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COAL MINING OPERATORS’ GEOTECHNOLOGY COLLOQUIUM

TRIBUTE TO
DR ALAN JAMES HARGRAVES

15 February 2001

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Illawarra Branch of The Australasian Institute of Mining and Metallurgy

in association with

Seedsman Geotechnics Pty Ltd
Strata Engineering (Australia) Pty Ltd
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PROCEEDINGS OF THE COAL MINING OPERATOR’S
GEOTECHNOLOGY COLLOQUIUM

TRIBUTE TO DR ALAN JAMES HARGRAVES

UNIVERSITY OF WOLLONGONG, NSW

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The Illawarra Branch
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FOREWORD

Four Aussies (Ross Seedsman, Ken Mills, Rod Doyle and Russell Frith), drinking late into the night in a bar in West Virginia, conceded that an Australian Geotechnical Conference - one to rival the one they were attending - was a worthy goal. On the one hand it would be exceedingly difficult to compete with such a successful array of geotechnical professionals, while on the other hand, no matter what the field of endeavour, Australians are always up to the challenge. It was also heartily agreed that it would be fitting to honour one of our own geotechnical founding fathers – Dr. Alan James Hargraves.

As successful Australian mines continue to improve production levels, to reduce costs and to improve general safety conditions, greater emphasis will be placed on all facets of geotechnical knowledge. This Colloquium is directed towards this end. Alan was very active in gas and outburst research; topics on gas and outburst are discussed at this Colloquium.

Almost all the papers presented at this Colloquium are invited papers and represent a cross-section of the coal industry in the States of Queensland and New South Wales. We are extremely grateful to the authors who accepted the invitation to contribute papers and present them at the Colloquium.

The Colloquium has been generously supported by the following organizations:

- STRATA ENGINEERING (Australia) Pty Ltd
- SRK Consulting
- GLENCORE COAL AUSTRALIA Pty Ltd
- Valley Longwall Drilling
- Illawarra Coal BHP Minerals
- Allied Coal
- Seedsman Geotechnics Pty Ltd
- Strata Control Technology Pty Ltd
- Ground Consolidation
- Jennmar Australia
- ASIMS

The AusIMM Illawarra Branch is extremely grateful to all for their support. The success of the Colloquium is also in no small part due to the personal commitment of a number of individuals who have worked so hard behind the scenes. Special thanks are due to session chairpersons, mine visit organizers, Maureen Prince, the University of Wollongong staff – especially Leonie McIntyre, Patricia Cooney and Stuart Chambers and finally the Organising Committee.

Ernest Baafi
University of Wollongong
PREFACE

After a long break I met up with Dr Hargraves once again. At this meeting I passed onto him a bottle of Black Label Johnny Walker. He also had a gift for me - a job! I have that instruction pinned to my wall at work, it says ‘Prepare a Summary of the Metalliferous Ore Bodies in the Shoalhaven River Valley.’ This may seem rather an inappropriate task for a Coal Geologist, but Alan and I explored the Shoalhaven together examining ore deposits and the wonderful Australian bush. He suggested that it would be a fitting way to remember my late father, also a keen explorer. This simple act of Alan’s reflects the man, while he was compassionate about others he was always thinking to further knowledge and understanding. At this meeting I raised the topic of this Geotechnological Colloquium and with typical modesty he suggested that it was Con Martin who should get the credit for recognising the need to establish a Rock Mechanics and Strata Control Department in the BHP mines. But recognising the need and setting out to achieve that task are two different feats in their own right.

Upon reflection, my experience of Alan highlights a quite, private, humble and reserved man. While working for him I learned little of his family life, but observed his intense passion for all aspects of mining, especially prospecting for gold and precious gemstones. Throughout his working life and into his retirement he was dedicated to the role of developing ‘Rock Mechanics and Gas’. He was also committed to: the Australasian Institute of Mining and Metallurgy, the University of Wollongong and all that furthered coal mining knowledge. At our meeting I finally learned that Alan was devoted to his late wife and family. In short Alan is a doyen in his field.

Today’s Colloquium honours Dr Alan James Hargraves for his pioneering role in both gas and geotechnics in the Australian Coal Mining Industry. But it also serves a secondary purpose and that is to encourage us all to continue with the example Alan has set us. Alan’s passion for mining, his humble focus on making mining safer and his constant focus and drive, urge us all to ‘pass the baton’ onto a new generation. If we accept that baton it will ensure that safe mining conditions are an eminent focus in all of our professional lives. I trust that we will continue with Alan’s pursuit of technical and safety education.

Rod Doyle
Geologist, Anglo Coal
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ALAN JAMES HARGRAVES

BY BOB KININMONTH

AND

PERSONAL TRIBUTES
CONTRIBUTIONS OF ALAN JAMES HARGRAVES

Bob Kininmonth

Retired Senior Inspector of Coal Mines, Wollongong

Alan James Hargraves is rightly regarded as a pioneer in the development of systems to understand the occurrence of gas in coal measures and control outbursts of coal and gas in Australian coal mines. His many colleagues throughout the mining industry have followed his achievements. This colloquium gives the opportunity to formally record the recognition which his work deserves.

Alan was born in Melbourne on 5/10/1919. He attended Melbourne High School with which he still has close associations. He was granted a scholarship to the University of Melbourne where he gained a Bachelor of Engineering Science and a Bachelor of Mining Engineering.

Between 1937 and 1941 he gained practical mining experience at Captains Flat in NSW, at MT Lyell in Tasmania and at Woods Point in Victoria where he was working at the time of the devastating bushfires of 1939.

In 1941 Alan became assistant to the Mine Manager at Aberfoyle Tin NL a position which he held until 1947 when he became the Engineer-in-Charge of Exploration for the Zinc Corporation Ltd. In this position he was involved in assessing deposits in Tasmania and South Australia. It was during these periods that he became interested in mineral specimens and gemstones, an interest which he still retains.

In 1950 he took the position of Assistant Tunnelling Superintendent with the SEC of Victoria which involved him in work on the Kiewa Project and the Yallourn Power Station. Between 1951 and 1953 he returned to Tasmania as the Tunnelling Engineer with the Hydro Electric Commission. It was his appointment in 1953 as Senior Lecturer in the Department of Mining Engineering at the University of Sydney which brought him into close contact with the coal mining industry and led to his pioneering work in the understanding of gas emissions and outbursts.

The fatal outbursts at Metropolitan and Collinsville in 1954 indicated a need to concentrate on reducing the risk of outbursts in areas of high carbon dioxide in the seam gas. Alan’s work at those two collieries involving large hole drainage and inducer shotfiring together with the introduction of the emission testing device, the isobaric desorbometer, improved the control of mining conditions. He used this work as the basis for gaining a PhD degree at Sydney University.

As mining was carried out under deeper cover and particularly away from the South Coast escarpment it became apparent that outbursts were liable to occur in areas of high methane content. The Management Plans now in place to control and prevent outbursts are a development from an understanding of the phenomena which has followed the experimental work of Alan Hargraves and later workers in the field including Les Lunarzewski and Ripu Lama.

Apart from his work on gas and outbursts Alan also has Mine Managers Certificates Metalliferous for NSW and Victoria and he has been a prolific writer of technical papers mainly on gas or gas and stress affecting coal mining. The majority of his 76 papers are sole author but he is associated in 24 co-author papers.

His international reputation in the field of gas drainage and outbursts has resulted in his appointment to a number of local and overseas posts namely:

- 1966 Visiting Professor, West Virginia University, USA
- 1969-76 Honorary Associate, Department of Earth Sciences, Macquarie University
- 1977 Visiting Professor, University of Hokkaido, Japan
- 1978-97 Visiting Professor in Mining Engineering, University of Wollongong
The need for research facilities particularly in the field of strata control was recognized by the industry in the early 1960’s and this resulted in the establishment of Australian Coal Industries Research Laboratories. BHP/AIS as a major producer with increasingly difficult and complex mining conditions decided to consolidate their in-house expertise and in 1968 Alan was appointed as Superintendent of Rock Mechanics and Strata Control based in Wollongong.

In that position he brought together a staff of specialists who were available to provide a service and a scientific approach to problem solving in the group. Alan’s personal interest in his staff coupled with his challenge and encouragement to individuals to perform to the best of their ability, a characteristic which he brought with him from his University days, proved successful in building staff loyalty and professional competence. Many of his past students and work colleagues who have had successful careers in industry recognize and appreciate the influence which he has had on their careers.

In 1981, Alan was appointed as Principal Research Associate Steel Division Collieries, Broken Hill Proprietary Co Ltd, a position which he left in 1982 to take up a private consulting practice.

Throughout his working life Alan has maintained a close association with the Australasian Institute of Mining and Metallurgy which he joined as a student in 1937. He became a member in 1956 and a Fellow in 1994. He has seen service as Secretary of the Sydney Branch between 1958 and 1965, as Panel Leader of the Coal Panel of the Proceedings Committee between 1983 and 1994 and he was elected Councillor of the Institute for two periods 1966-69 and 1987-90.

Alan’s work in the Illawarra as The AusIMM Branch Chairman, Secretary and Committee Member at various times and his enthusiasm and dedication to contributing, convening and editing various Symposia has been instrumental in making the Illawarra Branch so active and successful. His work in this area has been recognized by his election as an Honorary Branch Committee Member.

It is difficult to visualize Alan as being “retired” and in recent years he has made major contributions to the publications of the Institute by way of his editing with Con Martin of Monograph 12 Australasian Coal Mining Practice and his co-authorship of Monograph 21 History of Coal Mining in Australia.

We join in honouring him for his technical expertise and achievements and for his promotion of, and dedication to, the organisation of educational conferences, symposia and colloquia similar to the one we are now attending.
PERSONAL TRIBUTES

- I have known and worked with Alan Hargraves for thirty years or so. Alan was prominent, in hands on way, with letting scientists, mining engineers and mine managers to work together to overcome major problems facing higher capacity systems of mechanized mining in the NSW Southern Coalfield and then Queensland Bowen Basin. Of particular importance was his pioneering work in the understanding and prevention of instantaneous outbursts of coal and gas, and other gas control measures. Alan’s mostly direct and pragmatic approach and particularly his part in establishing and maintaining useful, longterm contact networks for frank technical exchange, both locally and worldwide has, in my opinion, been of great value to our underground coal industry.

  Alan Fisher, Secretary, NSW Coal Mines Managers Association

- Even though I was only at BHP/AIS Rock Mechanical and Strata Control (RM & SC) for a relatively short period, the encouragement I received during my stay and the interest kindled has kept me around the area of coal mine geomechanics for most of my working life. My subsequent time at CSIRO has allowed me to remain engaged in the evolution of coal mining geomechanics in Australia. A lot of that effort has directly built on the initiatives I witnessed while I was at RM & SC, and those areas with which I was directly involved. My on-going association with Alan over the last thirty years has continued to influence me, both professionally and at a personal level. My best wishes are with him at this moment.

  Jim Enever, Principle Scientist, CSIRO Petroleum

- Alan’s passion for mining, his humble focus on making mining safer and his constant focus and drive, urge us all to ‘pass the baton’ onto a new generation. If we accept that baton it will ensure that safe mining conditions are an eminent focus in all of our professional lives. I trust that we will continue with Alan’s pursuit of technical and safety education.

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  Bob Kininmonth, Retired Senior Inspector of Coal Mines

Photo 1  Alan and Con Martin at Illawarra Rock Bolting Symposium 1971
Photo 2  Alan and SEC engineer during the tunneling of power station cooling water conduits in a soft ground adjoining Yallourn Open Cut

Photo 3  Alan and other investigators of Collinsville Outburst 1954

Photo 4  Alan and friends on a Cockatoo train trip organised by Illawarra Branch of The AusIMM
Photo 5  Alan on his way to look for gold 1994

Photo 6  Alan and Illawarra Branch Members (Bob Kininmonth and Alison Ogden) at Oberon looking for gold 1994

Photo 7  Alan and The AusIMM Mineral Heritage looking for gold at Oberon 1996
EARLY DAYS OF ROCK MECHANICS AND STRATA CONTROL

BY JIM ENEVER
EARLY DAYS OF ROCK MECHANICS AND STRATA CONTROL

Jim Enever
Principle Scientist
CSIRO Petroleum

ABSTRACT: BHP/AIS's Rock Mechanics and Strata Control (RM & SC) Group was formed in the late 1960's to address a range of issues then facing BHP/AIS's underground coal mining operations. Building on a local tradition of innovative rock mechanics and a global body of research targeted at sedimentary strata behaviour, RM & SC tackled a number of critical areas and made substantive contributions in all of these. Alan Hargraves, as the leader of the group, had a significant influence on all activities, as well as continuing personal research in the area of seam gas management. A direct line can be traced from the initiatives implemented by Alan at RM & SC and many of the modern advances in coal mine geomechanics.

INTRODUCTION

When BHP/AIS decided to set up a small research and technical advisory group in the late 1960's, it was Alan Hargraves who Con Martin (the then Superintendent of Collieries) prevailed on to head it up. Alan had already enjoyed a fruitful relationship with BHP/AIS in the area of outburst prediction/control and gas drainage. I had the privilege of joining this group in its infancy in 1970 soon after it was established. During my two years or so with RM & SC I was part of an expanding sphere of activities covering gas related problems, mining subsidence, roadway support, longwall mechanics, chain pillar design, slope stability and the underlying basic rock mechanics. What follows is a personal recollection of my time at RM & SC and the impact of RM & SC on subsequent developments.

ROCK MECHANICS AT THE END OF THE 1960's

By the end of the 1960's rock mechanics had been around as an identifiable discipline for about 15 years. Much of the early developmental effort for the discipline had occurred in Australia, notably through the agency of the Snowy Mountains Scheme and the efforts of the Hydro Electric Commission in Tasmania. Australia's major metalliferous mining operations had formed separate rock mechanics groups.

Globally, rock mechanics as applied to the underground coal mining industry had made notable steps through the efforts of groups such as the United States Bureau of Mines, UK National Coal Board, the Bergbau Forschung in Germany, the Chamber of Mines in South Africa and the various research institutes of the Eastern European Block. Important work was also being conducted at various universities. Collectively, these efforts had raised the level of understanding of the mechanics of sedimentary strata behaviour to a point where solutions could be conceived to a range of problems then facing the coal mining industry.

In Australia, ACIRL had come into existence in the 1960's to apply this learning to the Australian black coal industry on a broad front. Outside of ACIRL, focussed research efforts were going on at a number of universities. To BHP/AIS the moment appeared ripe for the formation of a small group who could build on this background to address a range of problems facing BHP/AIS's operations.

THE AUSTRALIAN COAL INDUSTRY AT THE END OF THE 1960's

By the end of the 1960's, continuous mining was widely in use throughout Australia, with associated issues of compatible roadway roof support practice. The instance of secondary extraction had increased to a point where problems were being encountered with the impact of mining subsidence on surface structures and other...
infrastructure. Longwall and shortwall operations were increasing in frequency, with the pressing need for a better understanding of the mechanics of caving and design approaches for face supports for Australia's relatively strong (compared to Europe) and highly stressed roof sequences. Design of optimum coal pillar dimensions was an ongoing issue, particularly in conjunction with longwall and shortwall operations, in an attempt to maximise recovery. With the increase in coal production rates through the 1960's, problems associated with the management of seam gas became more and more critical. Spontaneous outbursts were sufficiently frequent to pose a real hazard, while increased rates of gas emission into workings was taxing contemporary ventilation practice.

By 1970, all of these issues had become significant for BHP/AIS's underground operations. At the same time, the impending expansion of surface coal mining suggested the need for an attack on potential problems of slope stability. This then was the scene facing the fledgling Rock Mechanics & Strata Control group when I started work there as a new graduate in 1970.

**ROCK MECHANICS AND STRATA CONTROL**

**General Structure**

During my time at RM & SC, the group was broadly structured to address the specific issues outlined above. To support these focussed efforts, a capability was developed in the areas of instrumentation and numerical modelling.

**Roadway Roof Support**

Work in this area concentrated primarily on the introduction of resin roof bolting to routine practice. At the time, resin bolting was a new technology requiring fostering and the establishment of basic operating criteria. Much of the effort went into pull-out testing of different resin/bolt/installation procedure combinations to determine yield and ultimate load capability. A wide range of combinations was investigated to establish suitability for a range of roof conditions and mining practice. Monitoring of roadway convergence and roof strata deformation was used as a measure of the success or otherwise of different patterns of bolting and bolt lengths. The outcome of this work program was the establishment of an empirically based set of guidelines for the use of resin bolts.

**Subsidence**

At the time of my arrival at RM & SC, an intensive program of subsidence monitoring had been instigated over areas of total extraction on the NSW north and south coasts. This program was continued and expanded during my time at RM & SC to produce a substantial database allowing the nature of subsidence in Australia's stronger/stiffer (compared to other countries) strata environment to be investigated. Survey results were converted to vertical and horizontal strains and related to parameters such as depth of cover and extraction width. This information was compared with similar data from other countries (notably the UK and US) to establish similarities and differences, and thereby the applicability of design procedures based on these overseas databases to Australian conditions.

**Mechanics of Caving**

Early attempts at mechanised longwall mining in Australia used equipment based essentially on UK practice. This equipment was found generally to be inadequate for Australian conditions. Design of powered support systems in the UK and other European countries, was, at the time, based on an understanding of caving mechanics developed from observations made in the relatively soft, "freely" caving, environments of these countries. Under these conditions, caving tends to occur regularly as extraction advances and the bulking behaviour of the caved material ensures that most of the load is carried through the goaf rather than via the face supports. The stiffer/stronger roof sediments typical of Australian coal basins tend, however, to cantilever out behind the face line, caving periodically and with a lower bulking factor, with commensurate higher loads on face supports. This style of behaviour was beginning to be appreciated at the time I joined RM & SC. The need for an appropriate means of selecting face support capacity for the new generation of longwalls then coming on stream, as well as the innovative shortwalls being pioneered by BHP/AIS at the time, dictated the need for an improved understanding of caving mechanics.

A program of face load monitoring and associated strata deflection monitoring was insigated to explore the relationship between strata behaviour and face support loading. This was complimented by detailed observations of the nature of the caving process occurring in the goaf. Information like this eventually provided a basis for the rational selection of support capacity for Australian conditions.
Chain Pillar Performance

To investigate the loading regimes on longwall chain pillars, a program of stress monitoring was instigated to study the magnitude and spatial distribution of the vertical load patterns occurring in chain pillars as the extraction line approached and passed. The aim of this work was to develop a means of optimising the size of chain pillars to meet the potentially conflicting needs of short term roadway protection and long term yielding to ensure that goaf loading remains as uniform as possible to avoid adverse stress abutments being developed.

A simple hydraulic cell was developed in-house to facilitate this monitoring. A large number of these were installed in a variety of geometric configurations and the behaviour of chain pillars tracked in time and space. Information of this type was important in the ultimate development of an approach for chain pillar design.

Seam Gas Management

The pioneering work undertaken by Alan on seam gas emission prediction and the prediction of outburst potential was incorporated into the activities of RM & SC from its inception. The Hargraves Emission Meter continued to be deployed widely and experience built up from its use in a variety of environments. In my time at RM & SC, innovative work was undertaken in longhole in seam drilling for pre-drainage, and surface holes for goaf drainage. A more complete account of Alan's contributions to seam gas management practice in Australia will be covered elsewhere in this colloquium.

Slope Stability

At the time of my stay with RM & SC, slope stability had not become the major issue that it became later on as the importance of surface mining increased. Initial aspects focussed on the major role of engineering geology in rock slope stability, through the pursuance of a number of case studies.

Numerical Modelling and Instrumentation

In the early 1970's the Finite Element Method (FEM) of Stress Analysis had been around for a number of years, although its application to problems in geomechanics was still relatively new. Up to that time, "modelling" in the coal mining context had generally meant either the application of some form of analogous analytical solution, or the use of a physical model. The advantages offered by numerical stress analysis had become obvious to those involved in the area. With this in mind, a program was instigated at RM & SC to introduce the FEM method. This involved a combination of refinement of existing packages and development of original software.

A range of instrumentation was developed during my time at RM & SC to compliment the various efforts outlined above. Of note were remote convergence meters and pillar load monitoring cells. Major effort was devoted to the development of a capability for *insitu* stress measurement, both from underground and from the surface. In the underground context, the overcoring technique was employed in conjunction with a soft inclusion strain measurement cell. From the surface, emphasis was placed on the use of the anelastic strain recovery (ASR) technique.

The Impact of Alan Hargraves

Apart from his personal contribution in the area of seam gas management, Alan was a major influence in all the areas outlined above. His vision and success in selecting appropriate personnel opened the way for the evolution of the integrated effort that became RM & SC. Alan's insight contributed significantly at various times to the progress of efforts in all areas.

**SUBSEQUENT YEARS**

During my on-going association with the Australian coal industry in the thirty or so years since my departure from RM & SC, I have witnessed the growth of coal mining geomechanics into a mature branch of engineering with a high level of acceptance by the industry. It is interesting to reflect on how the initiatives instigated by Alan at RM & SC, have progressed over the intervening years:

- the development of highly effective roadway support systems, tailored to specific situations and based on a sound understanding of the mechanics involved, and an ability to design support arrays objectively;
• the development of an ability to realistically predict mining induced subsidence, based on a sound understanding of the fundamental mechanics involved;
• the attainment of a comprehensive and detailed understanding of the mechanics of longwall caving, and the ability to reliably specify face support requirements;
• the arrival at a process to design optimum coal pillar dimensions to meet conflicting needs;
• dramatic advances in the technology of seam gas drainage and the management of seam gas, as well as the underlying understanding of coal seams as gas reservoirs;
• the emergence of the routine application of modern slope stability principles to surface coal mining;
• the widespread use of a number of numerical modelling techniques to address a range of problems, completely replacing alternate modelling methods;
• the general acceptance of *in situ* stress measurement as an essential component in investigations for new mining operations and the development of solutions to specific operating problems.
OUTBURSTS
MANAGEMENT AND CONTROL
BY JOHN HANES
OUTBURSTS – MANAGEMENT AND CONTROL

John Hanes
Co-ordinator of ACARP In-Seam Drainage and Gas Research

ABSTRACT: This paper summarises some of the development of the understanding and management of outbursts in Australia, reviews the current situation and provides some thoughts on necessary research and development. It cannot be and is not comprehensive. Outbursts are better understood, prevented or controlled than they were 20 years ago. This is partly a result of the considerable amount of research into and documentation of outbursts and gas by a relatively small group of outburst pioneers supported by the coal industry. The result has been fewer outbursts in general, and the safe mining through a few potential outburst zones without risk to personnel. It is not time to rest on our laurels. Outburst and gas investigation needs to be continued, supported by industry, so that all outburst related phenomena and advances in gas management can be documented and communicated to all players.

INTRODUCTION

Hargraves, 1980 stated “The problem of instantaneous outbursts remains unsolved after over a century of events and the investigation of the mechanism of the phenomenon and means of treating it…There is no simple litmus test to identify with certainty that a coal is outburst prone. There is no simple alarm device to give adequate warning time of an impending outburst in a place. The understanding of the mechanics of the phenomenon is lacking…We are learning but disappointingly slowly”.

In 2001, are we any further towards understanding and solving the outburst problem in Australia? There has been a considerable increase in the understanding of outburst mechanisms and the methods to control outbursts in the intervening years since 1980, but the cost in lives has been too high. There is more work to be done.

