 2**

a) A generic industry training resource package for self escape and aided rescue should be developed.

b) Guidelines for mine specific training in self escape and aided rescue should be developed.

**Recommendation 3**

Every underground mine should establish an escapeway from all parts of the mine to the surface or to an alternative place of safety. A detailed risk analysis of proposed escapeways will need to be undertaken and strategies developed to control risks.

**Recommendation 4**

From each part of every underground mine, there should be at least one route, other than an escapeway, which enable self escape to the surface or an alternative place of safety. A conveyor roadway or a return roadway are not precluded for this purpose.

**Recommendation 5**

An oxygen-based self escape system should be provided for all persons underground.

**Recommendation 6**

Each person underground should be equipped with, and carry on their person at all times, a self contained self rescuer (SCSR).

**Recommendation 7**

An industry committee/forum (technical experts and operators) should be established to coordinate the advancement of capabilities to alert, to communicate with, and to assess the status of underground persons during a mine emergency.

**Recommendation 8**
Fixed tube bundles and gas chromatographs should be made available at all mines as the primary method of measuring post incident mine atmospheric conditions.

**Recommendation 9**

Research into the development of robust telemetric sensors for gas analysis and other environmental parameters, over the ranges existing after incidents, should be prioritised.

**Recommendation 10**

A mine emergency reconnaissance vehicle should be made available for all mines for use in emergencies.

**Recommendation 11**

Both pre-installed and post-incident boreholes should be considered when developing Aided Rescue Management Plans.

**Recommendation 12**

Rescue teams should be provided with state of the art environmental monitoring equipment and on-line communications.

**Recommendation 13**

Equipment should be made available to ascertain the physical status of the mine environment (including temperature, humidity, pressure etc) using boreholes, and reconnaissance vehicles.

**Recommendation 14**

A demonstration of the capabilities of microseismic monitoring technology to detect, locate and monitor roof falls, outbursts and fires should be carried out.

**Recommendation 15**

The capability to model ventilation and the mine environment following an incident should be available at mines.

**Recommendation 16**

Guidelines, common to both Qld and NSW, should be developed for integrated emergency preparedness involving mine operators and emergency services.

**Recommendation 17**

Industry should develop an effective computer-based emergency decision support system for incident management and training.

**Recommendation 18**

A steering committee should be established to encourage and oversee the development of emergency rescue vehicles.

**Recommendation 19**

Mines should consider the need for refuge chambers when developing Self Escape and Aided Rescue Management Plans.
**Recommendation 20**

Aided rescue using shafts and/or large diameter boreholes should be considered for inclusion in mines Aided Rescue Management Plans where viable.

**Recommendation 21**

- A generic industry training resource package for self escape and aided rescue should be developed for mineworkers and management/others.

**Status of Recommendations - 1 January 1998.**

<table>
<thead>
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<th>Recommendation</th>
<th>Status</th>
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<tr>
<td>Recommendation 1</td>
<td>Legislated</td>
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<tr>
<td>Recommendation 2a, 2b</td>
<td>Likely to be a six month development process during 1998.</td>
</tr>
<tr>
<td>Recommendation 3</td>
<td>Accommodated within each mine's emergency evacuation hazard management plan</td>
</tr>
<tr>
<td>Recommendation 4</td>
<td>Accommodated within each mine's emergency evacuation hazard management plan</td>
</tr>
<tr>
<td>Recommendation 5</td>
<td>Legislated in Queensland, pending in NSW (not necessarily oxygen based).</td>
</tr>
<tr>
<td>Recommendation 6</td>
<td>Legislated in Queensland</td>
</tr>
<tr>
<td>Recommendation 7</td>
<td>Have met twice - will continue to meet</td>
</tr>
<tr>
<td>Recommendation 8</td>
<td>Not Legislated - chromatographs are in place at all Qld mines</td>
</tr>
<tr>
<td>Recommendation 9</td>
<td>Research not prioritised - mentioned in gas management guidelines - requires further thought for promoting research</td>
</tr>
<tr>
<td>Recommendation 10</td>
<td>In place - NUMBAT</td>
</tr>
<tr>
<td>Recommendation 11</td>
<td>Guidelines</td>
</tr>
<tr>
<td>Recommendation 12</td>
<td>Dealt with in gas management guidelines</td>
</tr>
<tr>
<td>Recommendation 13</td>
<td>Reconnaissance vehicle - work proceeding - borehole equipment available for purchase</td>
</tr>
<tr>
<td>Recommendation 14</td>
<td>Research planned</td>
</tr>
<tr>
<td>Recommendation 15</td>
<td>Needs to be proactively energised - C. Mallet to follow up</td>
</tr>
<tr>
<td>Recommendation 16</td>
<td>Guidelines are being developed</td>
</tr>
<tr>
<td>Recommendation 17</td>
<td>Recommendations for RandD are being developed</td>
</tr>
<tr>
<td>Recommendation 18</td>
<td>Done</td>
</tr>
<tr>
<td>Recommendation 19</td>
<td>Done</td>
</tr>
<tr>
<td>Recommendation 20</td>
<td>In progress</td>
</tr>
<tr>
<td>Recommendation 21a, 21b</td>
<td>Six month project for development 1998.</td>
</tr>
</tbody>
</table>

In conclusion, many of these recommendations have been actioned throughout the industry through legislative amendments, the development of Safety Management Plans, changes to the Mines Rescue Services and further development of research projects. The aim is to continue our focus on Risk Management and not become complacent with improving results. The potential risks in mining like mother nature are unforgiving and severely penalise those that become neglectful.
INTRODUCTION

The Warden's Inquiry into the Moura No. 2 incident contained a number of findings and recommendations aimed at reducing the likelihood of future accidents. Some of the recommendations called for further investigation, analysis and the development of safe operating guidelines. Several Task Groups, representing various industry stakeholders, were subsequently assembled comprising members with relevant experience and skills to determine the required guidelines and report back to the Chief Inspector of Coal Mines (Qld).

Five such Task Groups were convened, with Task Group No. 4 allotted the task of developing guidelines in response to Recommendations 9 and 10 of the Warden's Report. These related to the development of an Escape Strategy for persons involved in an incident.

An industry panel with suitable membership had been previously formed to develop and industry plan for enhanced mines rescue following an industry forum which considered the future of the "NUMBAT" remote underground reconnaissance vehicle. This committee was allotted the additional specific tasks required of Task Group No. 4.

This report presents the recommendations of Task Group No. 4 for consideration by the Chief Inspector, and the industry in general.

OBJECTIVES

The specific objectives for Task Group No. 4 were:

1. To recommend guidelines to the Chief Inspector of Coal Mines (Qld) on self rescue escape;

2. To present a report to the Chief Inspector of Coal Mines (Qld) addressing issues identified in Recommendations 9 and 10 of the Moura No. 2 Wardens Report.

These recommendations were as follows:

Recommendation 9

"......... it is recommended that a representative industry working party, containing appropriate expertise, be convened by the Chief Inspector of Coal Mines and that group be charged with the development of guidelines for the industry covering life support for escape."

Recommendation 10

"......it is recommended that the Chief Inspector of Coal Mines set up a working party, comprising persons with appropriate knowledge and experience, to examine and report on a range of issues relating to emergency escape facilities.

The group should investigate means whereby persons in any part of a mine, who are subject to disorientation or severely impaired visibility, are able to find their way out of the mine. Consideration should also be given by the group to the potential role for motorised transport in emergency escape arrangements."

