2002

Proceedings of the 2002 Coal Operators' Conference

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Publication Details
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FOREWORD

In arranging the COAL2002 Conference the Organizing Committee have taken the opportunity of remembering the contributions of the late Dr Ripu Daman Lama to the science of coal mining in Australia and honouring him for his work. Dr Lama made significant contributions to the coal mining sciences and was a world leading authority in seam gas research before his death in January 1997. The Australian Coal Industry was fortunate that Ripu came to Australia in 1975 and provided his expertise in so many technical areas. Included in this volume is a joint paper which was prepared during the last months of 1996 by Ripu.

Dr Alan Hargraves was honoured in the COAL2001- Geotechnology Colloquium and it is appropriate at this time to recognise the achievements of Dr Lama. Ripu's particular areas of expertise lay in geotechnology and outburst management and the COAL2002 Conference technical program has been oriented towards presenting the latest developments in these two fields. Ripu became an internationally recognised expert in those disciplines and we honour him for his achievements.

Ripu was a fellow of The AusIMM and apart from his many technical publications was a frequent presenter of talks to the Illawarra Branch of which he was a Committee member. One of his major contributions was his role as Convenor, Chairman, Editor and driving force for the International Symposium on Management of Gas Emissions and Outbursts held in Wollongong in 1995. This Symposium gave birth to the local Gas and Coal Outburst Committee which is a part of the organisers for this Conference.

The COAL2002 activities include a keynote address by Mr Bruce Allan, Vice – President of BHP Billiton Illawarra Coal, in which he challenges the coal mining industry “to attract, support and house potential Champions to find safe and economic methods to understand and manage the phenomenon of gas and coal outbursts”.

John Hanes, Coordinator of ACARP In-Seam Drilling and Gas Research organised one-day Pre-Conference workshop on Outburst and Gas Management. Thanks to John and his team of experts for providing an excellent professional development program. The workshop notes are included in this volume as an appendix.

The COAL2002 Conference has been generously supported by the following organizations:

- ENEX Resources (Major Sponsor)
- Coal & Allied
- BHP Billiton, Illawarra Coal
- Strata Control Technology Pty Ltd
- Ground Consolidation
- Roadway Reinforcement Services
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Our thanks go to the authors who have accepted the invitation to contribute and present papers at this Conference. They represent a cross section of the developments in the coal industries in Queensland and New South Wales.

The AusIMM Illawarra Branch and the organising committee are extremely grateful to all the above for their support.

The success of the COAL2002 Conference is in no small part due to a personal commitment by a number of individuals who worked so hard behind the scenes. Special thanks are due to the speakers, session chairpersons, Maureen Prince for her help at the registration desk, the University of Wollongong staff - especially Gordon Nolan and Gary Piggott of Printery Services, Bruce Robertson of the Audio Visual Unit, Leonie McIntyre of the Faculty of Engineering for typesetting the Conference proceedings, and finally the Organising Committee.

Ernest Baafi, Associate Professor, University of Wollongong
Bob Kininmonth, Chairman of Gas and Coal Outburst Committee
PREFACE

The Engineering Faculty has an excellent track record for organising conferences on Mining Engineering. The Faculty has brought together practising engineers, researchers and scholars from all over Australia and from the best Mining Engineering Centres in the world. Our colleagues in Industry have always assisted us in the organisation of conferences and we have had the support of The Australasian Institute of Mining and Metallurgy (AusIMM). In fact, the local branch of The AusIMM has been a key player in organising and hosting conferences, seminars and other professional activities. We are very proud of these close links between the University and Industry. This achievement of the Faculty reflects what is true of the University as a whole. As you well know, the University has been recognised as the University of the Year in two consecutive years and one of these awards was for outstanding R&D partnerships.

The Coal Operators Conference series is the most recent and significant example of a collaborative endeavour between industry, academics, researchers and The AusIMM. This is the third conference in the series and I am sure there will be more to come.

The third Conference is dedicated to the memory of the late Dr R.D. Lama. Dr Lama started his career in academia which included his first research projects and then he played a leading role in a senior capacity as a professional mining engineer and as a research manager.

He combined love of scholarship and research with a keen awareness of the many interesting problems and issues in modern mining engineering. Dr Lama worked closely with the academic staff of the faculties of Science and Engineering at this University and, in particular, he was closely involved with the Mining Engineering academics. His collection of research papers is currently housed at the Faculty of Engineering. He devoted many years of his life to the use of latest knowledge and emerging technology for advancing the profession. He directed his considerable research effort to solve real-world problems of significance. Consequently, he gained the respect of industry colleagues everywhere. It is entirely appropriate that the leaders amongst the mining industry have joined with the Faculty of Engineering in honouring the memory of this man who passed away prematurely. The profession is poorer for the loss of Dr Lama. However, his example will be valuable to the profession and especially to young engineers.