Outburst research and investigations have been conducted in 3 main fields: prediction, prevention and control. Considerable progress was made during the last 20 years in the fields of prediction and prevention with lesser progress in control, mainly because of the relative success of preventative measures adopted by industry.

PREDICTION

Rank, gas composition, geometry

Hargraves (1980) associated increasing outburst proneness with increasing rank of coal. This was due to the higher capability of coals of higher rank to sorb gas. Carbon dioxide (CO2) gas makes coal more prone to outbursts and the outbursts are more violent than with methane (CH4) because of coal’s higher sorptive capacity for CO2. Hargraves recognized differing proneness of places in a mine based on world experience: the most prone place was in cross measure drivage or sinking into a seam, followed by drivage in seam, longwall advance, longwall retreat and pillar extraction. The only Australian outbursts at the time had occurred in development headings. Since then, in 1998, two outbursts occurred on a retreating longwall face at West Cliff Colliery (Walsh, 1999).

Geological Structures

Experience in the Bulli seam is that generally all outbursts have occurred when the face has intersected a geological structure, sometimes as small as a few centimetres width of mylonite with no vertical displacement. There have been some exceptions, one of which was a very small burst at Tower Colliery and a few relatively small bursts in CO2 rich coal at West Cliff Colliery (Lama, 1994). Once a structure has been intersected with outburst signs in one heading it is assumed that its projected intersection in subsequent headings will also be outburst prone.
Outbursts at Collinsville occurred on structures (Biggam and Robinson, 1980) but outbursts at Leichhardt generally occurred without the presence of any major unusual structure. The Gemini seam was very heavily anisotropically cleated which reduced its bulk strength. The fatal outburst of 1978 occurred from coal which was partly brecciated. The brecciation was associated with thrust faulting indicated ahead of the face by high-resolution surface seismic.

In some locations, structures can be identified when intersected by drilling. This was more the case when rotary drilling was conducted as often the bit bogged on structures. With downhole motor drilling, a vigilant driller is required to detect minor structures during drilling. There is currently no device used during drilling to recognize structures intersected. Two devices have been developed with ACARP funding that promise to allow detection of structures during drilling. BHP Research developed a computer monitor and sensors which were fitted to a ProRam rotary drill rig to record changes in drilling parameters such as RPM, rod feed pressure, penetration rate, water inflow and outflow (Danell, 1999). Underground trials showed that minor structures could be identified during drilling and an outburst occurred on one identified structure during subsequent mining. Although this rig has been available to the industry for at least two years, it has sat idle. Sigra are developing a geosteering tool which incorporates a bit torque and thrust monitor for use behind a downhole motor. Laboratory trials have shown that the sensors can detect minor changes in the strength of the materials being drilled (Gray, 1997).

Although there has been some dalliance with other predictive tools such as microseismic and detection of radon gas, they have not been advanced to usable technology status for outburst prediction.

Gas pressure and gas content

Gas pressure measurements are useful indicators of the potential for outburst. Lama, 1983 stated that the safe gas pressure at a distance of 5 m of the face was 0.6 MPa. Wood and Hanes (1982) provided detailed seam gas pressures for outburst prone and benign coal at Leichhardt Colliery. Fig. 1 summarises the relationship found between gas pressure, gas content and outburst proneness. In one location which had outburst on shotfiring, a gas pressure of 2 MPa was subsequently recorded at a depth of 2 m into the face in a hole drilled perpendicular to the cleat. Lama (1983) described gas pressure measurements at West Cliff Colliery.

![GAS PRESSURE AND CONTENT GRADIENTS]

**Fig. 1** After Wood and Hanes, 1982
His early gas investigations lead Hargraves (1965) to develop his gas emission meter to measure an index of gassiness. This was a relatively simple device which had a container for sized coal cuttings taken from a standard length (2 m) hole in the face. The gas emitted by the cuttings pushed a slug of mercury or glycol along a clear plastic tube. The amount of gas emitted during a standard time (6 minutes) was recorded as the gassiness index. The index was used successfully at Metropolitan Colliery to identify when an outburst was imminent. The meter was also used at Collinsville (Biggam and Robinson, 1980) and Leichhardt (Moore and Hanes, 1980) Collieries. At Leichhardt Colliery, the emission meter showed a positive change from very low index numbers with benign coal to very high (1.8 cc/g) numbers with outburst prone coal (Wood and Hanes, 1982). The meter required dry coal for accuracy and outburst prone coal at Leichhardt Colliery was typically wet. The use of the emission meter was frowned upon at the time because it required a delay to mining while a hole was drilled in the face and the index recorded, and the procedure was prone to abuse. However, this author has no doubts that the emission meter, when intelligently used can provide useful information about outburst proneness for little effort and expense.

In the late 1970’s the USBM technique for gas content measurement was applied in Australia. In this method, coal core samples were sealed in a canister and the emitted gas was bled off into an inverted, water filled measuring cylinder. A plot of the initial desorbed volume against the square root of time in minutes allowed a back projection to estimate the amount of gas lost from the time the sample was removed from any hydrostatic pressure in the sampling environment to the time sorption volume measurement commenced. In some cases, after completion of desorption, the coal was crushed and the residual gas volume was measured. Different coals displayed different desorption characteristics, e.g. it could take 2 months to desorb a core sample of Bulli seam whereas the Gemini seam totally desorbed in a day or two. Residual gas in the Gemini seam was minimal, whereas some samples of Bulli seam have been reported to contain up to 50% of the total gas a residual after desorption. To speed up the desorption process, measurement of desorption during crushing of the sample was introduced. This introduced some problems which were investigated by ACARP Project C6023 (Saghafi and Williams 1998) which found interlaboratory differences of up to 20% of the total reported gas content. ACARP Project C8024 is investigating these differences in an attempt to improve the reliability of desorption testing.

Lama (1995) reported on his extensive work to determine safe working gas content thresholds for the Bulli seam. He stated “The proposed threshold values based upon total gas content using coal sampling from underground are 9.4 m³/t for methane and 6.4 m³/t for carbon dioxide under conditions when mining close to geological structures and high rate advance (25m/day). These threshold values can be raised to 12 m³/t for methane and 10 m³/t for carbon dioxide when it is known that no geological structures are present within 5m of the excavation during development of headings in virgin areas. The suggested threshold values present outburst coefficients for the Bulli seam that are safer when compared with those used in many overseas countries”. These coefficients were conservatively adopted by most of the Australian mining industry as a basis for safe mining.

Sigra developed a device, under ACARP funding (Project number C3072) (Gray, 1997) to allow drilling of in-seam drainage holes while maintaining pressure within the borehole above desorption pressure. This device can be easily adapted to any drill rig and has an attachment for sampling undesorbed drill cuttings. Its use will allow rapid determination of gas content or gas pressure of cuttings and will greatly assist gas investigations and outburst management. It also promises saving of time and costs associated with routine desorption testing. The device has been ready for field testing for some time, but no colliery has offered to test it.

Williams (2000) modified the gas content threshold approach of Lama to incorporate his desorption rate index. He believes that in an outburst, once the restraining coal barrier fails or is breached, then desorption rate becomes a key factor, and it is here that the more rapid desorption of CO₂ over CH₄ takes over. GeoGAS’s Desorption Rate Index provides a different approach to setting outburst thresholds according to inherent differences in desorption rate of the subject coal compared to the benchmark coals (Appin, West Cliff Metropolitan Bulli Seam). For different coal seams and regions, the threshold 900 DRI is reached for quite contrasting gas contents (Fig. 2).
South Coast Bench Mark mines have an established history of outbursting with proven effectiveness of the gas content threshold limits

Mathematical modelling of gas data incorporating geological structure and anisotropy permeability data is now becoming available to help in the understanding of outburst mechanisms and gas management. ACARP Project C9023, Numerical Modelling of Outburst Mechanisms and the Role of Mixed Gas Desorption, conducted by Choi and Wold, is an extension of work completed in ACARP Project C6024. Current ACARP Project C9023, Numerical Modelling of Outburst Mechanisms and the Role of Mixed Gas Desorption is an extension of work completed in ACARP Project C6024. The aim of the new project is to be able to predict the volume of coal and gas that may be involved in an outburst. The likelihood and the consequence are both important considerations in risk analyses. Outbursts can be considered to be caused by the interaction between fluid pressure and rock deformation, leading to the failure of the rock while enough energy is still trapped in sufficient volume of free gas under high enough pressure to cause an outburst. The latter (gas pressure) is related to the gas content (used in the threshold) and gas drainage prior to mining. The modelling requires input of reliable field measurements of gas pressure, rock stress etc.

PREVENTION

Hargraves (1980) summarised outburst preventative measures under the following categories:

- Mining method and geometry – avoidance of seam entries, use of longwall advance, and avoidance of leaving coal in the roof;
- Seam destressing – The use of large diameter holes and high pressure water infusion; and
- Seam gas pre-drainage.

Mining method and geometry

Australian coal mining appears locked into bord and pillar or longwall mining. Advancing longwall has not been favoured.

Seam destressing

Hargraves and others tested the drilling of large diameter holes into the face to reduce stress and gas concentrations at Metropolitan, Corrimal, Leichhardt and Collinsville Collieries. Hargraves (1983) reported that at Corrimal Colliery, drilling of 300mm diameter holes through structured coal was accompanied by stress release manifestations. Advance holes were drilled until drilling no longer produced stress release manifestations. He reported that “sometimes during the boring of routine advance relaxation holes of 300mm diameter and length up to 80m (using scrolls for cutting return) at Metropolitan Colliery some stress manifestations occurred, at least collapse
of the hole as with Corrimal... but perhaps a minor outburst in the hole with surges of gas emission from the hole. Further advance of the bit would stall the drill and the operators learned to continue rotation without feed until cuttings return ceased before restoring feed to the drill". He reported also "the effective diameter of de-stressing holes has been determined by progressive increase of diameter until the cuttings produced exceeded the volume of the hole".

At Leichhardt Colliery, 300mm diameter holes were tried with little apparent success. There was no obvious deformation of the holes and no obvious gas emission from the holes, but bumping noises were reported during drilling. The drilling of 100mm holes in a pattern of two rows of five holes each around 28m long was adopted for outburst prevention. The holes were drilled on a back shift and mining usually commenced the following day. Numerous outbursts occurred in predrilled coal. Where mining closely followed drilling, outbursts typically occurred. Long standing times after drilling reduced the frequency and size of outbursts. This was reported by Wood and Hanes (1982) summarising events leading up to the fatal outburst of December 1978. "In A North Intakes, the heading in which the major 500t outburst of 1/12178 occurred, 100mm relief holes produced a positive effect. The face was drilled in June 1978 with ten 100mm holes each about 28m long and left to stand until 7th November. The advance of the section covered by the holes was under good conditions and rate of advance increased. Mining induced cleavage was absent until near the end of the holes and the ribs tended to spall rather than be 'hard', i.e. pick marks over their full height. Near the end of the holes the ribs 'hardened' and some mining induced cleavage occurred. A further five holes were then drilled and mining recommenced. Three bursts occurred in the following three days with drivage over 12 m. The bursts ranged in size from 50t to 500t. It is concluded that the holes, drilled and allowed to stand for four months were successful in preventing outbursts and in minimising outburst related strain".

Lama (1983) described the effectiveness of large diameter advance holes to drain a fault at West Cliff Colliery in 1976. Three 100mm diameter holes were drilled in each of 4 headings to intersect a mylonite zone. “Violent ejections of pulverized coal occurred at intervals of almost every 10 minutes in the initial stages and continued for almost 24 hours with the material thrown out to a distance of 20m...Further driving of the heading after an elapse of 140 hours produced no violent outbursts...”.

Ward (1980) described the use of pulsed infusion shotfiring as an outburst prevention technique. Three holes, each 6m long were drilled in the face, infused with water and shotfired. He reported, “With the introduction of pulsed infusion shotfiring, the need for multi-entry development panels was reduced due to the quicker degassing and destressing effect on the coal. Working conditions were improved also as roadways could be driven better on line and productivity increased due to lesser man hours used than in boring large holes and less time was lost in flitting machines”.

Seam gas predrainage

Hargraves (1980) wrote “To prevent a gas phenomenon in the course of mining, it is necessary to degas the zone to be mined sufficiently to reduce the gas content below the content for proneness. Outbursting coals are almost invariably high rank which involves high sorptive capacity and low permeability, making degassing of virgin coal more difficult”.

A full-scale drainage program was commenced at West Cliff Colliery in 1980 (Lama, 1983). Kelly (1983) described the effectiveness of pre-drainage commenced at Appin Colliery in 1981. All holes were drilled with a Kempe U4-450 rotary drill rig; 50 % drainage was achieved after 60 days, 70% after 100 days and 90% after 240 days drainage.

In-seam predrainage of the gas is routinely successfully applied at Appin, Tower, West Cliff and Tahmoor Collieries in the Illawarra, at Dartbrook Colliery in the Hunter Valley and at Cental, Southern and North Goonyella Collieries in Central Queensland. It has dramatically reduced the number of outbursts in the Illawarra collieries, but learning has come at a price.

Predrainage was initially conducted with rotary drilling. In 1993, the industry drilled 300 km of rotary drilling and 160 km of guided drilling. In 2000, around 300 km of guided drilling and 50km of rotary drilling are conducted. It was accepted that rotary-drilled holes deviated from their initial trajectory, but it was assumed that they deviated consistently to align sub-parallel to the dominant cleat. Following a fatal outburst in 1994 from a face at West Cliff Colliery which was “protected” by 4 rotary holes, many rotary-drilled holes at the various mines were surveyed and found to be very inconsistent in deviation. Rotary drilling was rapidly dropped as a means of gas drainage ahead of development panels in outburst-prone coal in the Illawarra. Lama and Bodziony (1996) stated that the outburst occurred in coal with 18 m³/t mainly (98%) CO₂. The outburst occurred on a strike-slip fault which had been
previously intersected in another heading 45m away without incident. Four gas drainage holes were rotary drilled from the face 60m outbye the eventual outburst site. Each of the holes had deviated to the left and at the outburst site, they were on the left rib line, one above another. The strike-slip fault zone, projected from the previous heading, was intersected safely on the left rib, but the deputy implemented outburst-mining procedures and removed all personnel from the face except the miner driver who was in an enclosed cab. An unexpected second strike-slip fault occurred inbye the first and when cutting recommenced on the right hand side of the face, about 200t of coal was ejected. The miner driver was killed. Lama and Bodziony (1996) state “This outburst resulted in development of comprehensive outburst management procedures which form the basis of mining of the Bulli seam”.

Since the 1994 fatality, essentially all drilling for gas drainage in the major gassy mines of New South Wales and Queensland has been conducted using downhole motor and survey tools so that the locations of the holes are known within the accepted accuracy envelope of long hole drilling. Test cores are taken, usually in the “worst possible location” with respect to gas contents, to test drainage effectiveness prior to mining.

Predrainage has, in general, nearly eliminated the threat of outbursts in today’s collieries. The exceptions are represented by Tahmoor, Appin, West Cliff, Metropolitan and North Goonyella Collieries (to date) in poorly defined zones of low permeability. These zones, in some cases adjacent to geological structures and in other cases, best represented by Tahmoor Colliery, with little to no association with geological structure. In some cases, they can be drilled, but refuse to give up their gas. In other cases, such as at Tower Colliery, where they cannot be drilled, they become a drilling equipment graveyard. ACARP are funding a research project in 2001 to try to understand the nature of these low permeability zones.

**CONTROL**

Control of outbursts has mainly centred on controlling the time at which an outburst will occur so that no personnel are endangered during the outburst. Hargraves (1980) described the primary control method as “by inducer shotfiring, devised by Marsault in France in 1892. It involves a simultaneous round of shots to advance the entire heading face. by over-boring, over-charging and firing from a safe or remote distance... The number of inducements is significantly greater than the number of outbursts which would occur spontaneously”.

Simultaneous shotfiring was used at Leichhardt Colliery in 1974 to induce outbursts during drivage of around a couple of hundred metres. The drivage was in an area that experienced daily outbursts when driven by continuous miner (without predrainage). Ward (1980) described the use of mining by full-face shotfiring with millisecond delay detonators for roadway development at Metropolitan Colliery. He stated that many outbursts were induced. Phillips, 2000 reported that in recent work at Metropolitan Colliery, virgin gas content is 13 m³/t and in one area, the gas content remained at virgin level even after considerable drainage time. There was no obvious structural cause for the low permeability. The area was mined by grunching in 1998. Grunching in recent times has not induced any outbursts at Metropolitan. Shotfiring or grunching was also used at Tahmoor Colliery in 2000 to mine an impermeable zone which could not be drained to below the gas threshold for safe mining (Wynn, 2000).

Wynn and Case (1995) described the application of an ABM20 miner to remote mining of hazardous coal. They stated “Essentially all that was required was a flameproof video camera, a flameproof video monitor, a communications system to link the two and a system to link the radio control to the miner...”. A suspected fault was previously mined in the left hand heading using the encapsulated 12CM miner and outburst mining procedures. As suspected, a significant outburst occurred on the fault. The miner driver was adequately protected by the system but clearly of the opinion that it was not an experience he would knowingly go through again... The ABM was then advanced in the right hand heading towards the fault...Normal outburst rates of 2m/shift were achieved. When the fault was reached and the machine started to cut up through it, a major (80t +) outburst occurred. This was witnessed on the monitor by the crew in the fresh air base. We believe this was the world’s first televised outburst”. ACARP Project C3035 developed a separately ventilated control room for use in mining of hazardous zones.

Benson (2000) provided information on the successful mining of a gassy impermeable zone at Tower Colliery by remote mining. Tower Colliery experienced an outburst on Saturday morning 9/12/00 whilst remote mining in MG19. The event occurred on a thrust fault inbye of the dyke which had been mined through. The gas content was 13m³/t. “An outburst occurred in a controlled environment. No person was in the area at the time and no injuries were sustained and all correct procedures were followed. Remote mining was in progress in the maingate 19 panel B heading. This obligated the coal cutting to be performed from a remote operating station (ROS) located approximately 275m outbye the face area. No operators were allowed inbye this remote operating station or in the
return airway whilst the cutting process was in progress. The area the outburst occurred was inbye of a 1m dyke and the coal was in virgin gas state due to an inability to drill and drain the methane below the 9m/c threshold level. The procedure developed for this process was being followed strictly. The procedure was developed as an outcome of an extensive risk analysis process. The miner operator during the shift experienced two power losses at the miner due to gas tripping off the miner. No significant signs were observed during the face inspections, carried out by both the deputy and miner driver, after the first power loss. When the face was inspected after the second power loss, the deputy observed a cavity in the right hand side on the corner of the face and rib”.

CONCLUSIONS

Hargraves (1980) stated “The tendency to regard the maximum size of previous outbursts as the maximum size of future outbursts is an attitude fraught with danger…even outbursts regarded as small may involve hazards seemingly out of proportion to their size”. At Leichhardt Colliery, most outbursts induced by the continuous miner produced 1 to around 50t of coal. There was a tendency to think of these outbursts as the norm. The fatal outburst was 500t. There have been tendencies in the industry in the past and presently to refuse to recognize the early signs as outburst-related phenomena. Many names have been used to describe these phenomena including “dynamic gas phenomena, puffers, blowers, etc”. The early signs should be recognized as warnings and thorough investigations of virgin gas content and pressure should be conducted in conditions as conducive to outbursting as possible. The occurrences of outburst related phenomena and their investigations should be thoroughly documented, and preferably published to increase communication on outbursts throughout the industry. Without documentation, much valuable information is lost. There is also a need to involve researchers and industry consultants in the investigation and documentation of outbursts and gas phenomena to maintain some continuity in the development of outburst knowledge. In recent years, the industry has lost the active participation of two great outburst researchers, Dr Alan Hargraves and Dr Ripu Lama, who advanced the understanding of outbursts in Australia to new levels. Their publications represent a great legacy to the industry, but unless their examples of promoting outburst research and knowledge are followed, more outbursts will occur when not expected.

Fatalities from outbursts seem to have occurred when the industry thought it had outbursts under control. Each fatality has spurred the industry to improve outburst management and control. This learning cycle has to be broken. The industry cannot afford to lose more lives because it thinks outbursts are under control. There is a need to continually review and update the understanding of outburst mechanisms and to question safety procedures regarding outburst management. How do we know that the gas has been uniformly drained from the coal surrounding the entire length of a drainage hole? In branched holes, how do we know that all branches have drained effectively? Can we be certain that a drainage hole has not crossed an impermeable zone which has not drained? What causes impermeable zones? Can all personnel recognize when mining or geological conditions change from normal? There are many other questions which remain unanswered and there are probably questions not asked.

Understanding of outbursts and development of technology to investigate and prevent outbursts advanced during the second half of the 20th Century and must continue to advance during the 21st Century.

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NEW LEGISLATIVE FRAMEWORK AND THE ROLE OF GEOTECHNICAL AND GAS DRAINAGE PROFESSIONALS

BY IAN ANDERSON
NEW LEGISLATIVE FRAMEWORK AND THE ROLE OF GEOTECHNICAL AND GAS DRAINAGE PROFESSIONALS

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SYNOPSIS: In recent times legislators have moved away from prescriptive legislation to duty of care models. Such a move places employers and consultants under a need to demonstrate due diligence in the exercise of their functions. The principle legislation under which the coal industry operates is the New South Wales Occupational Health and Safety Act. This Act has substantial punitive measures including fines and imprisonment.

The Department of Mineral Resources has developed and implemented an Enforcement policy which includes a prosecution element.

These changes necessarily impact upon key industry personnel including:

i) Mine Managers
ii) Internal Consultants - Company Engineers
iii) External Consultants

This presentation considers the impacts that are likely to affect the above group and discusses behavioural and procedural measures that will be needed to address their obligations under duty of care legislation.

Consultants, both internal and external, will need to advise their clients of the potential negative aspects of any advice offered and ensure that their clients comprehend this level of risk and its associated consequences. Effort will need to be made to ensure that all limitations of numerical and/or empirical models used in analyses are articulated.

Mine managers will need to be more discriminating in whom they commission for advice. Mine managers will need to be more challenging of the basis and limitations of advice offered to them.

Australian and overseas case studies are the basis for the presentation.
HOW NSW AND QUEENSLAND COALFIELDS DIFFER – WHAT WE NEED TO DO BETTER

BY BRIAN NICHOLLS
ABSTRACT: There are obviously similarities and differences between NSW and Queensland underground coal mines. The major differences are related to rock strengths, stress regimes, location and operational approaches. Issues relating to these factors are addressed, with some suggestions as to how underground coal miners can do some things better to improve efficiencies and returns on investment.

The differences are many and varied. Some factors having a major bearing on mining efficiencies are discussed, with some examples of failures and successes being outlined.

INTRODUCTION

The subject of differences between the two major coal mining areas on the East coast of Australia is somewhat substantial. The elaboration of differences can be extremely detailed or can be outlined in general terms.

It is not intended (in this paper) to relate to detailed matters, but to outline the author’s opinions formed after a number of years operating underground mines in NSW and both open-cut and underground mines in Queensland. No comparison will be made of open-cut operations. This paper only refers to underground longwall mines.

The paper will briefly outline the major geological, geotechnical and operational differences between the major coalfields. Specific reference and comparisons will be made relating to longwall mining operations.