SCOPE

The scope of the study by Task Group No. 4 was established as follows:

1. to develop a strategy for enhancing emergency escape and rescue for personnel with respect to the hazards associated with fires, explosions and explosive/irrespirable atmospheres in coal mines;
2. to develop guidelines for the establishment and use at underground coal mines of an appropriate self escape system from all underground work areas which is independent of external or mechanised emergency support systems;

3. to develop recommendations leading to an effective aided rescue system for underground coal mines;

   to prepare a report addressing issues identified in Recommendations 9 and 10 of the Moura No. 2 Warden's Report, for presentation to the Chief Inspector of Coal Mines (Qld), by 28 June, 1996.

5. prepare a report to develop guidelines for self escape, gas management, incident management and establish training competencies and establish research projects for communication and aided rescue by Jan 1, 1998.

The report is to address a broad range of issues relating to emergency escape facilities at a coal mine. Subjects to be addressed should include:

- assistance to visually impaired or disoriented person;
- potential for powered emergency transport
- use of large diameter drill holes and possible emergency recovery routes
- rescue chambers or self contained life support refuges;
- use of designated escapeways as part of the mine design;
- communications options to determine the status of person underground
- identification of any limitations on the dimensions and operating range of emergency vehicles
- identification of the maximum distance an emergency vehicle may have to travel in any Queensland underground coal mine (currently operating) within the next 15 years.

The committee's focus spans from immediately after the initiation of an emergency incident up until the time that all personnel have been recovered from the mine, or when the emergency control team has determined that the emergency is over. The retrieval of any bodies, obtaining of evidence or recovering the mine are not considered emergency activities and should be conducted as a pre-planned operation using a risk management approach.

DEVELOPMENT OF STRATEGY

The Task Group has considered the post incident response issues in the context of the structure illustrated in the flow chart below (Fig 5.1). This flowchart tracks the development of an emergency from the incident until the mine returns to operational status or closure. It also recognises that in order to effectively prepare for incident response, management plans, rescue facilities and training need to be put in place. The critical elements of this process studied by the Task Group are the Self Escape and Aided Rescue components as well as aspects of pre-incident preparation.

Table 4. illustrates the core issues relevant to the development of an effective escape and rescue strategy.
The issues have been broadly categorised into those covering escape strategy, personnel status, environmental conditions, control strategy and rescue options. Several of the issues overlap these categories as addressed in the detailed sections of the report. Some issues have been extensively dealt with by other task groups (e.g. seals, inertisation, re-entry) and are listed here only for completeness.

The Task Group adopted an integrated systems approach to escape and rescue which resulted in a number of recommendations which will require prioritised implementation. Several recommendations require research and development, or embody training initiatives, some require a legislative response, and most require further industry collaboration. It is recognised that until all of the elements identified are available, the recommended system may not achieve its full potential.

It was also recognised that individual mines may develop specific plans that are subsets of a total rescue system. It is therefore up to individual operations to identify, develop and implement rescue systems appropriate for their needs in conjunction with relevant stakeholders. It is vital, however that the industry adopts a uniform approach to the upgrading of escape and rescue strategies in order to maximise the potential for incident survival and response.

Strategy development was considered for current as well as future coal mine scenarios. In looking ahead, it was considered impractical to forecast the nature of coal mine layouts, size and operating environments beyond 5 to 10 years, during which time similar characteristics to those in existence today and existing mines will simply increase in area and
distance from surface entries. Some may add man-winding shafts. It is expected that escape and rescue strategies developed today will be continuously reviewed and amended to maintain relevance.

**SELF ESCAPE - EMERGENCY ESCAPE SYSTEMS**

**Introduction**

Issues relevant to self-escape from Australian underground coalmines have been examined and recommendations and guidelines formulated in response to the terms of reference and scope for Task Group 4. The issues relate to the implementation of Task Group 4 recommendations 1, 2a, 3, 4, 5, 6, 19, 21a, 21b (see pp 10-12).

Self Escape - Emergency Escape Systems, covers the major points arising from the work of both the Queensland and New South Wales working groups, and the results of a study of overseas escape strategies that was undertaken by the NSW working group.

W. Barraclough, BHP Coal  
R. Bancroft, Dept Minerals and Energy Qld  
P. Eade, BHP Coal  
G. Dwyer, United Mineworker's Union  
G. Fawcett, Dept Mineral Resources  
G. MacDonald, Dept of Mineral Resources  
P. MacKenzie-Wood, Mines Rescue Service NSW  
F. O'Connor, Appin Colliery

**Escape strategy**

Evaluation of the various factors involved identified a number of major elements that need to be addressed in the development of an emergency escape system that enables persons to escape safely.

Some of these elements are:

- Early Warning;
- Self Rescue Apparatus;
- Communications;
- Guidance Systems / Lifelines;
- Escapeways / Transport;
- Refuge Chambers / Changeover Stations;
- Training of personnel; and
- Safety Management Plan for Evacuation.

The escape of persons underground will be enhanced by the use of a planned strategy that has been developed by consideration of these elements and recognition of the potentially difficult circumstances a person could encounter following an incident. Importantly the strategy will include the realisation that the mobilisation of rescue personnel could take time. The initial reactions of persons underground to an incident situation are a significant determinant on their survival. Planning, preparation and training for such emergencies are essential components required to improving their likelihood of survival.
Early warning

The role of an early warning system is to sense the first signs of fire or explosion and communicate an alarm so that evacuation of the mine or part can take place. Control measures taken at the earliest possible time would allow egress through reasonably smoke free escapeways and maximise effective escape.

Carbon Monoxide and smoke sensing systems offer considerable potential for early and more reliable fire detection than do other available systems.

A control system must be established to receive and analyse data on the underground environment. The system must include decision making protocols and enable control to be maintained and action to be coordinated during an emergency.

Consideration should be given to the incorporation of a communication system throughout the mine that can be used to immediately notify underground employees in all areas of the mine of the need to evacuate. The system should have the ability to provide employees with incident details and directions. Principal systems include telephone, traditional two-way radio, ground induction and leaky feeders.

Western Australian, Canadian and Mount Isa Mines metalliferous mines have introduced systems to release stench gas to the ventilation system to initiate emergency procedures.

Computer generated emergency alert systems are available where recorded messages can be transmitted to localities on an “at risk” basis. The Revmaux (BBL) mine in France utilises such a system with a maximum of 10 localities alerted at one time. An alert immediately triggers the escape procedure. A similar type of system has been used in a Queensland colliery.

The Personal Emergency Device referred to as “PED through the earth system” is capable of sending radio messages from the surface to wearers of receiving units on a mine-wide basis after an incident. The PED’s utility is limited by the inability to return signals from the wearer to the surface. Medium frequency partially inductive systems (eg Rimtech, Taiheyo) provide increased potential for survival after an incident because of the robust nature of the wave carriers used (pipes, cables etc.). Prototype units for locating trapped miners have been developed overseas but their application is limited to short range, direct line of sight and restrictive circumstances.

Self rescuers

Filter type self rescuers were introduced into the coal mining industry in the 1960’s in response to many fatalities that had arisen due to conveyor belt fires (e.g. Creswell Colliery, UK, 1950 - 80 fatalities). They are only effective where sufficient oxygen is present in the atmosphere.

The introduction of fire resistant anti-static conveyor belts, fire resistant oils, reduced use of timber supports and improved environmental monitoring technology has reduced the risk of mine fires and hence the principal reason for the use of filter type self-rescuers.

An explosion occurring in the vicinity of a working face is now the principal hazard that may require the use of a self-rescuer.

In reviewing previous explosion incidents, it was found that due to the reduced oxygen content of parts of the mine atmosphere following explosions, the use of a filter type rescuer would not have enabled persons to escape.