Professor Robin Chowdhury
Dean of Engineering, University of Wollongong
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OPENING SESSION
OPENING ADDRESS

Ian Goddard

I am grateful for the invitation to be with you today at the University of Wollongong, the University of the Year for 2001. This is my first official role as President of The Australasian Institute of Mining and Metallurgy for 2002, apart from chairing the first Board meeting of the Institute for the year on Monday.

It is significant that COAL2002, the Conference on Outburst and Gas Management and Strata Management Systems, is being convened by the Illawarra Branch, as it is one of the most active and committed branches in the Institute. The other organising entities are the NSW Coal Mine Managers’ Association, the Gas and Coal Outburst Committee and the University of Wollongong.

A few words about the current status of the AusIMM.

Over the past three years The AusIMM has introduced new Member Benefits and Services, has started to “speak out” on issues affecting the members and has restructured and revitalised the internal processes and communication channels. At the same time it is maintaining emphasis on the recognition of Continual Professional Development and Education and on the delivery of technical publications, conferences and events, Codes and Ethical procedures.

The needs of the professionals within the Minerals Industry are continually changing as the Industry becomes truly global and employment practices change. The challenge for our professional institute is to remain relevant to their needs. However, the central need continues to be support for and recognition of continual professional development.

The internationalisation of the industry is opening up new challenges and thus the Institute will be exploring opportunities for international recognition of qualifications, the production of international publications and events and the expansion of our codes into the international arena in 2002.

For example, The AusIMM will be hosting the CMMI Congress in Cairns in May 2002 with the themes of International Codes, Technology and Sustainability for the Mineral Industry. Leigh Clifford from Rio Tinto has agreed to be the Congress Chairman.

The Institutes, which represent the professionals in the industry, as well as the individual professionals, have an important part to play both internationally and regionally in addressing these opportunities.

This will require both the enhancement of advanced technical skills and of skills which assist in the engagement of all those who perceive themselves to be stakeholders in the industry.

Conferences such as this provide the opportunity for professionals in the industry to gather and share knowledge. They are a very efficient mode of information transfer and subsequent reference to the proceedings from this event will enable those not present to upgrade their knowledge as well. We need professional engineers and scientists to develop their skills in the management of gas and coal outbursts for the benefit of the wider community.

Speaking of Proceedings, I would like to acknowledge the efforts of Dr. Ian Porter and his team from this Branch, who made the offer to produce the Institute’s Proceedings to ensure that The AusIMM continued to publish a respected technical professional journal. That offer was accepted at Congress 2000 and I am sure you would agree that the quality of the volumes has been improved and it is now a significant journal in modern mining literature. So much so, that we are near to agreeing with the IMM in the UK (at their suggestion) for a joint publication with world-wide appeal.

In 2001, The AusIMM launched the report prepared by World Competitive Practices, on “Rising to the Challenge – Building professional staff capability in the Australian minerals industry for the new century.”

1 The 2002 President of AusIMM
This report raised the question “Are we confident that the minerals industry has the professional staff capability to appropriately position it to meet the demands of tomorrow?” The report concluded,

- There is an urgent requirement for those in the industry to work together to improve its image and perhaps build new employment options for the people it needs. The Institute can take a lead role on this.

- Companies must look to their organisational systems and structures to identify and attract the people they seek, and to motivate, develop and utilise their professional staff.

- The Institute must find ways to ensure that the universities and other education providers are better informed of industry challenges and needs.

- Industry professionals, on an ongoing basis, need to review and develop their competencies.

Mining has a proud tradition in Australasia, particularly in coal. It has spawned many of Australia’s most celebrated business leaders – leaders who have had a capacity to map a future for their companies and the industry itself. Current industry leaders must do this again if the industry is to achieve sustainable success.

Turning now to the subject of the Conference.

Although my career has been mostly with base metals and gold, I did spend 7 years in MIM’s Coal Division in Queensland. Most of this time was spent as a “sunshine Miner” as I was the manager of the Newlands mine for the first 6 years of its life. There were no gas issues at that time, although there were plenty of others to keep me occupied.

My real introduction into the coal industry as a mining engineer took place twenty years ago, when I was transferred to Collinsville as Planning Manager for half a year. The Collinsville mines had a long-standing gas problem, not with methane but carbon dioxide. In the early part of last century, an outburst had disastrous consequences, with large loss of life.