LOCATIONS AND PRODUCTION 1999/2000

The major operations are spread across all the NSW Sydney basin and the full length of the Queensland Bowen Basin. A major difference geographically, is the substantial distance between groups of mines in the Bowen Basin.

Almost all underground coal mines are now longwall operations. A number of mines continue to operate either partial or pillar extraction systems. This comment applies to both coalfields, but there are very few non-longwall mines in Queensland (Cook and Laleham).

Total underground coal production from both states for financial year 1999/2000 was 90,938,000 ROM tonnes, (NSW 52,763,000 tonnes; Queensland 38,175,000 tonnes). Longwall coal produced during the same period was 71,221,900 ROM tonnes.

BASIC DIFFERENCES – AN OPERATOR’S PERSPECTIVE

Geological

The major differences directly affecting operations and most supporting activities revolve about rock strength factors. Rock strengths generally overlying NSW coal seams are considerably greater (more competent) than those immediately overlying the Bowen Basin coals, particularly the thicker Northern Bowen Basin seams. Rock strengths in the Goonyella measures are as low as 10 MPa in some locations bringing about specific support requirements. The coal itself is sometimes the strongest member in the stratigraphical section. Compared with
NSW, particularly the Illawarra (Bulli Seam) and Newcastle (Great Northern), the operational support requirements can be much more onerous. Coupled with complex faulting and water bearing strata, operational issues can be somewhat daunting. Conversely, the massive and strong rocks surrounding both Illawarra and Newcastle district coals also bring about their own specific issues relating to ground stability.

In my experience, the nearest comparable mining environment to the Illawarra is found in the German Creek measures at Capcoal and Oaky Creek. This comment does not apply to the shallow open-cut and sub-crop measures at both locations where overlying rock strengths can be very low. This was highlighted recently with the Oaky North longwall issue. Generally, the faulting associated with NSW measures is relatively “normal” with magnitudes of throw varying, but not usually over short distances. The faulting is much less frequent compared to the high density complex faulting found in the Northern Bowen Basin. Very close grid drilling is required to determine faulting, particularly in the Goonyella, Wrangles and Newlands measures. Faulting in these measures can vary very considerably over very short distances.

The complication of frequent reverse thrust faults creates further hazards during development but more particularly on the longwall faces. Where reverse thrust faulting occurs, it is associated with an obvious increase in seam thickness and also with considerable areas of shattered coal and roof strata. Mining through these structures can be extremely difficult and costly, particularly if the problem is not contained. Very major roof falls have resulted from this problem.

These face profiles, used to enable more effective management of the longwall face by both supervisors and shearer drivers, show the rapidity of change experienced on Goonyella middle seam coal faces. Failure to get the face control right can result in massive face falls and extensive downtimes. Recent examples of this are North Goonyella (3 months) and Kenmare (6 months and resulting in the need to pull off the face equipment.)

Sudden, undetected major reverse faulting within a longwall block, if not detected before installation, can result in the longwall being shortened and equipment being relocated (Newlands). From these examples, it can be seen that in the more geologically disturbed soft rock measures worked in the Queensland Bowen Basin, intense geological exploration, detailed mapping and precise interpretation is essential to minimise the risk of interruptions to the business.

The obvious advantages enjoyed by most of the Queensland mines are generally shallower deposits (at present) and easy access (during the dry season) to surface drill sites for exploration and mine servicing. Location of the mines, surface infrastructure (suburban and rural development) and substantially deeper deposits precludes most NSW mines from this intensity of exploration drilling from the surface. Hence, other inseam techniques which are usually more expensive and less accurate have to be used in the deeper NSW mining operations.

Both coalfields have varying insitu gas levels with their associated problems. Frequent faulting in thick seams with fairly high insitu gas content is making gas extraction essential at some Bowen Basin mines. Separating gas extraction and utilisation from the mining process is becoming a major issue requiring a re-think of how to achieve this objective. More research is needed into surface gas well techniques of extraction where shallow but gassy measures are being mined.

The mine gas problems are, in reality, not very different between the two states. How the problems are solved may be different in each location. NSW mines have the CO2 problems while some Queensland mines have a H2S problem. Both environments must have very precise and detailed hazard management plans in place to effectively mine in these conditions.

Spontaneous combustion is an issue in the Bowen Basin, particularly in the thick seams where considerable volumes of coal are left in the goaf after mining. Once again, very specific detailed management plans must be in place and understood by management and workforce to enable effective mining in these conditions. A combination of high insitu methane levels and a liability to spontaneous combustion is not a desirable combination.

Geotechnical Issues

The stress regimes in both coalfields vary considerably, region by region. Generally, the horizontal stress levels experienced in the Bowen Basin are of less magnitude than in the Illawarra. A reduced horizontal stress component in a soft rock – thick seam environment can, however, bring with it substantial mining problems, particularly
around longwall gate roads. Following a number of major falls in gate roads recently, accurate and more frequent
determination of *insitu* stress levels and direction have become essential to more effectively predict potential
influences on the longwall operations.

Frequent faulting, *insitu* stress levels and mining induced stresses have all to be considered when determining strata
control requirements. Failure to adequately determine their collective impact will reduce operational efficiencies.

High quality coals with thick and shallow seams are not necessarily the longwall miner’s dream. They can become
the longwall miner’s nightmare if not predicted, planned for, and accommodated with detailed mining engineering
procedures.

**SO HOW DO WE DEAL WITH THESE ISSUES?**

The answer lies with a substantial collaborative effort in geological, geotechnical and mining engineering before
defining mining plans.

As previously stated, the shallower measures being mined in the Bowen Basin lend themselves to serious,
concentrated surface exploration of areas of coal down to the definition of individual longwall block structures.
Failure to dedicate resources and money to these requirements will put high volume longwalling at risk.

With the cost of unplanned downtime on longwalls now running at +$38,000/h, it is critical that as much
geotechnical knowledge of defined areas of extractable coal is gained before mining. To this end, detailed
geological and geotechnical plans must be of high priority to the longwall miner. This can only be achieved in both
coalfields with site specific and dedicated geological/geotechnical expertise. Whether that is by in-house or
external expertise is irrelevant, providing the information is available before mining commences and that detailed
monitoring takes place during the mining process. Continual up-dating of face and gate road data is essential.

Over 20 years ago, this type of assistance was put in place at some mines in the Illawarra. These requirements are
only now being recognised and accepted by some operators in the Bowen Basin, and only after some major
longwall strata problems. The provision of daily face profile data, horizon planning and gate end preparation by
secondary support will reduce the operational problems substantially.

The development of strata hazard management plans is both an inspectorial and an operating management
requirement. It is also good mining practice. Any operation not now having this type of management process will
not optimise the effectiveness of its production nor will it eliminate the source of major injuries and fatalities. The
important point is that this must be a “tool to work with”.

Effective geological and geotechnical management is essential in both coal fields. It is unfortunate that the
acceptance of this sort of management tool has been slow to arrive in some parts of the Bowen Basin.

As previously stated, one specific geological difference between the two coalfields is the frequency of faulting,
particularly the reverse thrust faulting in the Northern Bowen Basin. The geotechnical requirements for adequate
support of these types of structures on longwall faces or gate roads have resulted in substantial development of
active roof tendon support systems and strata stabilisation injection techniques. The strata stabilisation systems
now available through various organisations have basically resulted from “after the event” experiences in
recovering falls. By effective determination of seam structures, injection techniques can be applied before the
event (that is mining) instead of after the event (a major roof fall) has occurred.

The designs of injection systems and improvements in various stabilisation products are now available to assist in
minimising the effect of geotechnical issues caused by geological structures within mining areas. Drilling and
injection expertise is available from several companies. We need to involve them early, rather than later in the
mining process.
OPERATIONAL ISSUES

The issues here can be categorised as:
- Industrial
- Management structures

Industrial

The major difference between NSW and Queensland lies with the power given to front line supervision workers (deputies) within the Queensland legislation. The way the law is drafted in Queensland gives the deputy (who is predominantly in the C.F.M.E.U. in Queensland) considerable supervisory and industrial power. This is not the case in NSW, with some deputies being members of other industrial organisations. The effect of this type of structure is the frustration of plans when attempting operational improvements and management structure changes. When this is combined with the mining town or fly in – fly out syndrome where the employees live together almost continuously, then major constraints can be applied to the attempts of management to improve systems and management control.

This is not to say that all operations in Queensland are faced with this type of problem. There are indeed many good front line supervisors, but the fact is that I believe they could be better if the industrial reins were removed. This has, in fact, been proven at some locations. Until the law is changed to remove this power from certain categories, industrial progress will be more difficult to achieve. Accountability of individuals will be more difficult and performance appraisal systems will be harder to implement.

The NSW operations are not, in my experience, constrained to the same degree at supervisor level. This industrial environment makes it easier to bring front line managers into the management team.

In both states, there is nothing to stop operators from designing a structure to most effectively manage their production and industrial issues. They just have to have the determination to do things differently. Certified agreements, like longwall mining, can be beneficial or a burden. It depends on the wording of those agreements.

In Queensland there are some very good agreements, but there are also some that give a great deal of power to the employees. This power can be used to frustrate management in its efforts to improve operational efficiencies. At the end of the day, the employees have no direct responsibility for the fiscal health of the business nor are they ultimately accountable for the safety record of the mine.

Management Structures

With the exception of the deputy position as discussed above, management structures have, until recently, been similar in both states. The traditional mine management structure has prevailed to accommodate statutory requirements. The introduction of more flexible working arrangements brought about in part by new industrial agreements following changes to the Federal industrial relations legislation (Workplace Relations Act 1997), different management structures have been implemented in both states. It is becoming more common to have a supervisory structure designed to suit business needs and not to accommodate mining legislation as its main objective. The opportunities provided here are many and varied and managements who have grasped these opportunities are beginning to reap the benefits in improved business results.

The main resistance to these changes has come from middle management who seems to perceive a threat to their security instead of a challenge to stretch their capabilities. Being removed from their comfort zone is not accepted easily.

It is not practical, in this paper, to compare structures, but one area for comment is the longwall management structure. Reference has been made to front line supervision. With a business capable of generating in excess of $500,000 per day in revenue, a strong management structure has to be in place. Gone are the days when we can leave a longwall to a frontline supervisor and crew. We must now have a full technical backup and shift supervision group in the management team which is accountable for the longwall business.

In a recent paper (Nicholls, 2001), I espoused the requirement for the “face boss” position to be implemented on longwall faces. The benefits from having the right people in this position can be enormous. This, I believe, is the area most lacking in both coalfields. There are many longwall deputies who could fill this role, given the
opportunity. Neither coalfield can afford to ignore these structural requirements. This paper has not discussed the
differences in longwall management structures in the two coalfields but has highlighted what I believe is a common
shortcoming in the way we manage our longwalls.

Procedures associated with different longwalls are obviously determined by capacity, volume requirements and
industrial issues. All longwalls should be set utilisation criteria for which the management team is held
accountable. Increased utilisation is the key to improved efficiencies and productivity. We do not focus enough on
this particular aspect of the business. Again, this is not a comparison, but it is a common problem.

WHAT WE NEED TO DO BETTER

From the comments made in this paper, which are by no means a significant comparison or critique of the NSW
and Qld. Longwall mining operations, the following are just some of the suggestions I make for doing things better.
Substantial effort must be put into geological and geotechnical advice and data collection before, during and after
mining.

- Geological and geotechnical expertise must be attached to each longwall operation. They must be part of
  the management team which must be accountable for longwall results.
- Define and delineate any areas of abnormality predicted before mining following geological/geotechnical
  investigations and previous experience. Establish plans to deal with these predicted issues. Sign off and
  inform the operating supervisors and crews of the plans. Make sure that they are regularly updated on the
  plans and their responsibilities to the plan.
- Involve the crews in the reasoning behind the plans, seek their contributions – there is a wealth of
  knowledge out there.
- Put in place a management structure to drive the face at its optimum rate.
- The industry needs to treat each longwall as an independent business. Install the management structure
  with accountability to match. Use key indices on a regular basis comparing to the actual plan.
- Ensure that management, supervisors and operators in the key longwall businesses are the best you have
  available. Train them well and keep them focussed on continually improving their performance.
- Bring in external expertise if the business is not performing to its optimum level. Asking for help is not a
  deficiency – not asking for help is!

In summary, this has, of necessity, been a generalisation of some differences and some similarities within the NSW
and Queensland coalfields. There are many other issues which could have been discussed within this topic. Some
suggestions have been made which could be applied in both coalfields. It has been my experience that applying
these principles will improve the overall performance of the longwall mining operations in both states.

REFERENCE

Nicholls, B, 2001. What’s wrong with Australia Longwalls?, McCluskey’s Coal Forecast 2001 Conference,
Brisbane, Qld, Nov. 2001.
Fig. 1 The Coalfields of NSW

Fig. 2 Queensland Bowen Basin – Longwall Capacity
Fig. 3 Production from Longwall Faces Australia

Fig. 4 Coal Resource
Fig. 5 Coal Reserve Classification

Fig. 6 Longwall Face Survey (07.05.00 at 7:00am)
Fig. 7 Longwall Face Survey (10.05.00 at 12:30pm)
GAS DRAINAGE PRACTICES

BY LW LUNARZEWSKI
GAS DRAINAGE PRACTICES

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ABSTRACT: Coal seam gas problems, largely “gas-outs” and instantaneous outbursts of coal and gas, have created serious difficulties for the coal mining industry around the world. Typically, a single longwall face is now capable of producing an average 10 000 to 15 000 tonnes of coal per day. The total quantity of gas released in gassy mines could conceivably reach 5000 and 8000 litres of gas per second per single longwall block and for the total mine respectively.

The introduction of various gas drainage techniques in Australian gassy mines was necessary to complement ventilation systems and to satisfy the statutory gas limitation in underground workings. Methane drainage can simultaneously reduce the risk of dangerous methane concentrations accumulating as well as reducing methane emissions into the atmosphere; moreover, the methane recovered is a valuable energy source and can be used to considerable economic effect. The paper reviews Australian achievement in gas capture technologies, gas drainage

In many cases the quantity of gas emitting into coal mine workings is so high that the available quantity of air in the ventilation system is insufficient to dilute the gas to acceptable levels. In such a situation, other methods in addition to the ventilation system should be planned and introduced during different phases of coal mining activities.

The most effective additional system is the introduction of gas drainage (recovery) techniques. Seam gas recovery from underground mines is the capture of seam gas from strata or underground workings for the purpose of reducing gas flow into the mine ventilation system, controlling gas hazards, and utilisation of the gas, if applicable. This is carried out for safety purposes to avoid fatalities or injury, and to reduce the loss and delay of coal production and development drivage rates.
The classification of gas drainage methods is based on the phase during which degasification is performed in relation to the coal extraction. The six basic methods used are:

- pre-drainage by vertical or directional holes drilled from the surface,
- pre-drainage by horizontal longholes drilled from development headings,
- pre-drainage by in-seam headings,
- post-drainage of relaxed strata using cross-measure holes drilled into the overlying and underlying gas sources,
- post-drainage by holes drilled from the surface, and
- post-drainage of old or active goaf areas from underground.

Fig. 1 shows the logical and necessary steps required to design gas drainage systems for underground coal mine safety and gas utilisation purposes, taking into account the mine’s or longwall predicted gassiness.

**REVIEW OF COAL MINE GAS DRAINAGE PRACTICES IN AUSTRALIA**

This section reviews the practices of gas recovery in selected Australian underground coal mines. In 2000, Australian black coal mines produced 240 Mt of saleable coal, of which 80 Mt came from underground mines, almost 88 percent of this from longwall units.

The maximum and average depths of underground mining are 550 and 280 m, respectively. The *insitu* gas content of coal seams in deeper operations between is 5 and 20 m$^3$/t *insitu*, however, other gassy mines have experienced gas content levels between 2 and 5 m$^3$/t *insitu*. Specific gas emissions in the gassy mines range from 5 to 90 m$^3$/t of mined coal.

Full scale gas drainage is relatively new to Australia (Fig. 2), although some initial attempts were carried out much earlier; 1897 & 1945 - Balmain Colliery, 1925 & 1954 - Metropolitan Colliery, 1954 - Collinsville Colliery, 1965 - Hepburn Collieries, and 1970 - Appin Colliery (Hargraves & Lunarzewski, 1985).

Full-scale pre and post drainage of gas started in 1980 at West Cliff and Appin Collieries. Gas drainage is currently practised in fifteen (15) mines, most of which use the longwall mining method. Outbursts of gas and coal have been experienced in a number of mines. These occurrences have given impetus to the development of methods for draining gas ahead of mining to reduce *insitu* gas content and pressure. Both gas recovery methods - pre and post drainage are currently used in Australian gassy mines.
Pre-drainage

The term pre-drainage refers to a particular gas drainage technique. It deals with coal seam gas drainage from selected gas sources, prior to coal extraction, under conditions determined by their \textit{insitu} parameters. The ability of the drainage system to capture gas in the pre-drainage phase depends substantially on the permeability of the coal seams and adjacent strata, their gas migration properties, connectivity, conductivity, and the provision of sufficient lead-time.

The method to supplement the ventilation system is to install horizontal holes, a technique that has been used in coal mining since the 1800s. This technique consists of drilling holes from the mine workings into the unmined areas of the working coal seam (Fig. 2). These holes are typically tens of metres to hundreds of metres in length, and within a single mine several hundred holes may be drilled. The horizontal holes are connected to an in-mine piping system, which transports all of the methane released into the hole out of the mine. By draining methane from the unmined coal, horizontal holes reduce methane emissions into the mine workings and during mining. In some cases, 30 to 50 percent of the methane contained in the coal seam being mined may be removed with inseam pre-drainage holes.

Inseam pre-drainage holes are drilled as:

- fan-shaped single holes drilled from an adjacent double entry heading panel (Figs. 2 & 6)
- fan-shaped branched
- long holes parallel to a heading (Fig. 6)
- directional holes to the adjacent seams in the roof and/or floor drilled from underground or surface (Fig. 3)

The spacing and length of holes is dependent upon the permeability of the coal and the pre-drainage lead-time. Underground hole lengths of 300m are common, but long-directional holes up to 1800m in length have also been successfully drilled. Holes drilled from the adjacent panel are used to drain gas from longwall blocks as well as the next set of gate roads, in advance of mining using suction or very occasionally positive \textit{insitu} pressure. The spacing between such holes, when drilled parallel to each other, varies from 8 to 100m. Combinations of fan-shaped parallel and lateral holes are used in high gassy areas when only a short lead-time is available. All inseam holes are fitted with standpipes of lengths varying from 3 to 9 m and hole diameter of 96 mm. In high gas emission areas, drilling is done through the standpipe. In areas where gas flows are low, holes are drilled first and the standpipes are installed before applying suction. Some holes are cased with steel or PVC – protected perforated pipes.

\textit{Directional Drilling Techniques}

Considerable success has been achieved with inseam longholes drilling (Fig. 6) using down-hole drill motors and hole trajectory control techniques and equipment such as Directional Drill Monitor utilising Modular Electrically-Connected Cable Assembly (DDM-MECCA) & Drill Guidance System (DGS) survey instruments. The system provides rapid and easy underground hole survey measurements whilst drilling, including computer monitoring if required. It measures the earth's magnetic field and gravity in all three directions (x, y & z) with borehole placement accuracy of $\pm 0.1$ inclination and $\pm 0.5$ degrees azimuth. The instruments and connections are intrinsically safe, which allows for their application in underground gassy coal mines as well as fast and reliable data transmission irrespective of hole depth.
The first directional (guided) in seam longhole was drilled in Australia at Appin Colliery in 1987 to drain gas from the adjacent coal seam located 18m below the working seam. Guided longholes are used for pre-drainage of longwall panels and post-drainage of goafs, with the main part of drilling being in the targeted seam. This method is a proposed replacement for those previously used in Europe ‘Bleeder roadways - Hirschbach system; however, more research and practical applications are necessary to establish hole stability protection and lead time for various geological, mining and gas conditions.

**Water-jet cutting technique**

Recently a new drilling method for the accurate and efficient installation of long in seam boreholes has been tested in Australia. This involves the integration of pure water-jet drilling technology with conventional directional drilling technique. In effect, the system is similar to conventional directional drilling methods, but instead of relying on a down-hole motor rotating a mechanical drill bit for cutting, a pure high-pressure water-jet cutting technique is used.

**Hydro-fracturing technique**

Some surface vertical holes using the hydrofracturing technique were introduced in New South Wales and Queensland, however, majority of the experimental zones have not produced a gas flow rate of industrial significance. The results are not as promising as in the USA, this is mainly due to the different geology, coal seam characteristics (internal structure, connectivity and conductivity), insitu gas content, pressure and saturation as well as gas sources and surrounding strata’s low insitu permeability.

**Post-drainage**

The term post-drainage refers to another gas drainage technique concerned with the relaxed strata during and after coal extraction (Figs. 2, 3, 4, 5 & 6). In post-drainage methods, advantage is taken of the phenomenon of increased strata permeability and connectivity-conductivity due to stress relaxation of the roof and floor coal seams and adjacent rocks, which occurs as a consequence of coal extraction and/or other mining activities in underground mines.

While horizontal holes can degasify the target coal seam, they cannot effectively degasify the overlying or underlying coal and rock strata. To accomplish this type of degasification, cross-measure holes are used. This technology consists of drilling holes from the mine workings into unmined areas of the coal seam and surrounding rock. Cross-measure holes, angled into the rock and coal strata above and below the mine workings, are used to recover methane from the relaxed strata sources and goaf areas. These holes are typically tens to hundreds of metres in length. The holes are connected to an in-mine vacuum piping system, and the recovered methane is transported out of the mine.
Post-drainage holes are drilled as:

- underground cross-measure up holes (Figs. 2 & 4),
- underground cross-measure down holes (Fig. 5), and
- gas wells - goaf & strata free gas drainage holes drilled from the surface (Fig. 2).

![Diagram of drainage holes and strata relaxation](image)

**Fig. 4** Roofgas© version 3.0 output using rock strata curvature enforcement model - vertical cross section of half of the longwall with strata relaxation & gas release zones for 250 m distance behind the face

![Diagram of longwall and drainage holes](image)

**Fig. 5** ‘Floorgas©’ output - vertical cross section of half of the longwall with single entry heading, strata relaxation and gas release shape and vertical stresses (MPa) as well as projected down holes

Post drainage hole diameters vary depending on the drilling technology used. Shorter length rotary holes have diameters of 65 mm to 75 mm. Longer holes drilled using downhole motors are 96 mm in diameter or larger.
Slotted steel casings are sometimes inserted along the length of a hole to maintain its stability. Gas drainage efficiencies typically vary from 30 to 80% for the districts and 25 to 60% for the total mine. The highest efficiency for an individual longwall was achieved for the ‘protected’ by pillar holes when drilling from double entry panel second heading, cut-through or stub as well as for perforated-cased holes. The diameter of surface holes is 150 to 300 mm, which depends on working seam depth, gas and mining conditions as well as source of gas transportation (suction or free flow - buoyancy effect). Hole position in relation to the strata relaxation zones shape and longwall goaf geometry is essential, and is designed using computer simulation for specific local geology and mining conditions.