For this reason, it is considered essential that all persons underground be equipped with a self-contained self-rescuer (SCSR), ie a self-rescuer that provides the wearer with respirable air.

There are many brands and types of SCSR’s currently available. These are mainly manufactured in either Europe, the USA or South Africa, each country having different testing and approval criteria. The only international standard currently available for the testing of chemical type (K02) oxygen self-rescuers is EN 401 (B.S., 1993). The testing of compressed oxygen self-rescuers in Australia is covered by AS/NZ 1716 (DMR, 1996).
Because of the differing test criteria used, and the confusion that this can create when evaluating different brands, EN 401 has been recommended as the standard for the testing of chemical oxygen SCSR's. EN 401 is being adopted until an Australian standard is developed.

Immediately following an underground mine explosion, visibility can be significantly reduced causing irritation to eyes in smoke laden atmospheres. This impairs the self-escape of persons who can become disoriented. Combined with the lack of communication, serious limitations are placed on the ability to effect escape.

South African research and experience with chemical oxygen SCSR's has shown that poor visibility and disorientation can reduce the distance traveled to 60% of that expected under normal conditions.

Many cases have been cited where persons have not been able to find their self-contained self-rescuer immediately adjacent to them (DMR, 1996).

Due to this disorientation and lack of visibility, it is essential that all people underground carry an SCSR with them at all times.

Another factor that can play a major part in the escape of persons using self-rescuers is body mass. This subject is dealt with comprehensively by Paul Mackenzie Wood in his paper "Deployment of self-contained self-rescuers in coal mines".

There is a requirement in all Australian underground coalmines for the use of approved self rescuers. The minimum requirement in NSW is for filter type self-rescuers and from January 1 1998 self-contained oxygen self-rescuers have been required in Queensland.

Communications

There is a need for a communications system that would survive an incident and provide ongoing two-way communications between escaping or trapped miners and rescue personnel on the surface. The system should be compatible with the type of self-rescue breathing apparatus to be used and the likely escape or refuge options available to survivors. As power to the mine is likely to be interrupted during an incident, self-contained battery powered backup should be integral to the system. Whilst voice is the highest priority for transfer, systems which can also transmit data and video signals should be encouraged to assist the rescue process.

The minimum coverage requirement is for a communication system to be established along escape routes.

The location and tracking of all persons (and most vehicles) in underground mines should also be considered in any escape system. Effective two-way voice communication will contribute to this requirement but more efficient electronic systems should be pursued.

Current communications systems for underground mines are limited for emergency conditions but there are commercial leaky feeder based systems which have good potential provided that transmission networks can be stiffened to survive incidents or equipoped with satisfactory redundancy. Low frequency "through the earth" technology is being researched for underground-to-surface capability. Once robust networks can be demonstrated, value-adding technology such as personnel and vehicle tracking and personnel status monitoring can be deployed. Management plans must embrace the support of such communication systems and link into emergency protocols and controls.

Escapeways

Rescue response following an incident involves a period of time that, in most circumstances, requires people underground to attempt an organised escape, rather than await rescue. In Australian collieries, the distance from the working face to the surface can be considerable, and in many cases the seam grade can be quite steep. These escape route difficulties, allied with the expected problems of disorientation and poor visibility, give rise to a requirement for a roadway to be established in each mine that meets the criteria of good trafficability.
This roadway should, as far as practicable, be capable of maintaining a respirable atmosphere that is free from fumes and airborne dust, after an explosion or fire. To achieve this, the escapeway should be an intake airway and protected from damage by being segregated from other roadways with stoppings capable of withstanding low intensity explosions.

Vehicular escape would, in most circumstances, afford the best chance of persons making a rapid escape from the mine, and escapeways should be designed to maximise the likelihood of facilitating vehicular escape, without precluding or endangering passage by foot.

**Guidance systems**

To assist in gaining access to escapeways, and in guiding persons along escapeways in conditions of low visibility, clear guidance systems that will survive an incident are required. Knotted ropes with directional cones fitted (lifelines) have been developed for this purpose. More recently, battery-powered guidance systems, such as the “MOSES” system in South Africa and “LEADLIGHT” in Australia incorporating directionally discriminating audible pitches and flashing LED’s have been developed to provide clearer guidance. The Australian system is also developing a tracking tag system which can be integrated or stand alone to determine where personnel are in the escapeways or mine workings.

Use of the term “second means of egress” is commonly applied to return airways, with little thought being given to which is the most desirable escape route. In emergency exercises involving different scenarios, employees invariably attempted to escape via the returns, even when this may have been the most inappropriate route. The concept of “second means of egress” as the primary escape route should be replaced by the concept of an “escapeway”.

Mine management should carefully consider which airway would make the most suitable escapeway. Because of the need to maintain a respirable atmosphere, the risk of fire in this roadway should be reduced to a minimum. This could be achieved by restricting the use of equipment in this roadway to those items that are either fitted with fire suppression devices, or which incorporate a fail safe system to prevent the outbreak of fire.

**Change-over stations**

Dependant upon the distance of the working areas from the surface and the duration of any self contained self-rescuers (SCSRs) to be carried, the provision of underground caches of SCSR's must be considered to facilitate the escape of persons to the surface. The number and separation distance between caches should be based on the assumption that the mine atmosphere is irrespirable all of the way to the surface, and that visibility throughout the mine will be very poor.

Caches installed throughout a mine should be constructed so that they are protected from the effects of low intensity explosions. Persons exchanging SCSR’s should be able to do so in a safe manner. This could be accomplished by being able to exchange SCSR’s in irrespirable atmospheres or by the provision of changeover stations equipped with respirable air. Consideration should also be given to equipping changeover stations with communication facilities, capable of surviving an incident, to facilitate escape co-ordination.

In addition to designated caches located at strategic locations in the mine, consideration should be given to the provision of either a cache of SCSR’s or some other system of respirable air, on board personnel vehicles. There are compressed air systems now available, comprising a storage cylinders and a number of face masks connected to a common supply regulator, that could meet this need.

**Refuge chambers**

Refuge chambers have an accepted place in rescue strategies in South African coalmines where workers are instructed to make their way to the section refuge chamber. This is mainly due to the large areal extent of the mine workings, the generally shallow depth of workings (enabling borehole recovery in the event of a disaster) and the differing cultural backgrounds and experience of the mine workers.

The majority of opinions sought on the use of refuge chambers in Australian coalmines indicates that Australian coal miners, in the absence of incident information, would attempt to reach the mine surface rather than stay underground in a Refuge Chamber.
In the first instance, escape systems should be provided to enable persons to escape to the surface of the mine or alternative place of safety. Operators should, however, examine their own circumstances and possible scenarios to ascertain whether or not there is a place for refuge chambers in their Self Escape Management Plan.

Current thinking indicates that it is very unlikely that rescue teams will be sent into a mine with explosive or toxic concentrations of gas, and that miners will generally need to effect their own rescue.

For this reason it is believed that regardless of whether a Refuge Chamber or a Change Over Station is used, the system should be mainly designed so that miners have a safe place to assemble.

The Refuge Chamber or Change Over Station should preferably be supplied with a respirable atmosphere and means of communication to the surface so that people can plan their escape and change from one self rescuer to another in safety.

While the system may be best designed to provide assistance to a safe and timely escape, it needs to be recognised that there may be injured persons that are unable to escape from the mine, but may be able to reach a place of safety if one is provided.

Training

Provision of oxygen self-rescuers, early warning systems and escapeways will be of limited value unless the people attempting escape can make the appropriate decisions when confronted with an emergency situation. It is essential that all mineworkers be given adequate and regular training in all aspects of the mine escape system.

Training exercises should entail more than just travelling through the second means of egress or escapeway.