![Diagram of gas drainage holes in a longwall system](image)

**Fig. 6. An example of in seam and cross-measure gas drainage holes in longwall system**

*Water difficulty*

Significant restriction for gas recovery from drainage holes could be caused by *insitu* and ‘drilling’ water, especially with cross measure down holes. A special technique has been introduced in Australia using PVC internal conduit for a self-dewatering system in each individual hole. Also, automatic gas-water separators, which do not require external power, are used for individual and grouped holes, as well as for horizontal and vertical gas drainage pipelines.

**GOAF DRAINAGE**

The term “goaf drainage system” refers to an independent gas drainage technique aimed at capturing a high percentage of gas from active or sealed goaf areas from both underground and/or surface.

The longwall extracted areas can be efficiently sealed off after completion of the extraction process. The exposed cavities and/or goaf areas form ‘free gas’ reservoirs that can be extracted by post-drainage techniques at a predetermined controlled rate. To avoid dilution of high percentage methane behind the seals, the quantity of captured gas should be in equilibrium with gas desorption rate from the strata gas sources. Part of the gas usually leaks directly to the ventilation system, however, the majority of high percentage methane could be recovered by goaf gas drainage systems.
The extracted gas can be diffused into the ventilated mine workings or transported by a methane drainage network to the surface for utilisation or controlled exhaustion to the open atmosphere.

The capture of gas from operating longwall goaf by an underground drainage system is rarely used in Australia, due to the common use of bleeder ventilation systems, however, some mines are capturing up to 75% of gas being released during longwall extraction, using surface gas wells. Following the collapse of the coal seam roof, the subsequent fracturing of the surrounding rock and coal strata allows the goaf wells to produce large quantities of methane in a short period of time. After the initial surge of methane, the quality of the gas may decline as it becomes mixed with air from the mine workings. However, in some cases, the methane concentrations have been kept very high for long periods of time. Over time, this production rate declines until a relatively stable rate is achieved. Most methane produced from the goaf wells is currently vented to the atmosphere.

When extraction is completed the systematic and appropriate sealing of individual longwall goaf and/or districts allows for capturing high concentration methane from sealed goaf areas. It allows for management of ventilation air pollution in the underground workings as well as protection of the environment by reducing greenhouse effects.

A prototype of automatic goaf gas drainage assembly, which utilises barometric pressure and methane concentration sensors for controlling both quantity and quality of captured gas was developed and installed in an underground gassy mine in Australia. This system substantially increases efficiency of every goaf drainage system.

COAL MINE METHANE UTILISATION PRACTICES

In some cases, extracted coal mine gas is vented to the atmosphere, however, many mines use extracted methane as a fuel for heating or power generation. A typical example is a power generation project (EDL-BHP) of a 94MW combined capacity in New South Wales. The coal mine methane is converted to electricity using state-of-the-art lean burn 1MW reciprocating gas engine technology. A key strength of this project is the adaptation of existing engine combustion technology with fuel system which enables gas engines to utilise fuel gas with very low heating value and of highly variable compositions typically varying from 30-80% methane, and utilisation of coal mine gas ventilation air containing typically 0.1-1.0% methane, as supplementary fuel for gas engines.

More recently a newer more advanced technology has been introduced for commercial use in Australia (Appin Colliery). It utilises ultra lean low pressure MVA as a primary fuel source. This approach represents leading edge technology in coal mine methane utilisation from ventilation air offering the opportunity to significantly reduce greenhouse gas emissions.

Coal mine gas could be diluted underground by the ventilation system, captured by gas drainage techniques, vented to the atmosphere and/or utilised. Coal mine safety, assurance of coal productivity and protection of the environment from greenhouse methane and carbon dioxide gasses emission, are the most important issues associated with gassy mines underground activities.

Since 1980 Australia has introduced various gas drainage technologies and developed some new techniques applicable to the local conditions. A comparison of coal mine gas parameters and achievements in Australia and other countries are presented in Table 1.
Table 1. Gas conditions for selected underground gassy coal mines (1997)

<table>
<thead>
<tr>
<th>Country</th>
<th>Underground coal production</th>
<th>Methane emission</th>
<th>Gas capture</th>
<th>Gas drainage efficiency</th>
<th>Methane utilised</th>
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<td></td>
<td>*10^6 t/year</td>
<td>m^3 CH4/min</td>
<td>m^3 CH4/min</td>
<td>%</td>
<td>%</td>
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<td>2400</td>
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GAS EMISSION PREDICTION, GAS RECOVERY LAYOUTS DESIGN & PARAMETERS OPTIMISATION

The quantity of gas to be released from gas-bearing sources and to be captured by various gas drainage techniques can be predicted with varying success using worldwide-recognised methods, including strata modelling and gas drainage hole design simulation software.

Gas drainage efficiency plays a very important role in the protection and prevention of gas and outburst hazards during various stages of mining activity. Both pre-drainage and post drainage techniques are essential in planning ventilation and in developing an efficient mine gas management system.

A strong direct relationship exists between the underground coal mining activities, strata relaxation, gas emission zones shape, and the quantity of gas captured by various gas drainage systems. Lunagas Pty Limited’s new software - ‘Floorgas®’ and ‘Roofgas®’, allow for the design of optimum parameters for various gas recovery and methane utilisation systems, as well as accurate prediction of gas emission from roof and floor strata gas sources.

The advent of computer modelling methods, particularly finite element techniques, has improved predictions based on the nature and extent of the relaxation zone surrounding the longwall to be made when using nominated local geomechanical, geological, gas and mining input data. Such a model has been developed and evaluated under the name of ‘Floorgas® and Roofgas® geomechanical & gas release model’, and has been commercialised to operate on a PC Windows based platform.

Outputs from both programs (Figs. 3, 4 & 5) are used for routinely designing gas drainage technologies, including cross measure (Figs. 2, 4, 5 & 6) and/or directional holes drilled from underground (Fig. 4), and gas wells drilled from the surface. Both programs are the most advanced engineering numerical tools available, designed to calculate active and inactive gas source contributions, and improve the accuracy and quality of underground coal mine gas management, and coal mine methane utilisation.

Floorgas® is the only known system to combine a precise rock mechanics analysis with gassy conditions to calculate stress and gas release zones in the floor strata of the working coal seam (Fig. 5).

Roofgas® is the only system known which can generate a roof strata break-line as a boundary between continuous and discontinuous rock masses (Fig. 3) and/or incorporates longwall face and side-heading edge effects, as well as the apparent geometry and shape of gas conductivity zones, due to bending curvatures and sag caused by dynamic edge migration specific phenomenon (Fig. 4).
GAS MANAGEMENT SYSTEMS

Current research emphasis is not on the search for new technology only, but on enhancing the organisational management to ensure the available, proven technology is used to best effect. A greater range of gas drainage technologies have been applied in Australian mines compared with those in the UK, as the higher production rates tend to induce problems with high gas emissions and the potential for outbursts at some mines. The support of an effective gas management framework is therefore even more critical. Following the explosion incident at Appin - 1979 & at Moura - 1994 as well as fatal outburst occurrences at South Bulli - 1991, Tahmoor - 1985 and Metropolitan - 1954, 1926 and 1896, (Harvey, 1994), new management concepts have been introduced in Australia including quality management system (Outburst and/or gas management plan), which specifies practices, resources, activities and responsibilities so that all procedures designed to manage gas hazards and the outburst risks are in place to guarantee the safety in underground gassy mines. Each outburst management plan is based upon a three tiered approach for managing the outburst risk by the prediction of outbursts, prevention of outbursts and protection from outbursts (Harvey, 1994).

An effective gas management system will include an operational framework to identify, provide, implement, monitor and develop the most suitable technology in an effective and sustainable manner. The procedures should ultimately address all relevant aspects in accordance with local legislation with regard to safety, gas use and the environment. Safety is always the highest priority. Effectively managing gas to ensure safe working practices essentially means achieving a correct balance between having sufficient ventilation quantities to dilute and disperse gas entering the general body of mine air at all levels of planned coal production and, where ventilation alone is unlikely to achieve this, gas drainage to ensure no more gas enters the mine airways than can be diluted to below statutory limits by the available ventilation air (Creedy and Lunarzewski, 2001).

UNDERGROUND CONTROL AND FUTURE DIRECTIONS OF COAL MINE GAS

Both safety and productivity in underground gassy coal mines can be substantially improved with the application of an appropriate gas management system, particularly if it includes gassiness calculations, implementation of appropriate ventilation, and gas recovery systems in various stages over the life of the colliery.

Underground coal mines with high coal production and gas emission levels require a combination of both ventilation and various gas drainage systems to be applied simultaneously in order to meet the needs and demands of each individual mine with regard to gas, geological, and mining conditions. Various systems of gas drainage should be applied in different phases over the life of the colliery, in order to minimise the gas hazard and to optimise the ventilation network and seam gas utilisation. Pre-drainage and post-drainage are gas recovery methods (Figs. 2 & 6), complementary to the ventilation system, which allow for the control of gas hazards in underground mines, as well as the utilisation of coal seam gas. Gas capture techniques are directly related to a colliery’s life cycle; before, during and after coal extraction.

Improvements in gas control in underground coal mines involve the introduction of safety management systems and utilisation of modern gas drainage techniques. Multi-entry access configurations for retreat longwalls offer improved gas control benefits in terms of at least one heading being maintained behind the face to protect gas drainage holes, and to improve access for gas drainage in general. Developments in technologies for drilling, monitoring and guiding longholes hold promise for application to both pre-drainage and post-drainage methods of gas capture in such conditions. Further experimentation is needed with longholes drilled above and below retreating longwall panels, as this approach has the potential to optimise methane drainage performances. Where coal seams are of sufficiently high insitu permeability, a proportion of the gas from the seam to be worked, or from adjacent seams, can be removed prior to mining through holes drilled either in-seam or from the surface. Pre-drainage from surface holes is generally a nonviable option for deep seams or in residential areas, and is a technique which is usually practised totally independently of mining to obtain coal mine gas for commercial exploitation. However, pre-drainage technology need not be confined to the conventional stimulated vertical well concept. A proposal to drill and complete long deviated holes from the surface and/or underground, into the target coal seam with enhancement using hydro-fracturing technology is currently being promoted as a viable alternative for reducing gas hazards ahead of mining. New techniques allowing the efficient sealing of single longwall or multi-longwall district goaf areas are of primary importance for the capture of huge quantities of gas currently being diluted and vented by the ventilation air (Creedy and Lunarzewski, 2001).
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GAS EMISSIONS MODELLING OF GATE ROAD DEVELOPMENT

BY RJ WILLIAMS, E YURAKOV AND DJ ASHELFORD
GAS EMISSION MODELLING OF GATE ROAD DEVELOPMENT

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ABSTRACT: This paper covers the modelling of gas emission during gate road development. Complexities related to the range of gas reservoir and mining data are briefly described. The rib emission response to initial gas reservoir parameters is defined using the SIMED II gas reservoir simulator. The rib emission decay curves are then used in an interactive EXCEL spreadsheet model that takes additional account of mining and ventilation parameters. It is an attempt at bringing to planners and mine operators an interactive tool that incorporates the important parameters and options.

SIMED’s strengths lie in it’s ability to:

- quickly convey a message that facilitates action toward improved ventilation and/or gas drainage design and implementation.
- contain all the important gas reservoir and mining parameters within a manageable package.

INTRODUCTION

The sensitivity of mining to gas emission is rising, with planned production rates in excess of 6 million tonnes per annum, increasing gate road lengths (> 4 km), wider longwall faces (to 350 m), thick seam mining and higher gas content coal. For long gate roads, supplying sufficient air to support auxiliary fans can be problematic, and is particularly sensitive to the number and quality of cut through seals. The added leakage from shorter cut through spacings (e.g. 60 m) in support of place changing can be significant. This burden is increased where intake bleed air is required for cooling conveyor belt tripper drives.

A high gas environment may not be able to support large capacity auxiliary fans given the limitations on intake/return pressures and air loss due to leakage through stoppings. Limitations in the ability to supply face air quantities through long gate roads and associated leakage result in gas drainage being required sooner than would be the case for shorter gate roads.

From a mine planning point of view, the following information is required:

- Intake gas concentration at the start of the hazardous zone
- Gas concentration in the face area
- Gas concentration in the return at the beginning of the panel
- Intake/return ventilating pressure difference at the start of the panel
- The quantity of air entering the panel that will meet the requirements for auxiliary ventilation and gas dilution.

In addressing these requirements, both gas reservoir and mining parameters need to be considered. This paper broadly outlines GeoGAS’s approach to gate road gas emission modelling for assessment and control.

PARAMETERS TO BE CONSIDERED

The rate of gas emission into a mine roadway is initially dependent upon the following main, gas reservoir characteristics:

- Measured gas content (Qm) at reservoir temperature
- Gas desorption rate
• Gas composition
• Gas sorption capacity at reservoir temperature
• Seam thickness and mineral matter (ash/density)
• Permeability (including directional) and relative permeability
• Pore pressure
• Coal porosity and compressibility

These parameters and their measurement are described in Williams, Casey and Yurakov (2000).

Gas reservoir characteristics in the Sydney and Bowen Basins can be highly contrasting. Where seams dip significantly (Bowen Basin, Hunter Valley), mining is conducted in a rapidly changing gas environment. Normally, permeability decreases with depth while gas content increases. These depth gradients can vary considerably (Figs. 1 and 2), even between adjacent areas from the same coal seam.

![Fig. 1 Example Actual Gas Content Gradients](AllGasContent.xls)

![Fig. 2 Example Actual Permeability Gradients](Allperm.xls)
Creation of the mine opening results in reduction in pore pressure, gas desorption and migration of gas toward and into the mine roadway. The rate of gas emission is dependent upon the above gas reservoir parameters, including the geometry of the mine roadways and their proximity to one another. The SIMED II gas reservoir simulator is used to define the time dependent rib emission decay curves.

SIMED II is a two phase (gas and water), three-dimensional, multi gas component (i.e. CO₂, CH₄, N₂ etc) with simultaneous modelling capabilities, single or dual porosity reservoir simulator. The dual porosity capability is used for coal seams, simulating the slow, concentration gradient driven, desorption from the coal matrix, and the pressure gradient flow of gas through the fracture network.

Gas content alone is a poor indicator of likely gas emission. The combined effect of permeability, gas content and seam thickness require gas reservoir modelling to define the emission outcome. Gas emissions (rib emission) can be just as high at low gas contents (e.g. 3 m³/t) as at high gas contents (e.g. 15 m³/t), with the reduced gas content compensated by higher permeability and thicker coal environments (Fig. 3).

The rib emission decay curves importantly define the rate at which gas will be emitted into a roadway, according to its age. The effect of varying the rate of mining (panel advance) is readily calculated from the decay curves. The result is still not realistic, requiring the inclusion of ventilation parameters and related geometry. Mining related parameters are:

- Number of headings (two or three)
- Panel length
- Geometry of the excavation (height, width, number of roadways, pillar width and length)
- Rate of panel advance
- Air quantity supplied to the start of the panel
- Roadway friction factors
- Stopping resistance

The last three factors determine the amount of air that will be supplied to the last open cut through.

Emission in the face area combines rib emission (according to the rib emission decline curves), with emission from cut coal. It is a snap shot of mining just prior to the cut through holing, at a time when the auxiliary fans are at their highest duty. The emission from cut coal is dictated by desorption rate characteristics of the coal, adjusted for lump size. Mining related parameters are:

- Face cutting rate
- Time for cut coal in the face area
- Mean cut coal lump size
- Pillar width
- Roadway drivage rate
- Shifts per day mining
- Days per week mining
Fig. 3 Rib Emission Decay Curves Showing Compensating Effects Between Gas Content, Permeability and Seam Thickness

THE MODEL

SIMED II uses fundamental numerical modelling to generate the rib emission decline curves. That achieved, the curves in themselves are still of little direct use to mining. An EXCEL spreadsheet model incorporates the SIMED derived rib emission curves and adds the effect of ventilation related parameters, mining rate, number of headings, face gas emission. With so many options and variables to be appraised, the EXCEL model has been designed for interactive use by mine personnel.

There are always varying degrees of uncertainty in the data. Provision is made to account for this by providing “High”, “Mean” and “Low” emission options that apply to rib emission decay curves, reflecting combinations of gas reservoir parameters modelled by SIMED. A more rigorous account of uncertainty can be done by using probability modelling within the EXCEL spreadsheet (e.g. using the EXCEL add on package “@RISK”).

The greatest amount of time and care is usually required in specifying parameters for SIMED modelling. A common situation is a gate road developing down dip, with changing gas content, seam thickness, permeability, pore pressure and gas desorption rate. Along the line of the intended gate road, these parameters are graphed (example Fig. 4). To make the SIMED modelling work manageable, the gate road is divided into regions where the gas reservoir parameters have been averaged. For the case in Fig. 4, three regions have been defined.

The resulting rib emission decay curves for each region’s set of gas reservoir parameters are incorporated into the EXCEL model. The model can chart the gas emission as the panel is developed. When development passes from one region to another (e.g. from Region A to Region B in Fig. 4), a different set of rib emission decay curves is invoked.
The main input and output work-sheet ("Emission" Fig. 5) shows a range of results including gas emission, gas concentration and ventilation quantities at preset locations specific to planning and statutory requirements, according to the options selected and values used. In this example, the emission decline curves used in the third region are for coal predrained to 4 m$^3$/t.

**Fig. 4** Example Gas Reservoir Parameter Profile Along an Intended Gate Road

**Fig. 5** Example of Interactive Worksheet “EXCEL” Model
The “FaceEmission” work-sheet (Fig. 6) allows for peak mining rates in the face area. It is independent of the rate of panel advance outbye the face area, where the rate of mining (“Panel Dev Rate (m/day)”, Fig. 5) refers to total panel advance in calendar days.

**CONCLUSIONS**

Gate road gas emission assessment is highly complex, with a range of gas reservoir and mining parameters needing to be considered. The modelling approach, beginning with SIMED II and being completed as the EXCEL spreadsheet interactive model is an attempt at bringing to mine operators and planners a tool that can incorporate all the important parameters and options into a manageable package.

While it has been widely applied in New South Wales and Queensland validation through back analysis largely remains to be undertaken. The face emission aspect is the weakest part of the model, and ideally requires hard data (continuous return gas monitoring results), to better specify parameters.

For now, its strengths lie in its ability to:

- quickly convey a message that facilitates action toward improved ventilation and/or gas drainage design and implementation.
- contain all the important gas reservoir and mining parameters within a manageable package.

**REFERENCES**

LONGWALL WEBSITE FOR AUSTRALIAN MINING CONDITIONS

BY NI AZIZ AND S CHAMBERS
LONGWALL WEBSITE FOR AUSTRALIAN MINING CONDITIONS

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ABSTRACT: The mining engineering course at the University of Wollongong is in the process of continuing restructuring to meet the challenges and changing demands on mining engineering education in Australia. An online interactive student resource on longwall mining has been developed at the University of Wollongong to supplement formal delivery of the subject material. The interactive website allows students to gain a deeper understanding of longwall mining in their own time and at their own pace. The website had been circulated to students and various industry quarters seeking their comments and advice for the future directions of this type of learning system. The comments received from the industry personnel were very encouraging. There was a general desire by various industry personnel to also use this website for industry training.

INTRODUCTION

All mining engineering institutions worldwide including those in Australia teach core mining subjects to undergraduate students. The techniques of mining are best demonstrated when formal lectures are supported by field visits and hands-on practical experience. Where active mine sites are in close proximity to tertiary institutions, this is not normally a problem. Unfortunately most mine sites nowadays are remotely located from universities and educational institutions offering mining programs. In recent years large group access to local mines has been less than convenient. This makes the learning of certain mining methods a difficult task for students. Conventional teaching methods include the static use of overheads and sometimes videos to explain simple operations to students. Concept of equipment sizes and three-dimensional visualisation of unit mining operations are not always easily grasped by students. As a consequence, an on-line student resource on longwall mining has been developed to:

- serve as a supplement to formal teaching of longwall mining to students enrolled in the subject of underground coal mining methods;
- gain a better understanding of longwall mining in each student’s own time and pace;
- allow informal on-line interaction between students and lecturers by the incorporation of a self-assessment component into the package;
- keep abreast of latest information and technologies used in Australian mines since the website can easily be maintained;
- gain access to various statutory mining legislation’s and laws as the website is linked to various government organisations and legislative bodies websites.

PROJECT DEVELOPMENT

A research assistant with mining engineering qualifications was recruited to develop the website. Reliance on a trained mining engineer was necessary in view of the nature of underground mining operations not being easily understood and visualised by others not trained in the discipline. Developing the website in-house provided an opportunity to master skills for future development of other sites as well as regular upgrading of the existing website. The initial introduction is to describe and illustrate the basic elements of longwall mining commencing with a basic definition of longwall mining and expanding to more complex issues related to the operation and problem solving associated with longwall mining. Accordingly, the structure of the developed website on longwall mining falls into the following components:

a) general introduction to longwall mining;
b) general design and layout of longwall mining;
c) longwall mining machinery and equipment;
d) ventilation and environmental aspects of longwall mining;
e) geomechanics and ground control in longwall mining;
f) longwall change over techniques;
g) punch longwall mining;
h) glossary of longwall mining terms and references;
i) student and staff interaction.

The website was developed in the standard html format that is commonly used on the internet. Access to the site is via any internet browser (i.e. Netscape Navigator or Internet Explorer) and the URL of the website is http://www.uow.edu.au/eng/current/longwall/. One of the aims of this project is to present the information to the students in a user friendly and technically attractive style. The subject that this website was designed for is a very descriptive subject and students previously undertaking this subject have found it difficult to comprehend and imagine in comparison to a real field situation. The incorporation of various video footage on actual coal winning and equipment functioning has contributed significantly to student’s better understanding of the subject.

WEBSITE CONTENT

The technical content of the website is based on class lecture notes of the academics involved in teaching the various longwall mining subject components. This is further supplemented with technical material from industry personnel in the field as well as specialist mining consultants. The reported case studies and the future ones to be incorporated will be supplied from mining personnel and expert industry consultants. Although the website is linked to various national and international websites it will not be used to actively promote any company, product or alike. The website’s primary function will be for educational purposes only, both for students and for knowledge upgrading of mining industry personnel. The inclusion of a self-assessment component is vital to the credibility and acceptability of the site. A method of determining the competencies of persons and their knowledge of the topics is becoming more critical in view of the mining legislation direction for future training of mine personnel working at the longwall face in Australia. Another section needs to be developed for the number of instances for the recovery of longwalls from disturbances of ground or inadequate maintenance of equipment or incorrect operation of the equipment. There is a lot of case history in Australia and overseas which has allowed innovative techniques to be developed and would also save many millions of dollars to the mining industry.

To make it easier for students to use the site a standard/template was developed and this structure was incorporated onto every web page. A general view of the front page of the website can be viewed in Fig. 1.