A feature of both USA and South African mineworker training is participation in regular evacuation exercises, often under simulated conditions of disorientation or low visibility.

Evacuation management plan

Consideration of all the various aspects of the mine when examined in the light of the previously enumerated factors should be incorporated into a mine evacuation or Self-Escape Management Plan.

The plan should be developed using the criteria established in guidelines for Queensland Safety Management Plans or the New South Wales Risk Management handbook for the Mining Industry MDG 1010.

This would provide all persons underground with the capability to reach a place of safety, recognising the difficult environmental conditions likely to be encountered following an underground incident.

Bibliography

Recommended Guidelines for Oxygen Self-Rescuers-Volume I, Underground Coal Mining, June 1981

Recommended Guidelines for Oxygen Self-Rescuers-Volume II, Appendices, June 1981


Person Wearable SCSR Task Force, Final Report by Jeffery H. Kravitz, Chief, Mine Emergency Operations, Mine Safety and Health Administration and John G. Kovac, Supervisory Mechanical Engineer, USBM

Federal Register; May 15, 1992; Part II; 30 CFR Parts 70 and 75 Safety Standards for Underground Coal Mine Ventilation; Rule


Physiological Responses of Miners to Emergency; Volume I, Self Contained Breathing Apparatus Stressors

Physiological Responses of Miners to Emergency, Volume II, Appendices

An Examination of Major Mine Disasters in the United States and a Historical Summary of MSHA's Mine Emergency Operations Program by Jeffery H. Kravitz


Chamber of Mine of South Africa - Position paper on performance and life potential of Self-Contained Self-Rescuers

European Structured EN401 - 1993 Chemical Oxygen (KO₂) escape apparatus

The introduction of self contained Self-Rescuers into the South African Mining Industry by HJM Rose - July 1988

Escape and Rescue from Mines - Approved Code of Practice and Guidance - HSE UK 1995


Moura Task group 4 Report Qld DME/QMC 1997

Various discussion papers by NSW Underground Emergency Systems working group

Report on Overseas study into escape systems R Bancroft Qld DME 1997

Report on Overseas study into escape systems P Eade BHP 1997
COMMUNICATIONS - RESEARCH and DEVELOPMENT PROJECTS

The task group concluded that communications following underground incidents could be enhanced through several initiatives some of which required demonstration and some which required further research and development.

The communication sub-committee was:
B. Robertson (Chair), Shell Coal Pty Ltd
J. Ruble, Moranbah North
T. Hancock, Moranbah North
G. Eaton, BHP Colleries
T. Willmott, Dartbrook Mine
D. Decker, CSIRO
J. Jacka, CSIRO
R. Wischusen, AMIRA
D. Pomfret, Power Coal

The recommendations and actions were:

- Develop alert/alarm systems - options to be considered and consolidated
- Investigate "reverse PED" concept - develop research proposal
- Develop robust telephone nodes - cooperate with suppliers to demonstrate
- Test escapeway medium frequency inductive systems - trials
- Reinforce mine-wide radio networks - trials
- Demonstrate tracking - trials
- Facilitate communication between employees - evaluate options with SCSR manufacturers
- Develop distress beacons - review prototypes, develop research projects if feasible
- Establish protocols - for industry panel
- Develop vital signs monitoring - source transducers, trial

An industry committee/forum was established to examine each of these recommendations and propose action plans for implementation. The committee determined that:

- Traditional alert/alarm systems were not ideal for coal mines but that a satisfactory full time radio-based communications system would satisfy this need.

- A "through-the-earth" radio system which allowed messages to be sent from underground to surface was highly desirable and potentially feasible but required significant development. An ACARP grant was made to CSIRO Division of Radiophysics to develop such technology.

- Reinforcement of telephone and radio systems through installation of redundant links to surface via boreholes was seen as a viable strategy and trials were to be encouraged. CSIRO is developing a cellular system of low cost radio repeaters which provides improved redundancy and reliability by virtue of overlapping cells.

- MF inductive radio systems suffer from voice quality and range but may be usefully deployed in emergency escapeways. Suitable manufacturers should be encouraged to demonstrate capabilities.
• Tracking of personnel and equipment was commercially available but depended on network reliability. Demonstrations would be forthcoming.

It was necessary to consider how escaping persons could effectively communicate with each other when wearing SCSRs. This was an issue for consideration by manufacturers and mine managers, but should await establishment of satisfactory escape management plans and equipment.

Distress beacons were desirable and some prototypes of limited range are available overseas. The application of radio-based tracking transducers should encompass this requirement. Protocols were not required.

Vital signs transducers exist in the medical/surveillance industries and could readily be deployed underground, given robust networks.

It was observed that the underground metalliferous industry was more advanced in communications than was coal and that much could be learned from this experience.

GAS MANAGEMENT GUIDELINES

Introduction

The Underground Mines Rescue Strategy Development group selected a gas management sub-committee to look at the gas management information required in an emergency and how it is to be provided. This request was related to the implementation of Task Group 4 recommendations 8,9 and 12 (see pp 10-12).

8. Fixed tube bundles and gas chromatographs should be made available at all mines as the primary method of measuring post incident mine atmospheric conditions. (Plans, RandD and legislation).

9. Research into the development of robust telemetric sensors for gas analysis and other environmental parameters, over the ranges existing after incidents, should be prioritised (RandD).

12. Rescue teams should be provided with state of the art environmental monitoring equipment and on-line communications (RandD).

The gas management sub-committee had the following membership:

P Mackenzie-Wood (Chair), Mines Rescue Service, NSW
D Kerr, Queensland Mines Rescue Brigade
R Moreby, Dartbrook Coal
D Cliff, Simtars
W Allison, CFMEU, Queensland Branch
W Price, Crinum Mine
G Fawcett, Dept Mineral Resources, NSW

Terms of reference

1. To prepare guidance material that identifies available systems that will provide adequate information on gases for the effective and safe control of an incident.

2. To identify research and development requirements to enhance the adequacy of gas management systems in an emergency.
Availability of critical information
- Design criteria for monitoring systems
- Location of monitoring points
- Integrity of information
- Data management (software)
- Interpretation

Mines Rescue requirement
- Underground personal gas monitoring
- Communication surface to FAB team
- Environmental team to FAB

Training/competency
- Control room persons
- Monitoring system
- Refresher
- Sampling
- Boreholes

Documentation
- Records
- Management system

Matrix 1
- Comparison of monitoring systems

Matrix 2
- Location of monitoring points
OBJECTIVE: To provide adequate information on gases during normal operations, during an incident and after an incident to enable effective and safe incident control.

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<th>Issues</th>
<th>Notes</th>
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<td>Availability of critical</td>
<td>Design criteria for monitoring systems</td>
<td>• Comparison of systems - See Matrix 1 (page 8)</td>
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<tr>
<td>information</td>
<td></td>
<td>• Redundancy of monitoring</td>
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<tr>
<td></td>
<td></td>
<td>- Tubes and sensors in same location</td>
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<td></td>
<td></td>
<td>• Continuity of information</td>
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<td></td>
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<td>- Power backup (uninterrupted power supply - battery or generator)</td>
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<td>- Boreholes as additional sampling points</td>
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<td>- Utilisation of exploratory boreholes</td>
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<td>• Robust (surface to underground) monitoring points to enhance the integrity of post incident information</td>
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<td>- Fire resistant (self extinguishing)</td>
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<td>- Explosion resistant (protected, secured)</td>
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<tr>
<td></td>
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<td>- Boreholes may offer protection</td>
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<td>• Mobile monitoring systems (NUMBAT) to gain information from additional locations</td>
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<td>• Concentration ranges</td>
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<td>- Avoid range switching at critical concentrations</td>
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<td>eg 5% CH₄ and 15% CH₄</td>
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<td>- High range capability - dual systems such as tube bundle and gas chromatograph</td>
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<td>eg 100% CH₄</td>
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<td>- High range methods and standards to be maintained</td>
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<td>- CO, CH₄, O₂, CO₂ and velocity - essential</td>
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<td>- Transducer for velocity, temperature and differential pressure at appropriate locations</td>
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<td></td>
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<td>- Infrastructure to support calibration and maintenance to include documented procedures and Australian Standard, trained staff, range of calibration standards for all gases to cover all ranges</td>
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<td>Third party calibration audit, participation in post incident correlation and cross-sensitivity exercises</td>
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<td>Consider pressure testing of tubes for leaks</td>
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<td></td>
<td>Consider cross-sensitivity of IR CO analyser to CO₂, N₂O, H₂O, the CO electrochemical cell to H₂, H₂S, higher hydrocarbons and the catalytic sensor to CO, H₂, low O₂ and poisons</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Note: N₂O has been found in the goaf of a number of collieries and is thought to originate from adjacent open cut operations and is a by-product from the use of explosives</td>
</tr>
</tbody>
</table>

Note: N₂O has been found in the goaf of a number of collieries and is thought to originate from adjacent open cut operations and is a by-product from the use of explosives.
<table>
<thead>
<tr>
<th>Sub Element</th>
<th>Issues</th>
<th>Notes</th>
</tr>
</thead>
<tbody>
<tr>
<td>Location of monitoring points</td>
<td>- All panel returns and bleeders</td>
<td>Note: Lag time should be minimised at all locations</td>
</tr>
<tr>
<td></td>
<td>- Intakes</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Belt roads (drive heads)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Active goaf edges</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Panel faces</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Seal areas (maintain adequate records)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Upcast shaft</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Surface monitoring room</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- See Matrix 2 for location requirements</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Research potential</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Investigate damage to tubes and sensors (in both borehole and roadway locations) in methane explosions</td>
<td>Note: Klopperboss explosion gallery in South Africa has this capability</td>
</tr>
<tr>
<td>Integrity of information</td>
<td>- Concentration (%, ppm), velocity, barometric and fan differential</td>
<td></td>
</tr>
<tr>
<td></td>
<td>pressure</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Recorded, trended graphically</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Sampling information</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Frequency must ensure significant information is available for trend</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Air free</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Computed, trended graphically</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Ratios and indicators of spontaneous combustion</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Computed, trended graphically</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Explosibility</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Concentration of CH₄, H₂, CO, O₂ and CO₂ need to be accurately</td>
<td></td>
</tr>
<tr>
<td></td>
<td>determined</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Computed, trended graphically</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Alarms</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Discriminated, altered for incident levels, passworded</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Gas outlet for mobile laboratory</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Archive capability</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Alarm setting, calibration frequency records for audit</td>
<td></td>
</tr>
<tr>
<td>Data management (software)</td>
<td>- Ease of use (user friendly)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Tailoring to suit site needs</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Standard output interface for access by Incident Management Team</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Gas data to stand alone (in addition to instrument readout and not to</td>
<td></td>
</tr>
<tr>
<td></td>
<td>be shared with other underground information)</td>
<td></td>
</tr>
<tr>
<td></td>
<td>- Network capability / exportable / transmittable to underground</td>
<td></td>
</tr>
<tr>
<td></td>
<td>locations</td>
<td></td>
</tr>
<tr>
<td>Sub Element</td>
<td>Issues</td>
<td>Notes</td>
</tr>
<tr>
<td>----------------------</td>
<td>------------------------------------------------------------------------</td>
<td>-----------------------------------------------------------------------</td>
</tr>
<tr>
<td>Mines Rescue</td>
<td>Interpretation</td>
<td>Availability of sufficient numbers of competent people</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Experience</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Qualification</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Training</td>
</tr>
<tr>
<td></td>
<td>Underground personal gas monitoring</td>
<td>Gases</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- O₂, CO and total combustibles (CH₄ + H₂ + CO) on a 0-100% LEL output, CO₂, H₂S</td>
</tr>
<tr>
<td></td>
<td></td>
<td>2 multigas instruments per team</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Defeated or manipulated alarms and identifiable from normal mining instruments</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Data logging and down loading capability</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Protection from radio frequency interference</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Other information</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Temperature</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Relative humidity</td>
</tr>
<tr>
<td></td>
<td>Communication surface - FAB - team</td>
<td>Rely on in-mine communications (telephones, DAC, PED)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Personal inductive radios</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Multistrand aerial (carried and doubles as lifeline)</td>
</tr>
<tr>
<td></td>
<td>Research requirement</td>
<td>To make existing communication more robust, flexible, safe to use</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Fibre optic cable for digital radio transmission, video (helmet cam)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Reverse Ped to include transmission of gas and other environmental data</td>
</tr>
<tr>
<td></td>
<td>Environmental team - FAB</td>
<td>Psychrometer (temperature and humidity)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Anemometer (M²/sec)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Velometer (M/sec)</td>
</tr>
<tr>
<td></td>
<td>Research requirement</td>
<td>Intrinsicly safe, direct read out of temperature, humidity, velocity and quantity</td>
</tr>
<tr>
<td></td>
<td></td>
<td>I.R. thermography</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Sonar</td>
</tr>
<tr>
<td></td>
<td>Training / Competency persons</td>
<td>Underground experience at least equivalent to that required to obtain a current deputy certificate</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Advanced first-aid / life support</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Normal background levels, action trigger levels and action plans</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Competency in modules from an accredited course dealing with:</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Emergency escape plans, location of caches</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Emergency response plans</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Principal management hazard plans</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Familiarisation with underground operations</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Mine gases / detection / monitoring / sampling</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Principles of mine ventilation</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Spontaneous combustion</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Mine fires / explosions</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Interpretation of associated gas mixtures</td>
</tr>
<tr>
<td></td>
<td></td>
<td>- Communication skills</td>
</tr>
<tr>
<td>Sub Element</td>
<td>Issues</td>
<td>Notes</td>
</tr>
<tr>
<td>-------------------</td>
<td>------------------------------------------------------------------------</td>
<td>-------------------------------------------------------------------------------------------------</td>
</tr>
<tr>
<td>Monitoring system</td>
<td>• Sufficient competent people on site to operate, calibrate and maintain system to ensure continuous relevant data is available when required and in the format required</td>
<td></td>
</tr>
<tr>
<td>Refresher</td>
<td>• Industry standards</td>
<td>• Challenge testing commensurate with frequency of exposure to procedures and technology</td>
</tr>
<tr>
<td></td>
<td>• Simulations to review adequacy of action plans</td>
<td></td>
</tr>
<tr>
<td>Sampling</td>
<td><strong>Boreholes</strong></td>
<td>• Use of pre-installed boreholes</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Emissions from rider seams</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Effect of underground and surface pressure changes on inward / outwards breathing</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Effect of stratification of gases</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Difficulties with precise location of sample line end</td>
</tr>
<tr>
<td></td>
<td></td>
<td><strong>Research requirements</strong></td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Better reliability of drilling location</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Method to precisely determine the location of the end of the sample line</td>
</tr>
<tr>
<td>Documentation</td>
<td><strong>Records</strong></td>
<td>• Archiving</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Average for discrete samples (eg skipping to every 5th point?)</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Training / competencies</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Calibration / maintenance</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Date of purchase / installation</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Duty cards / resource list (people, materials, support)</td>
</tr>
<tr>
<td>Management system</td>
<td><strong>system</strong></td>
<td>• Procedures</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Roles and responsibilities</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Audits</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Review / upgrade to best practice</td>
</tr>
</tbody>
</table>
Matrix 1 – Comparison of monitoring system