![Fig. 1 Front Page of the Longwall Mining Web Site](image-url)
The structure used for this website was as follows:

i. **Universal Navigation System:** This system allows the users to move around the website with ease. The agreed system consists of a simple menu bar that is located in a column on the left-hand side of every page. At the top of the menu bar is the Mining Engineering logo of the University of Wollongong. Below the logo is a list of pages/modules that could be accessed from that particular page. To the left of each of the menu buttons is a mining icon (the well-known hammer and pick crossed) used to indicate the user’s current location on the website by making the mining icon turn green. Located at the bottom of the menu bar on every page are the four ‘global’ navigation buttons:

- HOME: Returns the user to the front page of the web site.
- GLOSSARY: Takes the user to a comprehensive list of terms and their definitions that are associated with longwall mining.
- REFERENCES: This button takes the user to a list of references that are associated with longwall mining, enabling the students using the web site to further research longwall mining from these particular references. The students can also check the University of Wollongong’s on-line library catalogue to see if the references listed are available at the library.
- TOP: This button allows the user to immediately return to the top of the viewed web page when they have scrolled down to the bottom of it.

ii. **Content:** The content of the web page takes up the remaining space left from that used by the navigation system. The content is presented in such a manner as to allow the student to read about a particular topic and then view a graphical diagram of that topic. To accurately portray current longwall operations in Australia, the lecture material incorporated into the web site is well researched.

When the user accesses the site the index page in Fig. 1 is displayed on the computer screen. A banner is incorporated at the top of the index page that says “Longwall Mining”. Beneath the banner is a photograph of a modern longwall face from a current Australian Longwall face operation. In the navigation bar of the index page there is a series of menu buttons for the various learning modules that are available to the viewer. The topics incorporated into this site are:

a) **History and Methods:** This module provides an introduction into longwall mining in Australia and throughout the world. It incorporates a short introduction into the basic concepts of longwall mining and the various methods that can be used to extract coal by longwall mining. A unique feature of this module is that it contains details of every currently operating longwall face in Australia including:

- longwall production figures for the last calendar year,
- equipment used at each longwall mine site,
- layout plan of each longwall site,
- coal seam mined,
- coal transportation,
- method of underground access,
- contact details of the sites,
- geographical location, and
- commencement date of longwall mining operations.

b) **Equipment Overview:** A thorough description of longwall face equipment is described throughout longwall faces. These include:

- coal shearer
- coal plough
- powered roof supports
- armoured face conveyor
- pantechnicon
- beam stage loader
- communications
- environmental
c) **Ground Control:** An important aspect of longwall mine design is to understand how the surrounding ground stratification reacts when a tunnel is driven in the strata. The general stress build up around a longwall panel is described and demonstrated graphically as shown in Fig. 2.

d) **Ventilation:** The design of a ventilation system for a longwall mine is dependent upon the geological and atmospheric conditions found at each individual mine site. Many factors have to be considered to determine the most suitable system of ventilation.

e) **Longwall Changeover:** Once a longwall panel has been fully extracted the longwall equipment is dismantled and moved to a new panel. This operation is called a longwall changeover

f) **Punch Longwall:** A method of longwall mining from the highwall of an open cut operation, in which the stripping ratio far outweighs the production cost of coal mined as shown in Fig. 3.

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**Fig. 2** Longwall Panel Abutment Stresses

**Fig. 3** Punch Longwall Mining
WEBSITE SURVEY

During the development of the site an evaluation of students that had already completed the subject was conducted to gather their input and ideas for further development on the website. The students navigated through the website for approximately half an hour and were then asked a series of question at the conclusion of the session. Some of their responses are given below:

- “It is great because you actually get coloured pictures, movies etc right where your information is so you visualise what your reading”
- “Yes, very helpful. I wish we had it for our sessions work.”
- “It helped my understanding of longwall mining by a great deal due to the videos and diagrams.”
- “It helped a lot, doing a 2nd year subject it is the 1st time you are exposed to anything mining related and the concepts can sometimes be confusing but here they are set out logically.”

INDUSTRY RESPONSE

The following comments were received from different industry quarters:

- “As a non-engineer I found it a very easy site to navigate, with easily accessible links. The quality of writing is excellent, and I was able to easily understand the concepts involved.”
- “I would like to congratulate you on the new website, it is very good and extremely comprehensive. The website is a good general introduction to the areas of longwalls. It would be enhanced by the addition of two areas. (i) There needs to be a method of determining the competencies of persons and their knowledge of the topics, this is becoming more critical with the way mining legislation is going for all training in the future. Each of the sections needs an assessment module attached if this is intended for industry training. (ii) Another section needs to be developed for the number of instances for the recovery of longwalls from disturbances of ground or inadequate maintenance of equipment or incorrect operation of the equipment. There is a lot of case history in Australia which has allowed innovative techniques to be developed and would also save many millions of dollars to the industry. The case histories would have to be sought from the various mines who have had longwall failures due to those criteria I gave you and extended delays.”

CONCLUSION

The website on longwall mining is developed primarily as a tool for effective teaching in tertiary education. The website has been placed into the public domain to assist in the upgrading and training of mining industry personnel as well as raising awareness of the mining operation to the public in general. The website would be a valuable source and a useful library for those interested bodies in remote regions and rural areas of Australia and also throughout the world. Although the website is interlinked to various national and international websites. It is purely an educational website that in future will also be a website for advanced training in various aspects of mining engineering which will cover more complex issues for improved safety in mining operations.

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GEOTEchnical ENGINEERING
at GERman CREEK – a HISTORICAL and SOMETIMES HYSTERICAL REVIEW

BY MITCH JAKEMAN
GEOTECHNICAL ENGINEERING AT GERMAN CREEK –
A HISTORICAL AND SOMETIMES HYSTERICAL REVIEW

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ABSTRACT: The initial drilling exploration started in what we now know as the Bowen Basin coal region in the 1950’s, to open up and delineate the coal seams for the whole region. After Utah grabbed the easiest and more favorable large sandpits in the early stages, other companies started to understand the opportunities for the future.

In 1979, German Creek started construction and operations at the end of the boom and bought four draglines. This paper presents the geotechnical experiences at German Creek over 21 years of operations. Today, the operation sustains one dragline, two underground longwall operations and another underground in the project phase. Life long learning experiences and the need to understand what is happening has sharpened our focus about our mines and operational business risks. The task of digging out a longwall, whilst good for experience and character building, should only be ever done once or twice.

INTRODUCTION

The exploration drilling in the 1950’s was in the order of 2 holes per km². By the late 1970’s when German Creek went from feasibility to project status the spacing became 8 holes per km². Today it may go as low as 16 holes per km² dependant on the project risk. Current borehole densities in mining areas are 1 hole per 150m for structure and 300m for coal quality. Over the last 35 years the role of geotechnical engineering has become more important to understand the deposits we mine. The advances in bringing techniques from concept to maturity have gone from decades to years aided by such diverse areas as the space program and military applications.

The changes to legislation, the increasing requirements for more stringent corporate governance and shareholders expecting companies to manage all facets of their business has focussed our attentions on risk mitigation.

This paper looks at the use of geotechnical engineering in the operations of German Creek. There have been major advances in the technology from the planning predictive tools, the monitoring and modelling of what happens and some of the practical aspects of controlling. There is still room for improvements as at best we can at best only get it 70% correct for an underground operation, looking at the annual failure rates in our industry.

Geotechnical engineering impacts on all areas of mining. It affects ventilation, gas drainage, mine layout, mining methods, strata control, production and costs.

GEOLOGY

Stratigraphy

The German Creek Mine operations are based on coal reserves in the German Creek Formation and the Rangal Coal Measures. The former contain economic coal in the Pleiades, Aquila, Tieri, Corvus and German Creek seams (Fig. 1). In the latter, only the Middlemount seam is of economic significance.
Fig. 1 German Creek Mine Geological Setting

The mine is situated in the centre of the Bowen Basin and the operation is worked over a 12km-strike length. Seams dip to the east at an average grade of 5°. The strata containing the German Creek Group of seams are hard to very hard, well lithified, interbedded claystones, siltstones and sandstones with some massive sandstone beds overlying the German Creek, Tieri and Aquila seams. The sediments are well jointed with the primary joint set trending northeast and a well-defined secondary set trending southeast. In the mine area, the sediments were deposited in a fluvio-deltaic/paralic environment. The massive sandstone units found in the area have been attributed to beach bar deposition. Coal seams worked range in thickness from 0.5m to 4.0m.

The Middlemount Seam subcrops 8km to the east beneath a thin blanket of Tertiary clay and sand. This seam is mined by open cut strip mining in Pits T and U. The sediments overlying the Middlemount Seam are weaker than those of the German Creek Formation but overburden blasting is still required. Jointing is well-developed but less regular and pervasive than in the German Creek Formation.

Igneous Activity

Igneous activities in the form of sills and dykes have had a significant influence on mine design for both open cut and underground pits. Sills in the open cut have coked what otherwise would have been economic reserves in the Aquila, Tieri, and German Creek seams.

Early in the life of the open cut several dykes, ranging in thickness from 0.1m to 14m, were uncovered. Although having limited affect on the open cut operations, these dolerite dykes have had a significant impact on the underground operations (Fig. 2).
Central Colliery has encountered several dykes of variable thickness and hardness, which have hampered the mining operations. However, the layout of Southern Colliery was specifically designed to minimise the impact of dykes and in particular to avoid a 4m thick dyke encountered in open cut Pit A and the 14m thick Grasstree Dyke.

**Structure**

The German Creek Formation is characterised by a series of north to north-north-west trending normal faults. Faulting is more frequent in the area of the subcrop. The Grasstree-Central Colliery Fault system divides the mine into eastern and western development areas (Fig. 2). The structural features of the mine have been described by Whitby (1985).

**Seam Gas**

The deeper reserves in the mine area are characterised by moderate to high levels of seam gas, which is composed almost entirely of methane. Methane is encountered at depths from about 70m and at 250m gas content is approximately 8m$^3$/t. Seam gas content of 14m$^3$/t has been measured at 420m depth of cover in the German Creek Seam.

In those reserves containing a seam gas content of 7m$^3$/t or higher, methane drainage of the coal and strata is required to enable efficient and safe production. At Central Colliery methane drainage practice has been successfully applied to lower gas emission during development and longwall extraction. The process involves two stages:

- **Pre-drainage** – gas drainage ahead of development by in-seam drilling.
- **Post-drainage** – gas drainage of the goaf after extraction by surface boreholes.

Significant quantities of hydrogen sulphide gas were encountered in the early development of Southern Colliery. More detail will be discussed later in the paper.
UNDERGROUND GEOTECHNICAL AND HYDROLOGICAL INVESTIGATIONS

Central Colliery

Detailed geotechnical investigations were conducted over the Central Colliery mine area prior to commitment to longwall mining. Detailed geological mapping, surface geophysics, diamond and rotary drilling, downhole geophysics, permeability testing, laboratory testing of core samples and the determination of \textit{in situ} stress conditions were undertaken. Technical data on roof, floor and coal seam conditions, groundwater regime and inflow rates were provided for the selection of appropriate mining equipment for maximum production and a safe working environment.

Main heading pillars were designed on a conservative 50m x 50m basis. Chain pillars were designed using Wilson’s yield pillar design modified for Australian conditions and Commonwealth Scientific Industrial Research Organisation’s (CSIRO) Minlay numerical modelling programme. The design was validated by undertaking extensive in-pit geotechnical monitoring. Irad stress cells were installed in pillars, as were wire extensometers and rib extensometers. Sonic extensometers were used to determine roof and floor behaviour.

Chock shield design was based on a physical model constructed and caved at the Australian Coal Industry Research Laboratory at Wollongong. Chocks rated at 640t were initially recommended but following consideration of the geotechnical characteristics of the overlying strata, 800t chock shield supports were ultimately chosen.

Roof support in the development headings and gateroads was established by using beam theory. Bolt lengths were chosen to ensure adequate bond length in competent strata and to provide a stable bolted roof beam.

Southern Colliery

The experience gained from the development at Central Colliery was drawn upon to assist with the design of Southern Colliery roadways and chocks. Chain pillars were designed using the Minlay programme, as at Central Colliery. However, main heading pillars were designed using Bieniawski’s rigid pillar design for optimal pillar dimension to reduce development drivage. Final pillar dimension was 100m x 30m. Sonically derived uniaxial compressive strengths (UCS) were used to design roof bolting patterns under Southern Colliery’s massive and bedded sandstone roof.

The 800t capacity chocks used at Central were selected at Southern Colliery following detailed geotechnical study and finite element analysis.

Water Management

Both collieries have subsided strata under known aquifers. The principal aquifers are semi-confined igneous sills, which lie within the critical tensile strain zone above the German Creek Seam. The water contained is highly saline.

In the case of Central Colliery, the mine subsided the Tieri Sill aquifer, which, on initial goafings, caused a minor inrush of 25L/sec of water into the mine. This water was managed underground by pumping the water to the surface through gateroad boreholes and that magnificent technique of running the longwall AFC.

Studies into this event and subsequent field investigations enabled a pre-drainage programme to be designed for the Aquila Sill aquifer which overlies the 600’s block at Southern Colliery (Klenowski and Phillips, 1998). Techniques used to define the aquifer included routine airlift pumping in exploration drill holes, downhole geophysical and geological logging and upstage packer testing. Permeability and other hydraulic parameters were calculated from pumpout and pump-in test results to predict inflow rates. It was estimated that initial inflow into Southern Colliery would reduce this to the order of 165L/sec. Pre-drainage would reduce this to the order of 45L/sec.

Four pumphole sites were required and dewatering of 4km$^2$ of aquifer was achieved before longwall mining began, using downhole electric submersible pumps. The dewatering boreholes continued to pump a total of 600ml over a 4yr period to reduce the inflow rates to less than 5L/sec after initial caving.

Southern Colliery underlies several abandoned open cut pits in which water can collect during storm (cyclone) events. These pits have been shown to have direct connection to the mine on caving and thus provide a potentially dangerous environment. High rainfall events in excess of 10mm/hr can result in water ponding in spoilpiles and
open cut pits. This water can percolate into the mine through subsidence cracks at rates of up to 150L/sec, as was experienced during Cyclone Joy in January 1991. Considerable effort has been expended in protecting Southern Colliery from flood event by blanketing the floor of the open cut voids with compacted parting and reject material and by ensuring that all spoilpiles drain externally. In-pit surface pumps are also installed to ensure these pits remain dewatered during high rainfall events and before the area is subsided.

**OPERATIONAL EXPERIENCE**

**1980 – 1984**

German Creek was planned at the height of the energy crisis and coal boom of the late 1970’s. Everything looked rosy as Utah was scrapping off a bit of dirt and finding coal everywhere. Prices were expected to continue rising and margins were fat.

German Creek had a lease length of about 16kms and four economic seams to mine. With such a great prospect, four draglines were purchased to start production in 1981 to 1984.

During 1984, German Creek started development of Central Colliery and by 1990 we had sold two hardly used draglines.

The promised land was starting to tarnish. We had to contend with three creek diversions, two large silled out areas and a thinning of two seams. Suddenly our 16km of strike length was greatly reduced as our basic geotechnical knowledge was based on too few holes, highly extrapolated assumptions and a poor knowledge base for planning. This was rapidly improved during the feasibility stage for each underground mine before approvals were granted (Fig. 3).

![Fig. 3 German Creek Mine Leases](image)

**1985 – 1989**

This period was a combined operation with four draglines decreasing to two, and the first underground longwall mine in Queensland.
Central was now in full production and soon became one of the top consistent producing mines in Australia until the mid 90’s.

Better geotechnical knowledge and planning was necessary to get the Boards approval to start a new underground mine after a relatively short period of time in starting the whole mining operation. German Creek needed the underground resources to supplement the diminishing quantities from the open cut reserves to meet customer specifications and tonnages.

Although we knew the basic geotechnical data for Central in terms of seam thickness of 1.6–2.4m, dip of 5–6 % and coal quality data – there was a certain amount of guess work and good luck. The conditions were good with strong competent roof and floor, no gas except as it got deeper beyond 250m depth of cover, little structure and no major stress problems. In all of this and with the wisdom of hindsight of today – the mine layout was wrong and should have either had the main headings continuing off the main drift access or move the drift access to align with the main heading, instead of mixing the worst features.

After the success of the first few years of Central and the continued demise of the open cut, feasibility plans were started to bring Southern online. In 1988 we sold another dragline and Southern’s construction was completed down an old open cut pit haul road to access the highwall (Fig. 4).

![Southern Mine Access](image)

**Fig. 4  Southern Mine Access**

**1990 – 1994**

The start of the 1990’s had seen a complete change in the original intent of German Creek. It had changed within the decade from a large four dragline open cut to an underground operation with some open cut production.

The next five-year plan was based on seeing the end of the open cuts by 1995, Central producing 3Mtpa and Southern producing 3.75Mtpa.
Central did produce well for a very long time in benign conditions and reached 3Mt in 1992. Some of our technical designs on later analysis actually showed that we created bottlenecks in the system when we were correcting other problems. Much of our geotechnical knowledge was now substantially better but our approach to managing or controlling the issues it highlighted were poor or absent. Better exploration and mapping now indicated areas in Central which would be affected by shear zones, dykes or faults in a north east south west direction. The operational response was to mine in the good country only and leave any other areas as too hard. Consequently if you look at the plan of Central, many of our panels on the 300 side of the mine are a lot shorter than the 200’s side – but the coal on the 300’s side is better quality and thicker.

We also know that as we got deeper, then the stress problems due to depth of cover and increasing quantities of methane would have to be managed. At the start of the 90’s both of these issues were thought of at the time would effectively close Central when it reached about 250m depth of cover limit if nothing was done.

The story at Southern was different and many of the early design failings of Central improved. Southern was bounded and restricted by lease boundaries and some major regional faults in the initial layout of the mine. It also suffered from poor selection of personnel creating industrial problems for a long time. During the mining of the 600’s, we encountered for the first time in an underground situation, the presence of small pockets of low concentration hydrogen sulphide.

In late 1990 we got approval to start mining another open cut resource in the Rangles measures at German Creek East. This created a new set of operating conditions for low-highwall stability, blasting and box cut angles on set up, as the material had no structure. During this time we started highwall mining using Eltin to trial this in some of our completed pits. This showed some promise but again the geotechnical understanding of pillar design and controlling the direction of cutting caused some problems. From the underground experience at Central and Southern, the geotechnical knowledge of roof and floor conditions was well known. This has been used four times with mixed success by Eltin, MTA and Roche Bros. (Fig. 5).
1995 – 1999

During this period we were down to one dragline in German Creek East continuing to supplement the undergrounds and becoming more efficient to survive. Major organisational changes were starting to occur in an industry under pressure to survive.

By 1996 Central had to manage the increasing problems caused by methane. The earlier attempts started in 1993 around 306 longwall were not good enough now. Major delays were starting to occur both in development and from the longwall goaf. It was after Moura in 1995, that a trend to enclose active longwall goafs was adopted in Queensland to minimise the affects of spontaneous combustion. This created a more dangerous situation with a build up of methane in the goaf and the problem was recognised to allow controlled bleeding of goafs.

Considerable efforts were needed to introduce gas drainage into Central for extensive post-drainage of goaf areas. The use of gas drainage and some other enhancements has assisted the continued operation of Central.

More work was started on how to manage the increased stresses due to depth of cover and the dip of the seam. Chris Hansen (German Creek) and Russell Frith started to prepare a comprehensive modelling, measuring and monitoring program to provide practical controls for stopping the three or four roof failures we were getting each year. Extensive panel maps were prepared identifying all the geotechnical data extrapolated and known in section and plan. Overlying these plans we superimposed ventilation, roof and rib support and panel layout plans to identify modifications of controls that may have to be tightened. The use of drilling to verify gas and ground conditions, in seam seismic and use of extensometers to provide better information has allowed a more robust design of strata control devices.

Gate road pillars have been increased by 5m as we have passed the 300m depth of cover. The use of flexibolts, cable bolts and increased tension on the bolts as well as the judicious use of tin cans and wooden cribs have reduced the frequency and damage of falls. Ongoing work is going on with rib support and understanding the effects of directional mining and the minimising of regional stress effects.

Fig. 6  Southern Longwall 701
In late 1995, Southern was forced to relocate its longwall operation to the 700’s area, due to a major dyke dividing the lease. This altered the mining panel layout and the new area was bounded by four major geological inferred features. The drivage of the gate roads for 701 encountered a larger extent of H2S in the first quarter of the panel. Exploration drilling from the surface identified substantial beds of sandstone, which would create heavy weighting events in the last section of the panel.

What we didn’t pick was the change in floor condition, even though the continuous miner had some indicative problems during development of the face line and last three pillars. The longwall started production in January 1996, got bogged, the roof fell in and was finally recovered in July/August/September. It had moved about 100 metres in that time and we found out the floor strength was less than 7MPa. After the longwall was moved beyond the H2S zone and we had repaired the damage to equipment, we tightened our operation procedures and had a good run to the end of the block (Figs. 6 & 7).

702 Longwall started with all chocks retrofitted with base lift rams, very tight operational control and chocks operating at 100% system efficiency. We were faced with three geotechnical mining risks – soft floor, a larger H2S zone and large area of heavy roof. It would have been a courageous move (or short career) to start at the same place, so we moved the longwall outbye where the floor strength was greater than 10MPa. This worked well and we experienced no problems. We collaborated with the University of Queensland to:

a) try and predict where the H2S was,
b) it’s method of deposition, and
c) how to control it.

There are some excellent research papers on this by Harvey, Gillies and other (2000), but the parameters around the H2S changed each block. In summary, although we had some success with drainage by trying to put it into a solution, which we pumped into the area, the conditions then changed and became too variable. The best option was excessive ventilation with monitoring to get the H2S (which is only released whilst cutting) into return airways. The heavy roof cutting procedures developed during 701 longwall worked well for 702 and all later panels.

At the start of 703 longwall after an excellent start up and about 85m from the beginning of the panel, the longwall was suddenly subjected to a severe weighting event. From an operating height of 2.9m, the weighting caused over
a 1m leg closure in a 20 minute period. Fortunately due to the position of the shearer and quick thinking by the operators, the shearer was moved to the tailgate. Some back calculations after the event indicated that you would have needed chocks capable of withstanding a 2000t load to support the roof. After looking at the final result, we set a plan in place, which recovered the height over some 45 chocks, in 16 days without any more convergence or weight occurring (Fig. 8).

![Fig 8 Southern Longwall 703 - Heavy Weighting](image)

The following table shows the range of geotechnical risks and quantifies the impact on production that Southern has experienced. We had two sudden weighting events mid face in different locations resulting in about two weeks of lost production in 702 and 706.

<table>
<thead>
<tr>
<th>Table</th>
<th>FI</th>
<th>Weighting</th>
<th>Soft Floor</th>
<th>H₂S</th>
<th>Heavy Roof</th>
<th>Parallel</th>
</tr>
</thead>
<tbody>
<tr>
<td>701</td>
<td>-</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
</tr>
<tr>
<td>702</td>
<td>-</td>
<td>-</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
</tr>
<tr>
<td>703</td>
<td>-</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
<td>✓</td>
</tr>
<tr>
<td>704</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<td>✓</td>
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<td>-</td>
</tr>
<tr>
<td>706</td>
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<td>?</td>
<td>-</td>
<td>?</td>
<td>✓</td>
<td>-</td>
</tr>
</tbody>
</table>

The drivage in main headings encountered some increased quantities of CH₄ and change to the roof lithology which lead to two frictional ignition events. Although both events were small in nature and quickly controlled, the Southern bush lawyers and hysteria caused considerable delays and changes to operations.

- Gas drainage was installed to reduce *insitu* methane levels from 5.5 m³ to less than 2.5 m³.
- Design and operating parameters were changed on the continuous miner.
- Ventilation performance was improved and controls verified.
- Mine plans depicting the types of roof that had a potential for frictional ignition were developed and distributed together with a Management Plan on the system issues.
SUMMARY

Many of the speakers today that Dr. Alan Hargraves pioneered have been active participants in the quest to understand the medium we try to make a living out of. Some are gaining an expertise in predicting or monitoring what happens. Others like me, try to manage to some small extent the operational aspects and use the geotechnical skills that have evolved over the last four decades.

There are no quick and easy answers in underground mining, and even those that play with the big Tonka toys must try to understand the geological and technical constraints. Boards of Directors and shareholders are becoming less forgiving like Mother Nature.

Geotechnical advances have progressed from simple holes in the ground with a geologist looking at drill cores, cleats and stress directional joint systems to a wide variety of techniques. Geological mapping, surface geophysics, in seam seismics, radio imaging, geophones, drilling, permeability testing, insitu stress tests along with technical data on roof, seam and floor conditions are used to fill in the missing pieces for what risks can be encountered in your mine design, equipment selection and mining methods.

Strata controls have gone from the use of timber props and steel as passive supports; to bolts, flexibolts and cable bolts with emphasis on direction of drivage, sequence and width of roadways. The changes in Australia as we get deeper will have to look at roadway shape and whether we do our drivages in or out of seam, for longer term roadway stability.

The challenges for the geotechnical experts and operators for the future, is to look outside the box so we can still get the productivities to keep costs down, but also improve operational safety and reduce business risk.

Note: The views expressed in this paper are those of the author who has the luxury of using the wisdom of hindsight. It is not intended to be critical of any person(s) involved in any decisions at Capcoal but reflect what was planned and what actually happened as our theoretical and practical knowledge improved.
TOWARDS HIGH PRODUCTIVITY UNDER A CLAYSTONE ROOF

BY PJ HAYES
TOWARDS HIGH PRODUCTIVITY UNDER A “CLAYSTONE” ROOF: GREAT NORTHERN SEAM SUPPORT MANAGEMENT EXPERIENCES AT CHAIN VALLEY, MOONEE AND WALLARAH COLLIERIES

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ABSTRACT: Coal Operations Australia Limited operates the Chain Valley, Wallarah and Moonee Collieries which are located in the Newcastle Coal Fields of New South Wales in the Catherine Hill Bay area approximately 100km north of Sydney. Operations are in the Great Northern seam, which until recent years were all conducted under an immediate roof of a strong conglomerate. Large areas of the current reserves at all operations are in areas of a “claystone” immediate roof. All three mines have attempted at various stages of their history to find an economic mining method under these claystone roof conditions, with varying degrees of success at each operation. Moonee currently operates a longwall under claystone roof while Chain Valley and Wallarah are place-changing operations which have operated under claystone with some success. The nature of the “claystone” roof horizons in the three mines is often seen as different at each operation but in practice all of the claystones exhibit similar properties – low strength layers particularly at the coal-claystone interface. Strata control in conditions where the immediate roof above the coal seam is claystone appears to be most sensitive to coal beam thickness left as roof coal, roof bolting pattern and anchorage strength, drivage standards, horizontal stress direction, frequency and type of structures present, exposure time before bolting and possibly depth of cover. Coal beam thickness should be maximised wherever possible to achieve an optimum support density. Place-change mining using deep cuts up to 15m has been successfully carried out under claystone and can be a productive mining method in this environment. There may be potential at Moonee Colliery to introduce place-changing as a method of remnant area mining or even as a gateroad development method. The most critical roof support issues in achieving high productivity under a claystone roof are the setting and maintenance of very high support installation standards and the development of a robust support management plan responsive to changes in strata conditions. This paper sets out to describe and discuss the various attempts to achieve economic success at each of the operations in a claystone roof environment.

INTRODUCTION

Coal Operations Australia Limited operates the Chain Valley, Wallarah and Moonee Collieries which are located in the Newcastle Coal Fields of New South Wales in the Catherine Hill Bay area approximately 100km north of Sydney (See Fig. 1).

The mines are currently operated as part of the Wallarah Coal Joint Venture which is a joint venture between BHP Billiton Coal Australia Limited and the Japanese company Nissho Iwai. Chain Valley Colliery commenced production in 1962 as a supplier of domestic fuel to the Vales Point Power Station. It initially mined the Wallarah seam up until the early 1990’s when it recommenced operations in the Great Northern Seam. Wallarah Colliery began mining from “E” Shaft pit at Catherine Hill Bay in 1890. Upon exhaustion of the Wallarah Seam reserves it commenced mining in the Great Northern Seam in the 1960’s. Both Chain Valley and Wallarah have always been Bord and Pillar operations using continuous miners and pillar extraction methods. Moonee Colliery commenced shortly after World War II, mining the Wallarah Seam, initially for domestic markets and later for export. It was initially a Bord and Pillar operation and when the reserves of the Wallarah Seam were exhausted, the mine moved into the underlying Great Northern Seam. This proved un-economic prior to the introduction of the longwall in late 1997, principally due to the claystone, which forms the immediate roof below the massive conglomerate strata.
Moonee operates a 90m wide DBT longwall face. The depth of cover in operation is around about 200m to the Great Northern Seam at Chain Valley and Wallarah and varies between 60 -160m at Moonee Colliery.

Nearly all of the workings in the Great Northern Seam up until recent years were conducted under an immediate roof of conglomerate. This is an extremely strong but quite variable roof, which exhibits roof rolls but requires very little roof support, with the normal roof support pattern being around two 1.8 metre bolts per 3 metre of roadway drivage.

Large areas of the current reserves at all operations are within areas that have what is called a “claystone” immediate roof above the coal seam. These are in fact a series of Volcanoclastic rocks, which are actually tuffs but are locally referred to as claystone. (Seedsman, 1992).

These types of claystones are wide spread and occur in the roof and the floor of the Great Northern Seam and tend to cause both roof and floor instability problems where they are present. The claystone, where present generally exhibits very low shear strength and may soften, fret and fall simply due to its reaction with atmospheric humidity or under the influence of the major principle horizontal stress. Normally a layer of coal between 0.5 – 1m thick is left in the immediate roof as a support. Unless sufficiently thick, this roof coal may not have sufficient strength to withstand the downward pressure from the claystone if it expands.

All three mining operations - Chain Valley, Wallarah and Moonee have attempted at various stages of their history to find an economic mining method under these claystone roof conditions, with varying degrees of success at each operation. This paper sets out to describe and discuss the various roof support management systems devised to attempt and to achieve economic success at each of the operations.

GEOLOGICAL SETTING OF “CLAYSTONE” ROOF AREAS

A generalised stratigraphic sequence of the uppermost subgroup of the Newcastle Coal Measures, the Moon Island Beach Sub-Group, together with the immediately overlying Munmorah Conglomerate Formation of the Narrabeen group is given in Fig. 2. Constituent components include coal, conglomerate, sandstone, shale and rocks of pyroclastic origin which are of variable grain size up to coarse and are termed ‘tuff’. Moon island Beach Sub-Group includes 3 major Coal Members – the Fassifern, Great Northern and Wallarah seams all of which are currently worked at various mines throughout the locality. The Great Northern Seam worked at Chain Valley, Wallarah and Moonee Collieries is a high volatile low sulfur medium ash thermal coal, which is used for power generation. The seam is up to 4m thick but is generally worked at a height of 2.7 – 3 m due to quality, and roof and floor control reasons. Overlying the Great Northern Seam coal is the Catherine Hill Bay formation. This ranges in thickness from zero, where the Great Northern Coal and Wallarah Coal merge, to over 60 m. The immediate roof over much of the seam is the Teralba conglomerate, which is up to 40 m in thickness and exhibits compressive
strength of around 45 MPa. In other areas the Booragul Tuff member forms the immediate roof of the working seam termed the “claystone” roof, the subject of this paper.

The position and extent of the “transition zone” between the tuffaceous claystone roof and the conglomerate roof is controlled by the boundary of the original river stream channels, which completely eroded the original claystone roof and deposited the thick series of conglomerates which form the roof of much of the Great Northern Seam (Edwards, 2000). The accurate mapping of this transition zone is extremely difficult from even closely spaced boreholes, as it not only weaves and moves non-linearly but also the width of the transition zone from full conglomerate roof to claystone roof greater than 2 m thick can occur over many hundreds of metres. In addition the thickness of claystone can vary from zero to several metres across the width of a bord and be accompanied by the notorious “rolls” in the conglomerate roof.

MINING HISTORY UNDER CLAYSTONE TO 1995

The first mine to be worked in the Catherine Hill Bay area was the new Wallsend Colliery, which commenced in 1873 from the outcropping Great Northern Seam in the south-end of Catherine Hill Bay. The original entries to this mine are still visible although sealed now. This mine worked the Great Northern Seam under a claystone roof. A journalist from the Sydney Mail Newspaper visited the mine in July 1875 and described the engine room which had been excavated to the full seam height and to the bottom of the conglomerate. He wrote that “the coal rose an unbroken mass to a height of 14 feet [4.3m] with an almost imperceptible scale of slate in two places, then a layer of pipeclay varying from 6 to 14 inches [15-35cm] surmounted by a seam of splint coal.” This is a reasonable description of the Great Northern Seam in that area and describes the claystone roof above the top of the seam.

In 1874 in two separate instances two men were killed from falls of top coal. In the investigation of the first fatality of James Hall on the 1st June, the inquest as reported in the Miners’ Advocate newspaper on the 13th June 1874, referred to a statement by the Government Inspector of Collieries who visited the scene after the accident. The inspector commented that the bord was about 22 ft [6.7m] wide and the working height of the coal was nearly 8 ft [2.4m]. The roof was very jointy and open and there were three rows of props set in the bord within 5 ft [1.5m] of the working place. The fall of top coal that occurred had fallen back almost to the first row of props and between two very smooth parallel facings. The mine grew to be a difficult operation due to the undercapitalisation of the mine, the very hard nature of the coal seam and the loading of the coal being subject to the ability of the ships to load at the open sea jetty at Catherine Hill Bay. For the next 80 odd years, little or no attempt was made to mine the Great Northern Seam under claystone roof. This was due to both the discovery of the Wallarah seam and it’s better roof conditions and the subsequent working of the Great Northern Seam under conglomerate roof conditions.

Chain Valley Colliery was opened in 1962 and it drove the initial headings to connect the pit bottom to the upcast shaft within a claystone roof environment. Poor roof conditions created great difficulties in reinforcing the roof with the support methods used at the time, which consisted mainly of props and cross timbers. In the first attempt all of the coal was mined and the claystone was the immediate roof. The roadways were heavily timbered and the claystone is now considerably broken. Many areas have falls up to a height of 2-3m and roadways are up to 7m wide and 3m high, showing evidence of poor horizon and width control of the bords. Inbye this area the mining...
horizon was altered leaving a coal beam of approximately 1m supported with half round bars and the roadway width was reduced to approximately 4.8m. Roof bolts were then introduced and 2 bolts (mechanical anchor point type) were installed approximately every 2m. These changes resulted in greatly improved roof stability and some of those drivages are still intact today, although many have also failed and are on the ground. There were some directional problems and conditions in the headings (driven in a N-S direction) were much better than the cut-throughs with some evidence that cut-through angle drivage had been altered with some success.

In the early 1990’s as the Wallarah seam reserves were diminishing Chain Valley decided to recommence work in the Great Northern Seam. To access coal under the conglomerate roof on the other side of the lake, mining under claystone was required. In this area of Chain Valley, workings under claystone consisted of about 1.4m of coal tops overlaid by in excess of 10m of claystone. Standards of drivage were poor in the early stages and roof horizons and mining widths were inconsistent. Later in the panel as the claystone thinned a decision was made to cut up to the conglomerate.

Both Wallarah and Moonee Collieries in the early 1990’s attempted to drive headings and extract coal in the claystone roof areas of the mines with little economic success using similar methods as in the Chain Valley experience described above. Wallarah’s NW3 panel was driven in 1992 with minimum support rules for this panel under coal tops being 2 bolts (1.2m minimum) every 2m with 6 bolts in the intersections attempting to leave around 700mm coal tops. The approach at the time was unsystematic and standards not consistently observed. Several intersections failed without apparent warning and the panel was abandoned. Similar occurrences were experienced at Moonee when the Great Northern seam was first worked in the early 1990’s.

WALLARAH COLLIERY EXPERIENCE 1999

Dwindling reserves under conglomerate roof in the Great Northern Seam prompted the attempted development of the seam under a claystone roof. The area in which the panel under claystone was attempted lies at an average depth of 130m and was called NW4 Panel, and located in the northern area of the mine. The coal seam in this area is approximately 3.4m thick and is immediately overlaid by a variable thickness of claystone up to around 4m thick. Overlying this is the sequence of massive conglomerate and sandstone up to 50m thick. A previous empirical trial in an adjacent area of the mine where a limited length of headings were driven under claystone concluded that around 1.0m thick coal beam in normal conditions was sufficient to maintain a stable roof horizon in a road way around 5.5m wide. This trial was done using an unsupported roadway that was left for several months and no roof deformation occurred. In the same area where 0.5m thick coal beam was left unsupported in a 10m wide break-away the roof fell within 24 hours of the cut being formed.

NW4 Panel was driven within the transition zone from conglomerate roof to full thickness claystone roof in the Wallarah northern area (Fig.3). Old pillar workings from the Wallarah seam (35m interburden) also overlie this area. This transition zone was unfortunately very extensive and the conditions encountered including extreme roof rolls in the claystone/conglomerate interface did not result in high productivity. In order to leave a minimum of

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1 No monitoring such as tell-tales was carried out in this panel.
800mm of roof coal the working height had to be reduced to around 2.1 – 2.2 m. This proved to be difficult and resulted in the thinning of the floor coal to the extent that the trafficability conditions were extremely poor. This coupled with the roof rolls that occurred together with the claystone roof resulted in an unproductive panel. However the roof support put in place at the time of the driving the panel, resulted in an extremely stable environment and no roof falls or deterioration in conditions have occurred in the 12 months since this section was developed. While this panel averaged only 9m/shift advance rate in what was effectively a three heading panel (see Fig.4), it did result in a proven roof support pattern for the conditions encountered at Wallarah. It also indicated that place-changing under a claystone roof could be productive if the floor horizon could be maintained in a trafficable state. In an adjacent area some years previously NE7 panel had been driven where the claystone was up to 1m thick. NE7 panel was driven to very poor standards (wide roads and inconsistent support pattern and little or no coal top left) which allowed the claystone to fret and consequent rib and roof falls occurred (see Photos 1 & 2). The contrast of these two panels demonstrates the impact that good design and standards can have on roof control.

Fig.4 NW4 PANEL WALLARAH COLLIERY

Photo 1. Wallarah NE7 panel
CHAIN VALLEY COLLIERY EXPERIENCE 1999-2000

One North West Panel

In late 1997 a panel was driven towards the upcast shaft through the “transition” zone and into an area of full claystone. The claystone thickness increased within a cut-through from 50mm to the point where it was too thick to mine and a decision was made to leave a thickness of coal roof underneath the claystone. The places were generally driven too wide and the quality of the roof horizon control was poor, and enforcement of standards was insufficient. The mine was down-sized shortly afterwards and the unit finished with only about 100 metres of drivage completed under claystone, with disappointing results.
The same section was recommenced in late 1999 to drive the panel to connect to the up cast shaft. The original headings driven were down dip and had filled with water and there were some concerns that the claystone was saturated and would cause roof support problems. In preparation for this new attempt at mining this area, extensive training and education of the new workforce was carried out. A Jeffrey continuous miner with Hydramatic bolting rigs mined 3 headings with 50m x 30m centres. The concerns with the saturated claystone were realised when the first cut through driven guttered along the middle to about a metre wide. Drivages were supported with four 1.8m AX bolts at 1m centres but this was changed in better roof conditions further inbye, to 1.5m spacings when it was evident that the roof had become competent. The seam was generally flat and the places were driven at less than 5m wide and with approximately 1200mm of coal roof using a working section of 2.2m. Headings and cut-throughs did not appear to be any different in conditions and no problems occurred in this panel with respect to cutters. As an experiment stubs were driven off the main roadways with different beam thickness and seam height, width and cut out configurations and they were left unsupported. These varied from coal beam thickness of 800mm at 4.8m wide to 1200mm at 5.5m wide and over 12 months later these unsupported stubs are still standing and showing no signs of deterioration. After a series of roof bolt tests two-thirds of the way through the panel the roof bolts spacing was increased to 2m and this proved successful. The roof above the coal beam was predominately a sandy claystone and appeared to provide reasonable anchorage for the roof bolts installed. A roof core around No. 5 cut-through was taken and showed a hard fine-grained claystone. The productivity in this 3-heading panel reached 500 t per unit shift, but an average was around 250 t per unit shift (or 15m/shift). As a miner bolter was used the productivity was quite low, but in the same conditions it is believed that place changing would certainly have achieved 1000 t per unit shift plus.

Sump Headings – 2000 Panel

This panel was formed in an area of the mine that had been blocked out and avoided for many years and was known to be under extensive thick claystone roof conditions. Place changing was used as the mining method in the panel for the first time at Chain Valley Colliery. The panel was being formed as a sump area for a new pumping system for the Wallarah and Great Northern Seams. The area was highly structured with both predominating major and minor cleats and cutters running through the area known as well as some faults projected from the overlying Wallarah seam workings. The Wallarah Seam had been worked over the top of this panel with pillars being formed but not extracted and the inter-burden being around 35m. Places were kept to 4.8m wide in the cut-throughs and 5m in the headings. The depth of cover was around 200m and the pillar size used was 24m centres. The roof support management system specified four degrees of support requirements, determined by the minimum coal beam thickness at the last bolt installed at the face and the structures present. Table 1 shows these requirements; Level 3 support diagram is shown in Fig.5.

Table 1- Summary of Chain Valley Colliery Roof support in Sump headings under claystone roof

<table>
<thead>
<tr>
<th>Coal Beam thickness, mm</th>
<th>Cut depth max, m</th>
<th>No. 1.8m bolts in row</th>
<th>Spacing between Rows, m</th>
<th>Mesh Modules (y/n?)</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt;1200</td>
<td>15</td>
<td>4</td>
<td>2.0</td>
<td>No</td>
</tr>
<tr>
<td>800-1200</td>
<td>15</td>
<td>4</td>
<td>1.5</td>
<td>No</td>
</tr>
<tr>
<td>&lt;800</td>
<td>12</td>
<td>4</td>
<td>1.2</td>
<td>No</td>
</tr>
<tr>
<td>Broken roof beam</td>
<td>6</td>
<td>4</td>
<td>1.2</td>
<td>Yes</td>
</tr>
</tbody>
</table>
Photo 4: Typical roadway at Chain Valley under claystone with 1.2m roof coal in place.
The initial drivages in the panel were very difficult due to the structures and on occasions cut-outs were limited to a few metres. A few pillars inbye the panel was widened to 3 headings and roof support conditions improved enough to accommodate a 15m cut out. In this area the best month achieved averaged 30 m/shift advance rate. The cutters were extensively mapped and it was evident that they were the key criteria relating to poor roof conditions. Some areas exhibited such intense structures that the coal beam even at 1200mm was broken and fell out like broken bricks with no mortar. However, cuts that were going to fall always fell before bolting with the mobile bolter and there were no problems after the cavity had been meshed and bolted i.e. the support issue was one of an immediate problem that had occurred within a short period of time of exposing the cut. No intersections have fallen to date although some signs of guttering in the top coal have occurred. Inbye of this area the coal beam was reduced in thickness from 1200mm to 900mm to increase the sump capacity and a second bolter was brought in to keep up with the bolting required. Fifteen metre cut outs were regularly achieved and did not cause problems. When a cut did fall it often required 2 – 3 shifts to mesh and bolt it up and this is the reason that the 2nd bolter was required. Many shifts of more than 1000 tonnes per shift were achieved the best shift being nearly 1300 tonnes (66m). This is by far the best performance achieved in the Great Northern Seam under claystone roof at any of the COAL operations using continuous miner technology. Photographs 4 & 5 show this area in Chain Valley several months after the completion of the panel. Some incipient guttering has commenced in some areas but no roof falls have occurred in any of the mine roadways and the intersections all appear to be stable.
MOONEE 1995-2000

The Great Northern Seam at Moonee is approximately 3.7m thick over the areas proposed to be mined by longwall and the depth of cover 80 m to 160m. The stratigraphy of the immediate roof includes a very weak claystone, a moderately weak claystone and an anchor coal above which is up to 30m of a massive conglomerate with strength between 15 MPa and 50 MPa. The coal of the Great Northern Seam is extremely strong coal with a laboratory strength of 25-30MPa. The stress regime in which Moonee is worked is a maximum horizontal stress of 10 MPa in NE/SW direction. The key design issues for the roof support at Moonee is the thickness of the immediate roof coal left intact and the very weak claystone forming the lower and upper coal-claystone boundaries. The claystone softens considerably soon after contact with air or moisture. When the claystone is cut by a continuous miner it exhibits pick marks for some time before becoming the characteristic pliable clay material well known to Moonee personnel (Strata Control Technology 1994, p.3). Early work by Strata Control Technology showed that the majority of movement in the area between the roof coal and the conglomerate was in the coal claystone boundary area. (Strata Control Technology, p.6).

Fig. 6 is a log of an uphole drilled from SE mains panel and shows typical immediate roof lithology used in the initial support design. The maintenance of the roof coal integrity to attempt to resist the pressures imposed by the bulking of the failed claystone, is the most critical issue in roof support design at Moonee, as the vertical stress and the road ways at the maximum depth of 160 m on initial drivage is insufficient to even cause rib softening.

Longwall extraction was however predicted to cause significant rib spall once vertical abutments exceeded the unconfined strength of the coal. At 160 m depth of cover the rib spall in the main gate area within a few metres of the face would be about 2 m and that, moderate to severe guttering, with associated centre-line cracking would occur in this area (Strata Control Technology, p.10). This has not occurred at Moonee and rib spall in the longwall face area has not been experienced. Convergence measurements demonstrated that the additional convergence added by the passing of the longwall on Maingate 2 was around 1mm that the roadways were very stable. (Strata Engineering, 1999, p. 10).
The initial and indeed subsequent roof support designs at Moonee relied on full encapsulation along the bolt length (bolted into and above the anchor coal, which is above the claystone) and a minimum 700mm coal left in place in.

Fig. 6 Moonee roof horizons

Photo 7. Shearer about to break into Maingate on LW4B at Moonee

No weight on rib

No roof deterioration
The initial and indeed subsequent roof support designs at Moonee relied on full encapsulation along the bolt length (bolted into and above the anchor coal, which is above the claystone) and a minimum 700mm coal left in place in the roof. One of the key findings from the early modelling was the most effective method of maintaining integrity of the lower coal was to increase its’ thickness and thus a minimum thickness of 700mm was specified. The role of the bolts is simply to modify the behaviour of the claystone by improving the integrity of the lower coal. The philosophy was based on the fact of the claystone does not easily lend itself to reinforcement. Many of the falls in the early history of Moonee Colliery before the longwall was commenced were in areas where top coal was much less than 700mm thick and typically less than 500mm thick. The coal roof thickness is intended to be the principle means of controlling the behaviour of the roadway and the roofbolts could be installed up to a maximum of 2m from the face with the security that the coal beam provided. The full encapsulation of the bolt particularly at the coal horizons from the collar to the back of the hole was critical to maintaining the load transfer and reducing collar loading. Many of the early falls were attributed to collar loads and bolts pulling through the immediate coal strata. SCT maintained that if the roof coal thickness was reduced to even 500mm, collar failure is highly likely even with good load transfer and full encapsulation (Strata Control Technology 1994, p.13).