<table>
<thead>
<tr>
<th>System</th>
<th>Tube Bundle System</th>
<th>Sensor System</th>
<th>Gas Chromatography</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Advantages</strong></td>
<td>• No explosion proof instruments required</td>
<td>• Results in real time</td>
<td>• Complete analysis</td>
</tr>
<tr>
<td></td>
<td>• Easier maintenance</td>
<td>• Long distances are possible with some types</td>
<td>• No cross sensitivity</td>
</tr>
<tr>
<td></td>
<td>• No underground power requirements</td>
<td>• Sensor failure is immediately recognised</td>
<td>• Capable of measuring hydrogen</td>
</tr>
<tr>
<td></td>
<td>• Wide range of gases</td>
<td></td>
<td>• Capable of measuring ethylene and higher hydrocarbons</td>
</tr>
<tr>
<td></td>
<td>• Instruments can be calibrated on the surface</td>
<td></td>
<td>• Wide measuring range</td>
</tr>
<tr>
<td></td>
<td>• Readily attach additional instruments, mobile laboratory or gas chromatograph</td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Disadvantages</strong></td>
<td>• Results not in real time</td>
<td>• High maintenance</td>
<td>• Relatively slow speed of analysis with some systems</td>
</tr>
<tr>
<td></td>
<td>• Leaks are not immediately apparent</td>
<td>• Limited carbon monoxide sensor range (0.2%)</td>
<td>• High maintenance</td>
</tr>
<tr>
<td></td>
<td>• Condensation in tubes</td>
<td>• Limited sensors (limited for carbon dioxide / none for hydrogen)</td>
<td>• Complex controls</td>
</tr>
<tr>
<td></td>
<td>• Faults not immediately apparent</td>
<td>• Poisoning of methane sensor</td>
<td>• Requires expert attention</td>
</tr>
<tr>
<td></td>
<td>• Tubes may be damaged in an explosion</td>
<td>• In situ calibration</td>
<td>• Regular calibration</td>
</tr>
<tr>
<td></td>
<td>• Cross sensitivity of CO IR cell</td>
<td>• Cross sensitivity of CO sensor</td>
<td>• High CH₄ concentrations may interfere with low level CO measurements</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Loss of power (eg &gt;1.25% methane)</td>
<td>• Requires discrete samples</td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Limited sensor life (1-2 years)</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Unsuitable in low oxygen atmospheres (behind seals)</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>• Impedance loss in cables can occur with some types</td>
<td></td>
</tr>
</tbody>
</table>

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# Matrix 2 – Location of monitoring points

<table>
<thead>
<tr>
<th></th>
<th>M/Sec</th>
<th>CH$_4$%</th>
<th>CO$_2$%</th>
<th>CO(ppm)</th>
<th>O$_2$%</th>
<th>H$_2$ppm and % C,H$_4$ ppm</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Panel faces</strong></td>
<td>Sensor only</td>
<td>Sensor only</td>
<td>Sensor only</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Active goaf edges</strong></td>
<td>Sensor only</td>
<td>Tube only</td>
<td>Tube only</td>
<td>Tube only</td>
<td>Tube only</td>
<td>Tube/discrete sample</td>
</tr>
<tr>
<td><strong>Sealed areas</strong></td>
<td>Tube only</td>
<td>Tube only</td>
<td>Tube only</td>
<td>Tube only</td>
<td>Tube only</td>
<td>Tube/discrete sample</td>
</tr>
<tr>
<td><strong>Belt roads Drive Head</strong></td>
<td>Sensor only</td>
<td>Sensor (Drive Head)</td>
<td>Sensor</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Intakes</strong></td>
<td>Sensor only</td>
<td>✔️</td>
<td>✔️</td>
<td>✔️</td>
<td>✔️</td>
<td>Tube/discrete sample</td>
</tr>
<tr>
<td><strong>Panel returns</strong></td>
<td>Sensor only</td>
<td>✔️</td>
<td>✔️</td>
<td>✔️</td>
<td>✔️</td>
<td>Tube/discrete sample</td>
</tr>
<tr>
<td><strong>U/C Shaft</strong></td>
<td>Sensor only</td>
<td>✔️</td>
<td>✔️</td>
<td>✔️</td>
<td>✔️</td>
<td></td>
</tr>
<tr>
<td><strong>Surface monitoring Room</strong></td>
<td>Tube</td>
<td>Tube</td>
<td>Tube</td>
<td>Tube</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

✔️ - Tube and sensor required
AIDED RESCUE

Introduction

The issues relevant to Aided Rescue from underground mines have been examined to look at the equipment, systems, training and research projects that need to be considered. Recommendations and guidelines have been formulated with reference to Task Group 4 recommendations 1, 10, 11, 12, 13, 18, 19, 20. The Aided Rescue sub-committee included:

A. Sellars (Chair), Queensland Mines Rescue
M. Downs, BHP Coal
C. Mallett, CSIRO
J. Tapp, CFMEU NSW
B. Lyne, DME Qld

Terms of reference

- To prepare guidance material that identifies available systems and their limitations so that adequate provision may be considered in the design of mines or the development of a mines Hazard Management Plan.
- To identify research and development requirements on refuge chambers, microseismic detection systems, large diameter drilling and rescue emergency vehicles.

Refuge chambers used in underground mining

General background

The use of Refuge Chambers as a critical part of an underground rescue strategy is applied in both coal and metalliferous mines in the Republic of South Africa (RSA). There are also limited numbers of refuge chambers in use in coal mines in Australia and the United States of America, these being apparently less integrated into an overall escape strategy.

It is worthwhile noting that the South Africans have "operational" experience in the use of such chambers.

Discussions with Australian industry personnel who have recently undertaken a study tour of overseas mines and mining safety and research establishments, indicates that the concept of the rescue chamber is embodied in both statutory and operation aspects of South African mines.

It must be stressed that the Rescue Chamber concept is a part of an overall strategy involving self contained breathing apparatus, guidance systems, site infrastructure, chamber provisions and recovery methods. The Refuge Chamber is not considered a "stand alone" item.

Well developed management schemes are in place in the RSA, covering chamber inspections, placement of chambers relative to working faces, chamber facilities/provisions, associated infrastructure and training/practice schedules.

There are two basic forms of rescue chamber, these being either a portable chamber, or a static chamber, built in and limited to, a particular site in the mine. In Australia, a portable chamber design is marketed by MSA, whilst information from the RSA indicates the development of a relatively sophisticated portable design is reasonably advanced.

Static chambers are of myriad designs, each particular design being tailored to specific sites or applications. Experience in South Africa indicates that chambers should be built in "blind ends", ie not in redundant cut-throughs or between ventilation splits.

It is plausible to envisage a variety of construction methods being acceptable in the case of a static chamber, encompassing steel bulkheads, grout walls, block walls, etc.
Key points of overall strategy

The use of refuge chambers is predicted on the basis that:

- emergency egress to the surface is either prevented or is relatively more dangerous.
- entry to the refuge chamber can be achieved in emergency conditions.
- personnel can be sustained in the chamber pending rescue.
- personnel recovery from the chamber can be effected in a timely manner, either by stabilising external mine conditions to suit self egress, or by specialist teams equipped with self-contained breathing apparatus.

Each of the strategic factors above prompts a set of conditions or requirements that must be in place for the refuge chamber principle to be effective. The concept of the refuge chamber is unlikely to work in isolation.

System requirements

Significant training will be needed to gain behavioural acceptance of the refuge chamber as a "muster point" in case of an emergency, rather than "bolting for the surface".

Guidance systems such as cone/lanyard ropes and audio visual systems are required to enable the refuge chamber to be reached under conditions of extremely poor visibility, disorientation, confusion and shock.

Chamber location guidance will be a key feature, to promote the ease of access to personnel with limited travelling capabilities due to capacity limitations with self-contained breathing apparatus.

Refuge chambers will need to be capable of affording protection against explosions, fire and inertisation, both to withstand an initial incident and be available to survivors, and to protect the "inmates" against subsequent events.

The design parameters for refuge chambers therefore need to address:

- supply of fresh air;
- air-lock arrangements;
- water and food supplies;
- communication;
- fire and explosion protection;
- lighting ; and
- retrieval of personnel.

An inspection regime for the overall system will be required to maintain all facilities in effective working condition.

Recovery strategies for personnel in a refuge chamber will be required and needs to address the possibility of atmospheric stabilisation by inertisation prior to mine-re-entry by rescue teams.

Personnel recovery can be envisaged in three main ways, these being;

- through the mine to the entry points under relatively normal conditions, either accompanied or unassisted;

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• through the mine to the entry points under breathing apparatus and assisted by a rescue team on foot, or with a manned vehicle or with a remotely controlled vehicle;

• through a large diameter borehole directly to the surface.

Industry developments - Australia

There are instances of refuge chambers in use in Australia, the most notable being the Capcoal Central example. This chamber is a static chamber, built in a reasonably central proximity to a number of working areas and featuring a large diameter borehole (mini vent shaft) in close proximity for emergency egress.

The use of personal self-contained self rescuer breathing apparatus has recently become an imminent requirement in Queensland, and this together with a "cache" system to provide actively promoted for immediate usage in underground coal mines.

A research submission is currently being considered that if successfully funded will ultimately deliver a rescue vehicle. Further work is also planned in the area of inertisation, which with the recent purchase of the GAG 3 jet equipment, provides a strong inertisation capability for Australian mines.

An associated development in the Australian industry that is most pertinent to the overall safety of underground coal mines is the adoption of standards applicable to all structural ventilation devices. These standards and the resultant improved structures will enhance the possibility of personnel survival in the event of an explosion, such that the provision of post-explosion facilities may become more relevant.

It should be recognised that the adoption of refuge chambers and associated strategies is not an alternative to "escapeways" but that the refuge chamber may also have some benefit in the instance of a major mine fire.

MICROSEISMIC DETECTION SYSTEMS

Introduction

In the US, MSHA maintains a microseismic monitoring facility which can be deployed on the surface, and can locate signals generated by miners striking the roof and floor of the mine. There is also a portable system which could be taken into emergency zones to help locate people or significant events. These form part of the emergency response facilities of MSHA. The MSHA surface system is described and the potential for application of microseismic emergency monitoring in Australia is discussed.

The MSHA seismic location system

A seismic location system was developed by the USBM in the early 1980s to locate workers trapped underground. A truck with geophysical monitoring equipment was assembled, and a standardised routine for trapped workers devised. The deployment procedures and elements in the system are:

1. A truck with geophone arrays and recording equipment is deployed on the surface above a mine where workers are trapped. Up to seven arrays are used and the geophones located with GPS.

2. Trapped workers are notified that the monitoring system is in place by firing three explosive charges.

3. Signals are then generated by trapped workers striking the roof and floor of the mine 10 times in succession. If no response is received from the surface, the 10 blows are to be repeated every 15 minutes.

4. When a signal from the miners is received at the surface, and the source located, 5 shots are fired to indicated to the underground workers that their signals have been received and that they are located.
Experimental systems were reported by Shope et al (1982) and it was claimed the system was effective to 600m. An operational unit is maintained in a state of readiness by the Mine Safety and Health Administration at Pittsburgh Research Centre, Brucetown. It is reported that the current system can detect signals at a range of 450m and can locate the signal source to within 30m. The unit is tested in field trials 3 times a year.

Miners are issued with a hat sticker by MSHA which describes the procedures to follow if they find themselves trapped underground.

Although the MSHA system is directed to the precise location of trapped workers underground, it also performs an extremely important function of confirming that trapped workers are alive. This is of great value to the rescue operation even if their precise locality cannot be determined.

MICROSEISMIC APPLICATION FOR AUSTRALIA

Surface systems

The MSHA equipment is entirely surface based. The technique detects the characteristic signal made by the miner’s blows in the mine, and locates the signal source by triangulation of the wave paths detected by arrays on the surface. This utilises source location routines commonly used in seismology. Arrays of geophones are used to reduce extraneous surface seismic sources, so the signal initiated by the underground miners is enhanced. A surface system does not require any prior development of infrastructure by the mine site.

Requirements for an effective surface system include:

Access to the surface

This can be restricted by rugged terrain, cultural developments or overlying water bodies

Propagation of signals

An essential requirement for an effective surface system is that the underground signals must propagate to the surface. The American experience indicates that signals can be detected up to 450m in some strata conditions. Strata may attenuate or divert signals so they cannot reach the surface. Microseismic experiments in Australian mines have encountered difficulties monitoring signals from depth, and causal factors include

- source too deep;
- strata structurally disrupted;
- layered high and low velocity beds;
- rapid signal attenuation in goaf and gassy units; and
- overlying Tertiary and Quaternary sediments and volcanics.

From the Australian experience 450m detection distances is expected to represent the best possible performance, and this would only be in the most favourable circumstances. Areas with thick Tertiary and Quaternary sediments at the surface would prevent any surface detection of signals from working depths.

Potential conditions affecting signal detection in Australia coalfields include:
<table>
<thead>
<tr>
<th>Negatives</th>
<th>Positives</th>
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</thead>
<tbody>
<tr>
<td>Southern coalfield</td>
<td>Mines generally too deep</td>
</tr>
<tr>
<td></td>
<td>Surface access</td>
</tr>
<tr>
<td>Western Coalfield</td>
<td>Solid rock</td>
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<td></td>
<td>Some shallower workings</td>
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<tr>
<td>Newcastle</td>
<td>Surface access</td>
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<td></td>
<td>Some shallower workings</td>
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<tr>
<td>Hunter Valley</td>
<td>Localised surficial deposits</td>
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<tr>
<td></td>
<td>Access generally good</td>
</tr>
<tr>
<td>Bowen Basin</td>
<td>Extensive surficial deposits</td>
</tr>
<tr>
<td></td>
<td>Shallower workings</td>
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An analysis on a mine by mine basis would show that some locations potentially have excellent conditions for surface detection, and others are unlikely to be successful sites at the depths of current mining.

**Deployable systems**

A surface microseismic system would have to be maintained at the ready if it was to be used in emergencies. There would have to be a standing capacity and emergency access to trained operators. This would logically be a shared industry facility rather than the responsibility of any operator. The microseismic techniques and equipment required are well known and a number of Australian consultants and agencies could provide the technical requirements.

**Borehole systems**

Some of the difficulties with surface systems in Australia could be overcome by using geophones installed in boreholes. These can be placed in competent rock near to the worked seams, to reduce propagation distances and to avoid wave paths travelling through problem materials.

Many exploration boreholes are drilled around underground developments, and it would be simple and inexpensive to install a geophone during grouting of the holes. These geophones could be used during emergencies.