Initial roof support design recommendation was using six 2.1m bolts in a W-strap and the straps placed 1.5m apart. Some failures of the coal between the straps occurred in the early part of the Moonee longwall operation which resulted in a further re-evaluation of roof support design. Initially bolting density was increased to 8 bolts per 1.5m and the coal beam thickness reduced to 600mm to facilitate a reduction in bolting length to 1.95m and mesh modules introduced together with domed plates (Strata Engineering 1999, p.7). This pattern was further changed to mesh modules with 6 bolts every 1.5m set in a 4-2 pattern eventually settled upon.

Although the initial design of the roof support system at Moonee relied heavily upon the reinforcement of the coal beam, the focus at the mine level became principally one of ensuring that the coal beam (no matter how thin or thick) was anchored at least 300mm into the conglomerate. The mine has now successfully completed five longwall blocks which have retreated along 20km of gateroads without any falls of ground in either maingate or tailgate or any problems encountered during longwall salvage operations. This is testament to the generally high operational standards of drivage and the probable overdesign of the roof support systems. The accompany photograph shows the shearer holing the maingate on Longwall 4B at a depth of cover of about 130m. Note the ribs are intact and standing vertically even though the shearer has less than 0.5m of coal to hole the gate. The mine is now driving gateroads in an area where the claystone has thickened to such an extent that anchorage in conglomerate is not possible. Fortunately the claystone above the anchor coal appears more competent and is sustaining bolt loads (Tarrant, 2001). In this area the mine is using a pattern of four 2.1m bolts at 0.75m spacing on 1m mesh modules.

**DISCUSSION**

The nature of the “claystone” roof horizons in the three mines is often seen as different at each operation. Chain Valley claystone is seen as more competent than Wallarah or Moonee. Moonee is often cited as the weakest claystone. In practice all of the claystones exhibit similar properties – low strength layers particularly at the coal-claystone interface. At Wallarah Colliery in the same area that could not be supported years before and the claystone fretted away to a soft puggy material, a new panel was driven which has not deteriorated at all 12 months afterwards, thus demonstrating the benefits of high standards of drivage and a systematic approach. Chain Valley established that when working under claystone roof using place-changing technology up to 15m cuts could be driven successfully. The mining system productivity in a 6-heading layout was sufficient to be financially viable even in areas with a moderate degree of structure provided that floor trafficability could be maintained.

Strata control in conditions where the immediate roof above the coal seam is claystone appears to be most sensitive to the following, some of which are interdependent:

- **Coal beam thickness left as roof coal.** In good conditions at Wallarah, Chain Valley (and probably Moonee) a coal beam thickness of 1m resulted in good drivage conditions that enabled a 15m cut.
- **Roof bolting pattern & anchorage strength.** A lower coal beam thickness was adequately compensated for at all mines by increasing the support density. Conversely a thicker coal beam enabled support density to be reduced successfully at both Wallarah and Chain Valley. The ability of the upper claystone to provide sufficient anchorage for bolting is critical if there is no conglomerate in which to anchor the bolts.

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2 The indications at Moonee are scant because the ability to leave a coal beam thicker than 700mm is constrained by equipment height considerations. However an unsupported stub heading was driven in MG6 which left 1m of coal roof intact inbye of a cut-through which required secondary support and the stub remained stable for many months.
• Drivage standard. The older panels under claystone at both Wallarah and Chain Valley failed at least in part because of poor alignment of drivages and poor width control.
• Horizontal stress direction. As would be expected, drivages perpendicular to the major principal horizontal stress tended to fare worst.
• Frequency and type of structures present. Increasing coal beam thickness to even 1.2m did not prevent coal roof falls freshly-driven cuts in highly structured areas of Chain Valley.
• Exposure time before bolting. Face falls even in structured ground are very rare at Moonee which does not expose more than about 2m of roof at one time. Falls of top coal in cuts, where often 10-15m of roof was exposed at one time, occurred at Chain Valley in highly structured areas. However no falls occurred after the cuts were bolted, even though this was a higher stress environment than Moonee because of a greater depth of cover and the superimposition of the Wallarah seam workings above.
• Depth of cover. The conditions in the claystone drivages at Wallarah Colliery at 130m depth show little sign of deterioration after 12 months whereas Chain Valley’s drivages (using the same equipment and mining method) appear to be showing some early signs of guttering in some headings. The difference may in part at least be due to depth issues.

At Wallarah, Chain Valley and Moonee, the intersections under claystone appear to be often more stable than the headings. This is particularly evident at Chain Valley where 4-way intersections show no signs of deterioration but headings leading into them show early signs of guttering. At Moonee there are several outbye lightly-supported intersections on the belt road where the cut-throughs have fallen right up to the intersections on both sides and the intersections show no sign of deterioration. It is unclear as to the reasons for this.

CONCLUSIONS

1. Working the Great Northern Seam under claystone requires good risk management practices.
2. There may be minor differences in the claystones between Chain Valley, Moonee and Wallarah, even indeed within the same mine, but all exhibit low strength properties especially in the contact area with the coal seam. They appear to become more competent the higher up in the strata.
3. Coal beam thickness should be maximised wherever possible to achieve an optimum support density.
4. The influence of structures on the roof support requirements is immense. Structures can ultimately result in the need to “stringbag” the roof in order to maintain stability.
5. Place-change mining using deep cuts up to 15m has been successfully carried out at Chain Valley Colliery under claystone and can be a productive mining method in this environment. Place-changing under claystone at Wallarah Colliery has the potential for similar or even higher productivity levels provided the floor trafficability issue is addressed successfully.
6. Depth of cover may prove to be a limiting factor in achieving high productivity under claystone.
7. There may be potential at Moonee Colliery to introduce place-changing as a method of remnant area mining or even as a gateroad development method.
8. The most critical roof support issues in achieving high productivity under a claystone roof are the setting and maintenance of very high support installation standards and the development of a robust support management plan responsive to changes in strata conditions.

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LONGWALLING AND ITS IMPACTS IN THE SOUTHERN COALFIELD - RECENT BHP EXPERIENCES

BY P EADE AND J WOOD
LONGWALLING AND ITS IMPACTS IN THE SOUTHERN COALFIELD – RECENT BHP EXPERIENCES

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ABSTRACT: Illawarra Coal BHP Minerals currently operate five underground coal mines in the Southern Coalfield of New South Wales in Australia. Each mine operates a retreating longwall extraction system as the primary production process. Economic reserves at several of the mines are nearing exhaustion and a feasibility study is under way to maintain production levels by development of a new mine. The level of maturity of the current operations and the nature of the remaining coal reserves in the Southern Coalfield present significant challenges for continued viable longwall mining. A general overview of the challenges and recent BHP experiences in relation to a number of more specific issues are presented in this paper.

INTRODUCTION

The Southern Coalfield, in which the Illawarra Coal BHP Minerals underground mines operate, produced almost 11.8 million tonnes in the 1999-00 year of raw coal of which 10.4 million tonnes was saleable. This production rate has been in steady decline over at least the last 10 years (JCB Statistics, 2000). Two Southern Coalfield mines, Brimstone and Avon, ceased production during the 2000 year. This leaves only eight remaining coal mines currently in production.

Illawarra Coal BHP Minerals operate five of the remaining underground coal mines, viz. Appin, Tower, West Cliff, Cordeaux and Elouera. Cordeaux Mine is due to cease production in March 2001 and will be continued only on a care and maintenance basis until its ultimate future is decided. The feasibility for a new mine, Dendrobium, is well under way, with the aim of replacing dwindling Elouera Mine production and reserves by the 2004 year.

The relative age of the remaining mines logically means that the higher quality and easier access reserves have already been mined. As Bulli seam workings progress westward from coastal outcrop escarpments, depth of cover to reserves generally increases and mining conditions become more challenging.

Challenges facing longwall mining in the Southern Coalfield can be broadly slotted into the following categories:

- Safety and Legislation
- Economic viability
- Environment and Community
- Technical issues
- Surface Effects.

SAFETY AND LEGISLATION

Illawarra Coal recognise the imperative of constant improvement in safety performance. We have come a long way but can not afford to relax our efforts.

There has been a general move away from prescriptive legislation to more of an Occupational Health and Safety style legislation, which tends to be organisation and systems based. Risk Assessment principles are now widely used to identify potential risks and hazards, an assessment is made of their consequence and likelihood and
appropriate control measures developed. Critical safety risks have been identified and appropriate Management Plans developed to control them to an acceptable risk.

The New South Wales Coal Mines Regulation Act is currently under review and is likely in the near future to change significantly. A greater utilisation of standards, guidelines and management plans is expected.

Illawarra Coal continues to improve our Safety Culture through workforce participation, commitment and involvement. The systems and Management Plans in place must be robust and regularly reviewed to ensure they are appropriate to control hazards and to cope with continual change.

**ECONOMIC VIABILITY**

When compared to other Australian longwall mines, Illawarra Coal operations productivity is less than half that of a number of our competitors. This is also associated with relatively higher costs of production.

The significant reasons, and hence major challenges to be overcome, include:

- the mines are generally older as is the associated infrastructure,
- mining and coal clearance systems tend to be lower capacity and to some degree outdated,
- the mines are relatively deep with associated higher stress and gas regimes,
- due to reducing throughput to export from the Southern Coalfield, unit costs are increasing, and
- due in part to the age of our workforce, work practices have generally been slow to change.

Export coal prices have been in steady decline over many years. Between the periods 1990-91 and 1999-00 the average FOB price has fallen from $A 64.25/t to $A 51.51/t, a reduction of almost 20% (JCB Statistics, 2000).

Illawarra Coal has seen considerable change in recent years and much effort has been expended in addressing the challenges to our economic viability. Workforce cooperation has allowed greater flexibility, less demarcation and a general acceptance of external labour and contractors where it is efficient and cost effective to do so. To remain viable, Illawarra Coal mines have been forced to shed a significant proportion of its workforce over recent years.

**ENVIRONMENT AND COMMUNITY**

The expectation of the community and the environment in which we operate is of ever reducing impacts resulting from mining activities. The approval process and conditions of approval to extract coal by the longwall process reflect the less tolerant community attitudes. The tendency to wider longwall faces to improve mine productivity is sometimes at odds with a reduction in mining activity-induced impact.

Subsidence impacts and surface effects of longwall mining will be discussed later in more detail. Illawarra Coal has faced and continues to face significant challenges in controlling subsidence impacts on natural features such as watercourses, cliff lines and river valleys as well as man made features such as road bridges, water supply tunnels, canals, aqueducts, dams and pipelines.

Environmental licensing conditions are tending toward the ideal of minimal discharge of pollutants with ever increasing restrictions on pollutant type and quantity limits. Load based licensing, introduced into general industry, will soon become a reality for coal mining operations with no doubt added cost and restriction to the business. Coal mines will need to become even more innovative in such areas as materials reuse and recycling, solid and liquid waste production and disposal, dust, noise and gas emissions.

Illawarra Coal, at a number of sites, is trailing waste water injection into relaxed strata above worked out longwall areas. Injection is via boreholes drilled from the surface to intersect the relaxed strata above the goaf areas. The quality of the waste water that is injected is better than the water already contained within the targeted zones and boreholes are strategically lined and grouted to prevent aquifer mixing from multiple horizons.

Disposal of the refuse from coal washing operations is an issue for longwall mines both from the visual and land use aspect of the waste material as well as the necessary emplacement activities. Waste material transport and emplacement potentially have traffic, noise and dust control implications which must be controlled to community expectations.
Illawarra Coal operates mines in the Bulli Seam which is considered a gassy seam by world standards. To enhance safety three of the mines practice methane drainage via in-seam, cross measure and goaf drainage drilling and collection. The predominant gas present within the coal seam being mined and those coal seams adjacent is methane, although carbon dioxide does become prominent in localised zones. Partial recovery of costs associated with the methane drainage systems is accomplished by utilisation of the drained methane to generate electricity which is subsequently sold to an electricity distributor. In general terms, as the width of longwall blocks increase to improve productivity, the mining influence is expanded and the potential for methane to impact on operations is increased. There is also some evidence that strata above the Bulli seam may contain gas that has the potential to impact mining operations. This aspect is currently being actively investigated.

Methane and carbon dioxide are both considered to be gasses which contribute to the greenhouse effect. The longwall mining process liberates gas that is present in the coal seam being mined as well as a proportion of the gas that is present in adjacent coal seams and strata. The greenhouse effect can be reduced to some degree by collecting gas into a drainage system and using the drained gas as a fuel to generate electricity. Approximately half of the gas emissions from three of the Illawarra Coal Mines is collected and utilised in this manner. Burning the methane to produce electrical energy and carbon dioxide reduces the greenhouse effect. Carbon dioxide has a much lower greenhouse effect than methane, with additional greenhouse gas reduction being achieved by displacing the need to generate a quantity of electricity from alternative fuel sources. The major challenges here are to increase mine gas capture and to improve the energy conversion efficiency of the gas utilised.

Environment and community interests demand improved performance. The BHP Charter states that we are successful when the communities in which we operate value our citizenship and that BHP values an overriding commitment to safety and environmental responsibility. Success will only come with higher levels of community involvement in longwall approval processes and, along with other stakeholders, early input into project definition and planning. Communication with community, councils and statutory authorities needs to be ongoing and at an increased level.

TECHNICAL ISSUES

Exploration

The longwall mining process can be a highly productive coal extraction method. However the system is somewhat inflexible compared to other forms of extraction. For optimum productivity, blocks to be mined by longwall should be as large as practical and free of geological features such as faulting and igneous intrusions. An unplanned relocation of a longwall panel can be both difficult and costly. Exploration is the major tool available to define suitable longwall domains and to reduce the risk of unexpected longwall interruptions due to unpredicted features.

Exploration in the Southern Coalfield is not without its challenges. Surface access is often difficult due to steep topography and much of the surface is administered by Sydney Catchment Authority (SCA) and National Parks and Wildlife Service (NPWS). The SCA administered area includes water catchment and a number of major water storage reservoirs which are integral to the Sydney water supply system. There are obviously restrictions and tight conditions placed on accessing these areas for exploration. The population growth and expansion to the south and southwest of the Sydney area also presents access issues and restrictions over exploration activities.

The deep reserves over 500 m and presence of significant sandstone elements in the sequence leads to relatively high drilling costs associated with surface drilling.

Geophysical and electromagnetic surface based exploration techniques have limited success on the Southern Coalfield. Sill and dyke material is often non-magnetic and/or weathered. Location of surface power lines and the presence of basalt aggregate along the partially built Maldon Dombarton railway line route can make interpretation difficult.

High resolution surface reflection seismic is relatively successful in locating half to full seam dislocations and indicating coal seam continuity. Gas and water present in strata above the coal seam horizons may reduce integrity of data collection and complicate analysis in some areas.

In the mines employing gas drainage, underground in seam drilling doubles as a valuable exploration tool. The drilling and pre-drainage of a longwall block will normally reveal any anomalies within the block but the timing may not be sufficient to prevent a major production dislocation in all cases. In seam longholes can be used to prove the existence or otherwise of targeted features.
Illawarra Coal holds a number of mining exploration licences in the western portion of the Southern Coalfield. The resource in this area is indicated, in some parts, to have a depth of cover in excess of 700 m. Portion of this licence area contains an overlapping petroleum title which is held by a coal bed methane explorer who is assessing production potential from a number of surface wells drilled and fracced in the Bulli Seam. There will likely be future challenges to be faced during exploration and production in an area occupied by a coal resource company and a gas resource company who are separate entities.

In parts of the Southern Coalfield coal reserves are present in a number of coal seams, with the overlying seam having been mined out, in some cases, many years ago. Exploration of the lower coal seams presents a challenge, particularly when mining records of historical mining are incomplete.

**Outburst Risk**

The Bulli Seam in the Southern Coalfield is known to have some propensity to outburst. Outburst prevention is currently dependent upon reducing gas content within the seam being mined to below a threshold limit. The limit is based upon history, composition of the seam gas and ultimately the volume of gas calculated to be in a cubic metre of coal.

Until relatively recent times threshold limits have been restricted to development mining. However in the last couple of years evidence of several outburst events has been associated with longwall mining.

A major challenge exists to develop a better understanding of the outburst phenomenon. Significant research has been applied to this issue in the past and is continuing, seeking a fundamental understanding of the mechanism which results in an outburst occurrence.

**Depth of Reserves**

As indicated above, Illawarra Coal hold mining exploration licences over resources that are indicated to have a depth of cover ranging to over 700 m. Apart from one mine in the Sydney Basin, which had limited operation, this indicated depth to potential reserves will be deeper than any other in Australia.

Longwall mining in these conditions will no doubt present a new set of challenges. Roof support design and ratings, expected stress regimes, and gas characteristics will need to be accurately predicted to ensure longwall mining in this environment is successful.

**SURFACE EFFECTS**

Almost all of the current coal extraction is under a surface veneer (to 150 m in thickness) of Hawkesbury Sandstone. Minor thin Wianamatta Shale covers parts of hill tops and ridges in the current Appin and Tower Colliery extraction areas. The Wianamatta Shale supports a thick fertile soil profile, while the Hawkesbury Sandstone is usually overlain by a poor, thin sandy soil. The Hawkesbury Sandstone is an interbedded massive to thin bedded unit with crossbedded units, shale beds and lenses. This unit is moderately jointed, with some major joints penetrating a series of individual beds. The surface expression of this unit varies from competent to deeply weathered.

**Geomorphology**

The surface is composed of gently rolling plateau area that is deeply dissected by major watercourses. The Cataract and Nepean Rivers are enclosed in gorges with sub vertical cliff lines to 60 metres in height. The rivers follow a zigzag course with major changes in direction aligned sub parallel to the prominent local joint direction. In most cases the cliff lines are steeper on the concave side of the river valley.

Cliff lines are typically sub vertical, or composed of a series of sub vertical steps, and are typically undercut into cavernous zones some five metres in depth. These cavernous zones are typically associated with cross bedded or thinly bedded quartzose sandstones with water seepages from the bedding planes.

**Mechanisms**
The typical classical maximum subsidence parameters associated with supercritical extraction include some 1.1 metre of vertical subsidence, tilt of less than 8 mm/m and strains of less than 2 mm/m. Superimposed on these classical subsidence parameters is the concept of horizontal movement. This mechanism was recognised and documented as a result of precise surveying while mining under and adjacent to significant surface structures.

Horizontal movements in excess of 5 mm (GPS accuracy) are recorded some 5 km from the extraction. Movements are towards gorges or surface notches. These movements increase with proximity to extraction areas and gorges.

The largest horizontal movements are a direct result of horizontal compression induced by mining in the vicinity of deeply dissected areas. Horizontal compression is concentrated in the base of gorges, mining induced movements promote delamination of strata reducing strength and result in horizontal shearing in the base of the topographic lows. This mechanism results in closure of the walls, and hence crests, of the gorge cliff lines and relative upside (when compared with subsidence predicted for flat topography) of both the base of the gorge and, to a lesser extent, the immediate shoulders of the gorge. The major planes of relative horizontal movement are located in the strata immediately below the base of the gorge.

**Surface Infrastructure**

Economic coal extraction at both Appin and Tower Collieries required mining under, or close to, major surface infrastructure including:

*Twin Bridges – SH2 Hume Highway crossing the Nepean River at Douglas Park*

Each bridge consists of two carriageways and the decks are 14 m in width, the maximum deck length (Northbound Bridge) is 286 metres and the maximum pier height is 34 m.

*Cataract Tunnel*

This tunnel connects the weir at Broughtons Pass to the Upper Canal, has a gradient of 0.67m per km (0.067%) and a capacity of 704 megalitres per day.

*Upper Canal and Aqueducts*

The canal was excavated, where possible from solid rock. Excavated material was used to construct masonry walls in soft earth. In shallow cuttings or bad ground concrete walls or cemented rubble was used. Two aqueducts at Elladale and Simpsons Creek are 8 feet [2.4 m] in diameter wrought iron structures of lengths 655 [199.6 m] and 150 feet [45.7 m] respectively, are multispan structures with spans up to 60 feet [18.3 m].

*Broughton’s Pass Weir*

This weir diverts water from the Nepean Tunnel and the Cataract Reservoir into the Upper Canal System via the Cataract Tunnel. It also supplies water via the low lift and high lift pumping stations to the Macarthur Water Treatment Plant. The weir is a concrete gravity structure, which was built around 1885 as part of the Nepean System. The dam has a nominal storage capacity of some 50 megalitres. The Low Lift Pumping Station, the High Lift Pumping Station and the rising mains were built as part of the Macarthur Water Quality Project during the period 1994 – 1995.

*Gas pipelines*

The AGL natural gas pipeline, ethane pipeline and Eastern Gas Pipeline traverse the area of Appin Longwalls 403 to 407 and also cross under the Cataract River and Simpsons and Elladale Creeks. The pipelines consist of fully welded steel pipes laid in the ground with a minimum cover of 0.8 m.

*Others*

- Transmission lines, telephone lines, roads and residential structures
- Numerous residential and utility structures are undermined throughout the area.

**Natural Features**

*Cliff Line Stability*
A total of nine rock falls were identified during extraction at Appin and Tower Collieries during undermining of the Cataract and Nepean Rivers. The observed rock falls were small in size (some 50t average) and were all associated with significant chemical weathering, erosion and overhangs. The majority occurred in the middle third of cliff height.

Because of the natural instability of cliff lines that are moderately well jointed and contain significant chemical weathering, the prediction of areas with the potential to fail during the mining process is difficult. It is significant that only a small proportion of potentially unstable cliff line has exhibited failure.

No evidence of massive slope failure (tension cracking behind the slope of slip planes for the full height of the cliff line) was observed.

**Water Loss and Cracking of the River Bed**

Localised water loss was recorded from the Cataract River during the extraction of Tower Colliery Longwalls 8 – 14. This water loss was restricted to pools behind rock bars. Horizontal fracturing consistent with failure due to horizontal stress accumulations was observed in rock bars. Water lost from the pools surfaced further downstream at the extremity of cracking or where the river gradient flattened. No water loss was recorded when undermining areas of permanent water.

A shallow grout curtain was installed from vertical holes in one rock shelf. This curtain proved successful in reducing the rate of leakage from the pool located immediately upstream.

No water ingress was recorded in the underground workings. This agreed with the conclusions of numerical modelling.

There is evidence that natural siltation processes will successfully reduce leakage with time.

**Gas Emission**

Areas of gas emission into water were recorded from the Cataract and Nepean Rivers subsequent to longwall extraction. This emission was typically observed as a number of bubble streams in localised areas within the main stream or from under rock ledges. Two significant gas emissions, with sufficient gas flow to ignite, were recorded. These more substantial gas emissions persisted for some 6 months after undermining.

The gas chemistry confirmed that the major component was methane. Higher hydrocarbons were detected while the gas had a petroliferous odour similar to that of the Bulgo Sandstone gas. Atmospheric gas measurements indicated a total gas emission for the Cataract River to Longwall 14 of some 20 L/s.

**Vegetation Effects**

Four localised areas of vegetation stress were identified during the extraction of Tower longwalls 10, 14 and 15. These areas were located in relatively deep sandy soils between the main river channel and the cliff line.