The position of boreholes is based on other specific needs so it is unlikely that a mine would be able to set up a network which would allow triangulation coverage of all underground areas. Most mines however, would get a coverage of boreholes which would provide at least one monitoring point throughout the workings. This would at least allow confirmation that trapped workers were alive, and indicate that their location was within the detection area of the activated geophone. If triangulation and location detection were not required, each borehole would need only a basic monitoring device to record the geophone signals, so that any repeated seismic waves generated by the miners could be identified. These units could be deployed along with other mine rescue service activities, and would not require specialised personnel.

**Small portable microseismic systems**

Small portable microseismic systems can be transported and operated in any emergency situation. MSHA has such a unit which has also been used at natural disasters such as earthquakes. These units use well known technology and commercial groups could either provide equipment or a service.
Introducing the technology in Australia

The lack of any local experience discourages the implementation of microseismic technology for rescue operations in Australia. This could be overcome by undertaking demonstrations of surface and borehole configurations by using existing geophone installations and conventional seismic survey instrumentation.

Success in these trials could lead on to the provision of seismic personal location service for emergencies, which could include the following elements:

- adoption of routine geophone installation in boreholes;
- establishing local mine performance;
- training of workers to use the system; and
- training and equipping of mines rescue services.

SUMMARY

Microseismic location from the surface, of underground miners hammering, is demonstrated by MSHA in the US, for distances up to 450m at 30m accuracy.

Only a small proportion of Australian mines are believed to be amenable to surface detection methods because of unfavourable access problems or geological conditions.

Mines with favourable conditions could do local tests, and there are a number of Australian groups with the technical capability to advise mines.

Geophones installed in mine boreholes could provide a good coverage of most mines and provide a way to verify miners were still alive, and their general location. A simple instrument would need to be developed to monitor the geophones, and could be deployed by mine staff or mines rescue brigades.

Widespread take up by the Australian mining industry is dependent on successful demonstration of the technique.

References


Large diameter drilling

Status report is currently being compiled an the current worldwide best practice (published Q1 1998)

Rescue emergency vehicle

A large amount of effort and research development has gone into the first concept machine called the NUMBAT over this decade. Trials have been undertaken over the last 3 years at selected mines in New South Wales and Queensland. All of these trials have proved certain parts of the technology and directed other developments in line with current thinking in mines rescue response from the industry.
The current vehicle has proved that it is suitable in some instance for Emergency Response to determine environmental conditions (gas monitoring, atmospheric conditions) with video response as a tool to provide a low risk technical exploratory capability before rescue or recovery decisions are made.

During 1997 the changes in approach to rescue and recovery techniques defined shortfalls to our overall response, which had relied largely on luck and hope.

The changes now focussed on a risk approach where we could control the conditions, accurately assess the environmental changes and minimise the consequential damage to other persons.

Legislative changes in both NSW and Qld to the use of hazard management, Management Plans, self rescuers and systems, self escape training and in Queensland the provision of (2) GAG engines for mine inertisation have changed our "Rescue" response.

The provision of portable refuges fixed or and the probable inertisation of a mine to effectively minimise the risk of uncontrolled explosions means that a rescue response vehicle needs to be developed. Both NSW and Qld cross industry groups are working together to develop a vehicle capable of high speed access with life support systems for up to 15 people that can operate in an inert atmosphere. Much of the original NUMBAT technology will be directly transferable with the main focus on the cabin design, life support and engine systems. This RandD project will be reported on later in 1998.

MINES RESCUE STRATEGY DEVELOPMENT

Introduction incident management

The Underground Mines Rescue Strategy Development group selected a committee to look at the information, equipment and other criteria needed in an emergency to manage an incident. The issues were related to the implementation of Task Group 4 recommendations 15, 16, 17.

15 Capability to model ventilation and the mine environment following an incident
16 Integrated emergency preparedness guidelines for mine operators and emergency services, common to both Qld and NSW
17 Development of computer-based emergency decision support system for incident management and training

The incident management sub-committee included:

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<tr>
<th>Name</th>
<th>Organisation</th>
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<tr>
<td>Gibson</td>
<td>MRSNSW (Chairman)</td>
</tr>
<tr>
<td>English</td>
<td>QDME (Secretary)</td>
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<td>Garland</td>
<td>QMC/North Goonyella</td>
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<td>Gillies</td>
<td>University of Qld</td>
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<td>Hendricks</td>
<td>NSWMC/BHP Collieries</td>
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<td>Hutchings</td>
<td>QMRB</td>
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<td>Stothard</td>
<td>CFMEU</td>
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R15 Ventilation and environmental modelling

Ventilation modelling to include:

- modelling of post incident mine ventilation and atmosphere to be a required element of Mine Safety Management Plans;
- development of learned ventilation and fire control responses for different incident scenarios and locations, pre determined for each mine, with plans prepared and personnel trained in appropriate action plans.
• determining the explosibility of atmospheres;

• distillation profiles for the coal in each mine to be determined and incorporated into models;

models to interface with standard mine planning packages and kept up to date;

R15 Progress Report

• Survey of industry ventilation modelling;

Of 18 mines using MINVENT 17 operated by consultants;

Of mines using VENTSIM only 2 use consultants;

5 other mines use 3 other modelling systems;

10 (smaller) mines currently do not model; and

No contact/responses from 9 mines

• VENTSIM

Windows based system designed and supported in Australia; and

User friendly, favoured by experienced mine ventilation engineers.

• MINVENT

Currently favoured by consultants, perhaps because of better printing facilities; and

Limited support available.

• Post-Incident ventilation/environmental monitoring

• Two post-incident ventilation/environmental monitoring systems identified

M-FIRE, developed by MSHA for simulation of mine fires; and

POZAR, developed by Polish Ventilation Academy to enable mines to simulate mine fire effects in multi-face/seam mines and evaluate intervention strategies.

• M-FIRE

– Public domain software, limited adoption and development, only validated once

• POZAR

– Routinely adopted by Polish mines, intervention evaluations accepted by Polish authorities;

Trialed at Collinsville, and to undergo further evaluation at North Goonyella; and

Current marketing strategy limits access.
R15 - Selection Matrix for Industry system

- User friendly (Windows based);
- Capable of being broadly adopted as industry standard;
- Acceptable to new generation Ventilation Officer;
- Integrate with mine planning/survey systems;
- Capable of modelling dynamic situations; and
- Capable of integrating real time P, Q, and T data.

R15 - Recommendations Being Considered

- Support concept of statutory Ventilation Officer and development of appropriate competencies
- Competencies to include ventilation modelling in static and dynamic post-incident applications
- User friendly ventilation software should be adopted as industry standard
- Develop capability to integrate real time P, Q, T

Develop integrated industry network and expert system for post-incident monitoring and evaluation

R16 emergency preparedness Guidelines

Guidelines should address:

- Roles and responsibilities of mine management and emergency services in an emergency; and
- Consolidation and integration of emergency response procedures developed through principal hazard management plans

Development of a common training program as a joint pre-requisite for mine managers and undergrounders accreditation in Qld and NSW

Development, maintenance and assessment of appropriate competencies.
R16 progress report

- MRBNSW has developed Guidelines for mines rescue organisations and personnel, currently being evaluated by QMRS Sub-Committee to develop Guidelines for operators, with initial draft due end-November 1997

R 17 computer-based emergency decision support system

- Incident Management enhanced by decision support system that:
  - provides strategic information on the mine and the incident;
  - provides an analysis of the developing situation;
  - presents prioritised options available; and
  - provides training system.

R17 PROGRESS REPORT

- ECAS system developed by ACIRL under NERDDC funding in 1989-90 requires significant enhancement and more user friendly platform;
- Literature search underway to identify other possible systems; and
- Systems utilised by armed forces and emergency services to be investigated

CSIRO trial of Virtual Reality Modelling System including mine emergency applications supported