Field investigations concluded that the temperature of the soil increased to excess of 40°C and that the temperature peaked at a carbonaceous clay horizon some 40 cm from the soil surface. Laboratory analysis indicated that the temperature increase was due to bacterial degradation of hydrocarbon gas migrating through the soil profile. A combination of anoxic conditions, high temperature and dehydration resulted in the death of deeply rooted vegetation in the very localised areas associated with this gas emission. When gas migration subsided, self revegetation of the areas was soon evident. No vegetation effects were recorded from the lower reaches of the Cataract River or the Nepean River.
CONCLUSIONS

Many of the issues discussed above are complex and interact with each other. To manage effectively we need to understand their causes, mechanisms and characteristics. Management needs to be able to predict outcomes more accurately and this can only be achieved by higher level of understanding.

Communication and interaction with potential stakeholders who may be affected by mining operations is of paramount importance. It needs to be proactive and ongoing. Stakeholders include outside bodies and entities and include surface infrastructure owners, landholders, community interest bodies and government regulatory bodies.

Management Plans are an effective way of managing and controlling specific issues associated with the mining effects. They provide the rigour and discipline in the components of planning, observation, measurement, reaction and review. Management Plans are developed with significant stakeholder input and knowledge and provide a transparency to the process and control of an issue.

The Southern Coalfield in general and Illawarra Coal in particular face a challenging future. Longwall mining will likely continue to be the extraction method of choice, with widths increasing in an effort to address economic viability demands. Control of the environment in which we operate and of the impacts of underground longwall mining is essential to maintain acceptable productivity levels and the licence to operate.

REFERENCE

SUCCESSES AND FUTURE CHALLENGES FOR LONGWALL MINING IN STRUCTURED GROUND

BY I STONE
SUCCESSES AND FUTURE CHALLENGES FOR LONGWALL MINING IN STRUCTURED GROUND

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ABSTRACT: The experiences from the recent 3 years at Springvale Mine have highlighted once again the importance of understanding the geological structure at a mine site. Springvale Mine previously had a history of difficult roof conditions, although not clearly linked to geological structure. From mid 1998 a series of complex strike slip fault structures were encountered, one set running with and along the commencement of the 404 Panel gateroads. The unexpected faulting, its critical location, and unknown extent had a serious impact on panel development and business performance. Even the most carefully constructed business plan and budget is quickly damaged by an unwelcome geological surprise.

INTRODUCTION

The experiences contained in this paper are focussed on Springvale Mine. In doing so the uniqueness of the Springvale site is stressed. Particular geological interpretations, methods used, geotechnical analysis and roof support systems that have been used with success at Springvale may not be appropriate elsewhere. Each mine has its own “geological grain” and geotechnical character. Before dismissing the Springvale methods as site specific, they are worth checking for a broader application at other mine sites.

Geological structure at Springvale impacted in a number of ways:
♦ physical weakening of the strata caused by fracturing
♦ changes to the stress direction and an increased stress magnitude
♦ dramatically increased strata support requirements for both development and longwall roadways
♦ different mining systems to support the ground
♦ required radical changes to the panel layout to keep the panel moving forward

This paper is not intended to provide a description of the Springvale geology. Rather it attempts to look at the nature of the issues encountered with geological faulting and the types of approaches taken to plan for mining future fault zones. Springvale mines the basal 3m of the combined Lithgow – Lidsdale seams at a depth of 260m – 380m.

FOUR PRINCIPLES FOR SUCCESS IN STRUCTURED GROUND AT SPRINGVALE

To the extent which it is possible an attempt is made to encapsulate 4 principles used at Springvale which might be transferable to other sites. In any case the Springvale experience will be explained within those 4 categories as they were the areas within which work concentrated.

1. Reliable geological structural model for the mine site. To be comfortable that the types of faults, their most likely orientation, continuity, frequency and the mining impact, for both development and longwall, is understood to, say an 80% reliability.
2. Reliable geotechnical understanding of typical mining conditions. The ability of a mine site to extend a geotechnical understanding to variable fault affected conditions will depend on knowing zone locations.
3. Appropriate strata control hardware or methods to support both development and longwall extraction.
4. Efficient strata control installation systems, which include equipment, systems, manning and specialist teams.

Three other strategies important to success were:
♦ allowing the site personnel to have responsibility and authority for their area of work. Success would depend on developing skills to understand and “own” the problem. In 1997 Springvale
corporate management, in the midst of a number of unresolved geotechnical based issues, gathered together a group of some 20+ industry experts to comment or recommend. You will not be surprised to learn that no unity of thought emerged. The subsequent operational strata control methods were ultimately unsustainable. A consequence of this process was the very limited control or ownership of solutions by site personnel.

- a clear understanding of the business risk associated with geological structure. Budgets and mine plans need to be developed with realistic time and cost allowances. For example, de-rate advance rates using historical information.

- focus on returning balance to the development progress of Mains Panel and the Gateroad Panel. The lack of appropriate Mains development did not allow access for the required inspection, and exploration for future mine and business planning (not to mention the level of inventory required to cope with any future poor mining conditions)

**STRUCTURAL GEOLOGY**

**Background**

At the time of unexpectedly mining into a complex N – S strike slip fault zone in mid 1998 it was difficult to forecast fault location because:

- the structural model of the western coalfield, based essentially on lineament zones, following rigorous work by John Shepherd and associates, (and to some undefined extent on the presence of swilleys) did not fit. From Springvale’s viewpoint the lineament model was not mine site specific to the extent it provided the likelihood of fault location, intensity, type and extent. It was a zone, of variable width, which contained “slackey rolls” – seemingly a catchall term for very soft faulted ground including complex thrust structures.

- Springvale had not intersected any previous significant faulting, and in particular none which was associated with high horizontal stresses – providing extreme mining conditions.

- It was likely that these structures have very little displacement (< 1m and typically less than 0.3m)

Springvale’s working roof is approximately 4m of coal with inter-bedded claystone units. Within this unit:

- faulting can fracture the coal into fine “sugar coal” which has no inherent strength; and

- faults use the claystone to extend laterally.

Fig. 1 shows, as example, the location of the main N – S fault zone at the start of the 404 Panel. The development of 404 panel was delayed by approximately 3 months because of the unexpected fault zone and its difficult mining conditions. To progress the panel and meet production contract commitments, some key roadways were driven around the faulting. Roadways were driven into both longwall blocks 404 and 405, with obvious longer term cost consequences.

Initially some inseam drilling was used to navigate roadways around the immediate structure zone. The continuity of the structure northward remained uncertain with the existing information base.
Fig. 1 Faults mapped in the 404 and 405 Panel area
Geological Model

An alternative structural model was developed with assistance from SRK who bought a high level of structural geology skills and a fresh interpretative approach. The Springvale structural geology framework was considered with respect to its place in the broader framework of southeastern NSW. Looking successively at the different scales of regional, district and ultimately mine site gave SRK a likely tectonic model in which to interpret Springvale.

For the detailed mine site focus SRK used an existing set of aeromagnetic data which had been flown as a collaborative exercise between mines in the Western Coalfield. Aeromagnetic data proved effective at Springvale because:
- the Lithgow seam was stratigraphically within 100m above the older basement rocks of the folded and deformed Lachlan Foldbelt.
- The structural grain of the older basement rocks appears to be related to structures within the Lithgow Seam.
- Aeromagnetics effectively picks up the basement structures.

The revised Springvale structural model was based essentially, but not entirely, on the following basement trends
- importance of the N – S trending structural grain of the basement;
- recognition of a set of regional NW - SE structural zones.

Both of these structural trends have been found in Springvale mine (Fig. 2). In addition the structural interpretation provided a hierarchy of severity of the different faults – with areas adjacent to the intersection of the N – S and the NW – SE zones considered to provide the most severe conditions (e.g. in Mains Panel just inbye 405 Panel).

Impact of the Geological Model

The revised geological model has proved to be invaluable to the development of robust mine plans and preferred layout of gateroads to avoid structure. It was able to provide a level of confidence to operations staff and mine owners of the fault zone locations, and that unexpected structures were unlikely. The presence or absence of structures projected by the model are proving to be of the order of 80%+ accurate (Fig. 2).

Some consequences of the model are:
- exploration using in seam drilling to confirm structures is routine;
- the level of detail in the Springvale structural model is remarkably high. Some zones with little faulting, and little development effect, provide noticeably poorer longwall (LW )extraction conditions;
- significant structures are recognised in the mine plan by appropriate de-rating of development or LW extraction rates;
- appropriate roof support can be planned ahead;
- the severity of different structural zone directions vary;
- the intersection of N-S with NW – SE zones has so far produced the worst conditions;
- some projected zones (NW-SE) show little faulting but have higher stress;
- by observation, stress fields around fault zones may increase in magnitude and rotate from E-W to N-S;
- detailed mapping of structures and mining conditions to confirm and refine the severity of differently oriented zones, including their intersection, is ongoing. This includes the status/impact of the long recognised surface lineaments.
Fig. 2 Projected and mapped structure zones
GEOTECHNICAL UNDERSTANDING

Springvale had a history of poor roof conditions, for example, difficult sections of gateroad during LW extraction, and time dependent development roadway failure.

In response a considerable body of geotechnical work accumulated, which identified the mechanism of roof failure in typical Springvale conditions. These issues included:

- relatively weak ribs which allowed rib failure to increase the roof span and failure to propagate above the installed support;
- the location of the Mains beneath a series of “valleys” which caused a relative increase in the usually E-W trending stress field and poorer NS roadways. The development strata conditions under the deeper plateau areas (320 to 380m) were typically stable;
- a dominantly E-W stress field which was magnified on the MG corner of the retreating longwall. Areas toward the end of the LW were notoriously difficult, the maingate was heavier approaching cutthroughs and certain maingate areas had very significant floor heave;
- various technical assessments identified the part played by the presence of softer wet clay bands within the roof bolting horizon.

The resultant was a record of very heavy LW maingate, areas of poor tailagtes and isolated roof falls on development. One outbye fall in the development panel stopped the panel for a period of 3 months. In addition the NS roadways in the Mains deteriorated over time; many became unusable due to falls, extensive passive support was installed in others.

Valley Effects

Analysis of roof monitoring data by Strata Engineering clearly linked the onset of the “valley” stress effects in N-S roadways to time dependent failure of the particular strata at the 4 to 6m in the roof. Installation of secondary support within 4 weeks of mining would normally prevent an acceleration of roof displacement.

Longwall Gateroads

The reasons for poorer stress induced maingate conditions near the end of the longwall block were understood. Unexpected maingate roof falls elsewhere where not understood. Before the event it was not visually possible to differentiate these fall areas from normal gateroad conditions.

Consequently the LW403 the gateroad was extensively supported with a range of secondary tendon types. The tendons were installed in parallel with development. This approach was driven by:

- lack of geotechnical understanding of the issue;
- need to secure the business from unexpected production loss.

The ability of site personnel to target the maingate areas at most risk of roof failure was linked to the development of the structural model. For example, the N-W trending structure zones, which may have had little or no expression during development, usually became areas of higher roof displacements. In addition acceleration of roof displacement would occur much further ahead of the face than the usual 10 to 20m, (Fig. 3). While not yet fully understood the structure zone is likely to have a higher stress which mobilises during longwall extraction.
Understanding the Fault Zones

Geotechnical understanding of the more variable fault zones is based on grouping the conditions into four types. The weaker ground around faults includes the seam, its clay bands and the floor strata. Higher stress levels can be relieved on many horizons.

<table>
<thead>
<tr>
<th>Fault Characteristic</th>
<th>Geotechnical Characteristic</th>
</tr>
</thead>
<tbody>
<tr>
<td>Structured ground with minor faults</td>
<td>Common, not necessarily near severe fault zones, usually have little impact but are bolted with long tendons. Usually associated with increased stress in MG in vicinity of LW face. Secondary support here necessary</td>
</tr>
<tr>
<td>Increased stress</td>
<td>Usually in the vicinity (&lt;100m) from more severe faulting, usually where 2 fault zones intersect. Stress often rotated to N-S, and causes guttering on development. Essential to bolt with long tendons at time of mining.</td>
</tr>
<tr>
<td>Fault zone (no stress)</td>
<td>Are generally stable with a routine amount of bolting, and have soft roof only at the fault zone.</td>
</tr>
<tr>
<td>Fault Zone with high stress</td>
<td>Very severe mining conditions. Recommend spiling and arches. Have used cable bolts, square steel sets and PUR etc without stabilising.</td>
</tr>
</tbody>
</table>
Monitoring and the SMP

Fundamental to the development of any consistent geotechnical understanding is the availability of good data. At Springvale monitoring was used to both understand the changing geological conditions and act as a warning of unstable roadways. Extensive use of extensometers and tell – tales at all intersections and along key or at risk roadways was routine. Up to 80 monitoring points might be active at any time, including outbye stations.

A systematic collection of data was supervised by Technical staff who also had developed a comprehensive data management and warning package. Roadway sites which had exceeded predetermined levels of roof movement were listed for inspection, resupport or further close monitoring.

The monitoring was part of a Strata Management Plan which had monitoring, inspection and remedial work as its core. It was managed by a system of clear documentation and authority. A monthly Strata Management Plan meeting was important to maintain the link between the needs of the Operations employees and the direction of the Technical employees.

**DEVELOP STRATA SUPPORT HARDWARE**

**Background**

If so much was known of the geotechnical behaviour why then did the mine have so many roof support issues? The answer to this was the understanding probably developed quicker than the appropriate support systems. Ideally Springvale needed a secondary support system which could be pre-tensioned and fully grouted. Installation would need to be within weeks of development, or sooner in poorer ground. Installation processes which could be at least partly in parallel with the continuous miner would reduce the impact on development rate.

Springvale over time have tried just about all types of secondary support systems available to the industry. Initially the fully grouted truss and cable bolt systems had mixed success as they were probably placed too late to provide the expected results. Consequently passive support was extensively used in outbye roadways.

Analysis of the roof behaviour in “valley” conditions required early placement of secondary support for the long term support of north – south roadways. Therefore the mining cycle also included a secondary bolting phase in the valley areas. Different point anchored tendon truss systems were used to improve the mining and secondary support cycle time. This point anchor design was flawed for the conditions but it was a step toward placing long tendons within 2 – 4 weeks after mining.

In the period mid 1997 to mid 1998 significant trialling of new systems was achieved. The value of pre-tensioned long tendons placed soon after mining was recognised from monitoring data. Valley north – south roadways and areas of higher stress were targeted. Various single long tendon support designs were used during the same period because of the different capability and design of the secondary bolting rigs. These tendons were all essentially point anchored, except for the 8m tendons - largely encapsulated by 2 or 3 slow and extra extra slow set resin anchors.

Different bolting rigs would likely be installing different tendon supports, which made supplying bolting hardware, resin anchors and the different sized bits more complex than desirable.

Probably the most useful improvement was during this period was developed by a continuous miner crew who discarded the nut on the flexibolts for the barrel and wedge. This allowed superior pretension to be installed at the face and reduced the range of tendons and accessories.

By mid 1998 Springvale had available an assortment of long tendons and bolting rigs. In essence long tendons installed at the face were 4m to 6m long point anchored. The more fully encapsulated (resin cartridge) 8m tendons were completed at a later stage by a mobile bolting rig.
Support of Structured Ground

Mining into the main fault structure of 404 Panel used the secondary support systems described above. The higher stress zone adjacent to the fault saw up to 4 or 5 phases of support. The initial continuous miner support patterns were supplemented by 8m tendon support placed by bolting rigs in the mining cycle. In some sections bolting was repeated, and in time, areas had steel square sets placed. Additional or alternative work such as fully grouted cable bolts/trusses, PUR injection and rib fill were used in some sections of roadway. Monitoring showed continuous roof movement over many months.

This was a very inefficient process, with mining rates in faulted zones less than 1m/shift.

Mining along the main north south fault zone in 404 panel was very slow. A pattern emerged of the CM mining for up to 10m with long tendon installation and/or steel square sets installed within 5m of the face. The decision to install sets was not made in some sections until it was obvious that the tendon support was inadequate. All of this work was made more difficult by a very soft floor.

The effect of the mining in the fault stress areas was not only a series mining support process but was compounded by the repeated efforts to stabilise the strata.

Improvements to Support of Structured Ground

For Springvale to improve development in structured ground two aspects were addressed:
- installation of pre-tensioned long tendons at the face, for more effective longer term support, and those tendons to be fully grouted;
- mining across complex strike slip fault structures affected by higher stress levels.

Post Grouted Tendons

Springvale developed and improved a pre-tensioned, post grouted tendon for installation at the face by the CM crews. They are normally placed:
- within 50m of projected structure zones;
- in north south “valley” roadways which are subject to higher stress;
- in special roadways, such as those driven higher in the seam for overcasts.

The 8m post grout bolt is installed and pre-tensioned to approximately 10 t at the face by spinning the bolt into a single resin cartridge and locking off with a barrel and wedge. The bolt is a normal Jennmar Superstrand which is bulbed at the inbye end. At a later time the bolt is fully grouted by specialist crews using a nil slump grout developed with MBT, which is pumped by grout tube to the inbye end of the bolt and down the hole to the collar. Campaign grouting routinely achieves above 40 bolts per 8 hr shift.

The post grout bolt allows long tendon support to be placed at the face where the effects of pretension gives the best result, and where the grouting can be done off the mining cycle. The improvement in development rates, in roadways where 8m tendons are needed within weeks of mining, has been of the order of 35% to 65%. For example, installation of 2 x 8m bolts per 2m to support N – S “Valley” roadways improved development rates from 4.4m to 7.3m/shift.

Patterns of post grout bolts installed maybe 2 per 2m to 4 per 1m. A typical cycle time for the installation of a row of post grout bolts is 16min. The post grouting of bolts is normally left from 1 to 4 weeks in typical higher stress areas. In the few intense stress structure areas it has been necessary to post grout within 10 to 20m of roadway advance.

Spiling and Arches

Mining across the most highly deformed and stressed structures does not have a simple answer. Springvale in being able to identify the location of these zones will be able to move the gateroad location if necessary. The mains roadways still will intersect fault zones.

The strong N – S structure zone intersecting the Main Panel outbye of 406 panel (Fig. 1) was initially mined with normal bolting systems. Those methods could not control formation of large cavities forming across the roadway.
(up to 5m high and 2m deep), in the crushed coal at the fault zone. A system of spiling and setting arches was implemented to regain control of ground conditions using a planned and organised process.

Spiling is a technique that had its origins from driving solid bars into fallen ground so the bar action provides a protective canopy. In recent times this crude technique has been improved with rails replaced by drill steels. The use of spiling in fallen ground has been extended by its application to solids.

Spiles were drilled into the solids ahead of the face and across the fault structure at approximately 300mm centres. Their purpose being to contain the extremely friable material as the face was advanced. This worked extremely well. A set of 18m long spiles, in an arch pattern, were drilled in 4-5 days. Drilling conditions played a role in timing. Important features for success were spile alignment, self drilling spiles drilled into place, reinforcing and grouting the spiles.

Arch sets were used in response to the failure of square sets used in combination with spiling on an adjacent roadway. The stress acting on the soft ground was not able to be contained by the square sets, although successfully mined using spiles.

Four piece arches, for ease of installation by Springvale crews, were designed by Connell Wagner based on their civil engineering experience and expertise. The mains travel road contains 49 arch sets at 750mm spacing. Key features were; 3 rounds of spiling and arches, use of shuttering and ribfill with CMT grout (at 8 to 10 Mpa strength) to stabilise and support the arch legs, and attention to packing the arch segment. Long tendons were installed prior to setting the arches.

This work was completed off cycle, initially by two selected Springvale crews. In the longer term it was thought using specific Springvale crews would be better than the issues of mobilisation and installation of large roadheaders. The process was refined on site, and severe fault zones intersected in the future can be handled in a controlled manner by experienced crews.

**APPROPRIATE INSTALLATION SYSTEMS**

One area in which geotechnical consultants could take a greater role in is working to match the best geotechnical solution with the appropriate mining cycle systems. The constraints on equipment at any site may limit the short term geotechnical options. Longer term goals should not be lost by either the mine site or the advising consultants.

Springvale were fortunate in being allowed to acquire a variety of bolting rigs to allow installation of long tendons (e.g. cable bolts, spinbolts, trusses, megabolts). The capability of each rig to install different types of tendons, in different diameter holes was in itself a difficult operation.

Development mining with secondary bolting in a series process did not satisfy the real geotechnical and mining cycle needs. In other words, maximise face installation of essential secondary support and minimise work done in series with the development process.

Implementing the use of post grouted bolts, installed from the CM during roadway development has simplified the following areas:
- pre-tensioned long tendons are set at the face
- subsequent grouting is done in parallel with development, except in the most severe conditions.
- CM’s do not require bolting rigs with the more complex through the chuck motors
- The range of drill bit sizes used reduces to 3 (from 7)

The use of specialist teams of contractors has proven to be very effective at Springvale as they develop efficiencies by focussing on one task. The main areas of their work has often been related to the presence of structure zones.

For example
1. A specialist weekend crew to post grout the long tendons. They have an effective rate of 40 bolts per 8 hr shift.
2. Bolting crews who install long tendons in the maingate ahead of the retreating longwall. The ongoing task is focussed at the maingate stress notch approaching the longwall take-off. Importantly they are used to add supplementary support to identified structure zones which cross the maingate and usually have heavier conditions.
3. Project crews to prepare the special purpose roadways prior to longwall extraction. For example, the roads specifically driven into longwall blocks to avoid faulting. Such work involves removal of steel sets and replacement with cuttable cans, rib filling and pumping sections of roadways.

SUMMARY OF SUCCESSES

Springvale Mine have experienced a range of difficult mining conditions associated with geological structure zones and roof falls. A series of methods have been developed to lessen the impact of faulting on the business.

- A geological structure model has been developed which has been directly responsible for highlighting the zones, particularly in the longwall maingate where roof falls are most likely.
- There have been no substantial roof falls in production areas since the development of the geological model and associated roof support measures.
- The geological structural model for Springvale was developed by placing Springvale into a regional structural context. This recognised the most likely origins and orientations of the structures in the mine. It also provided exploration targets for in seam drilling.
- Springvale has developed a risk ranking of the different structure types and associated mining rates which are fundamental in building the production schedule. Be aware that geological models always need to be upgraded and their blinkered acceptance is unwise.
- Analysis of roof monitoring and observation data has shown that the installation of pre-tensioned fully grouted tendons as soon as possible after mining improves the long term roof conditions.
- The development of the 8m long Post Grout Tendon at Springvale, which could be installed and pre-tensioned during development, has improved development rates by up to 65% in areas which need such support.
- Severe mining conditions such as crossing highly stressed fault zones, were mined in a planned and controlled manner using spiling and arching methods.
- Strata control issues which are significant enough to pose a business risk were best resolved at Springvale by developing a committed Technical team with responsibility and authority for the task.

FUTURE CHALLENGES

The most obvious challenge for mining into structured ground is to know the geology, in particular:

- Where the fault is located;
- The intensity of the fault zone;
- The mining rates through the zone;
- The support intensity and effectiveness.

This work starts with the ability of the geological personnel to provide a level of data for assessment. Experience would show that the geotechnical solutions are usually satisfactory if the necessary data is available. Any mine site that does not have a clear picture of the geological structure and its variability will have to rely on luck to always meet its business plan without the impact of unscheduled delays.

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