At the time of my arrival, a number of research projects into the management of gas outbursts were underway. The Collinsville Research Manager was one Dr Ray Williams. I look forward to his paper later today. Ray educated me about coal in general and the problems of controlling gas release. He was very understanding of the limitations of a hard rock miner. I continued to visit Collinsville regularly during those years and kept in contact with Ray and progress on the research projects.

In his talks, the name Ripu Lama came up frequently. I realised that he was highly regarded in the industry and was recognised as the international “guru” on outbursts. I am pleased to acknowledge the presence of Ripu’s wife, Barbara, here today. This conference is dedicated to Ripu and we will be hearing more of his magnificent contribution to the AusIMM, to this Branch and to the study and management of gas emissions and coal outbursts.

I would like to ask Barbara to come forward and receive a commemorative satchel from the Conference.

I note that the technical programme for the next two days has a wide range of subjects, from keynote address, to history, to theory, to case studies and an open forum on both days. Yesterday there was a Workshop. I congratulate Alison and Ernest and the team on their efforts. I would also like to acknowledge the sponsors, without whom conferences would not be so successful.

The management of outbursts and gas is extremely important to this nation for safety, economic and environmental reasons. I am sure that the Conference will be a great technical, professional and social success, meeting all of its objectives.

I have much pleasure in officially opening COAL2002 – the 3rd Australian Coal Operators’ Conference.
CONTRIBUTIONS OF DR RIPU DAMAN LAMA TO GAS OUTBURST AND CONTROL AND STRATA MANAGEMENT SYSTEMS

Satya Vutukuri

GAS OUTBURST AND CONTROL

With a rapid increase in underground coal mining in Australia, the problem of outbursts and high gas emissions has become very serious. In 1978, while working with the CSIRO Division of Applied Geomechanics, Dr. Lama started to work in the area of outbursts of gas and coal and control of gas in coal mines. Since that time, he was deeply involved in this work. The main thrust had been in the following directions:

1. Gas content estimation.
2. Characterisation and prediction of outbursts and control of outbursts in operating mines.
4. Post-drainage and control of gas in longwall operations.

The main concepts put forward by him can be listed as follows:

1. For successful and efficient control of coal and gas outbursts, gas content determination up to 100 m ahead of an advancing heading requires development of new methods which give accurate values within six hours of sampling.
2. There is a need to develop techniques for the prediction of dislocations (shears) in coal to predict outburst sites.
3. Because of the contraction of coal matrix on application of suction, the flow rates can be greatly increased when high suction is applied to the boreholes.
4. Flow rates (post-drainage) are highly dependent upon the joint direction and fracture development in the floor of the coal seams.
5. While planning for ventilation, gas emission rate cannot be taken proportional to production. This relationship is not linear. Also in mixed gas situations gas the liberated has a different composition from the in-situ gas content of coal.

Summary of the research findings of Dr Lama:

- Research in the area of gas content measurements showed that the use of cuttings can give gas content and gas composition values within 10% of the actual value. This method has the advantage that there is no need of a core and can be applied in underground conditions where core recovery is difficult.

- A number of methods for the prediction of shear zones in underground mines have been proposed and researched. These include methods based upon differential sorption properties of coal as these change under the effect of a shear structure, gas pressure measurements which change as a result of changes in the permeability of the structure and fracture density measurements. The fracture density and sorption properties change up to 20 m away from the shear structure, but gas pressure changes can occur up to 100 m away from the structure.

- Threshold values of gas content of coal both for CO₂ and CH₄ were proposed for the mining of the Bulli seam and these have now become more or less an industry standard.

- A new method for the prediction of outburst potential of a coal seam on regional basis was proposed and was applied in a Queensland mine.

1 University of Wollongong
Research in the area of in-seam and post-drainage indicated the effect of high suction on increasing flow rates from the Bulli seam by a factor of three. This allowed successful drainage of the Bulli seam hitherto considered as undrainable. This was acknowledged as the first most successful research project funded by the Commonwealth Government of Australia. Research also showed that the optimum spacing of holes, effect of jointing and stress, and geological discontinuities, all play an important role in improving gas recovery. Together with power generation, this research project was awarded the Engineering Excellence Award of NSW in 1985.

In the case of mixed gas emissions, research showed that the gas compositions change with time. A model was developed that allows prediction of changes in gas composition. Research showed that even when the gas composition in in-situ coal is 60% CO₂, the gas liberated will be 85% CH₄ and this aspect must be taken into account in ventilation planning, evaluation of gas reserves and in the estimation of threshold values for safe mining of the Bulli seam. The method suggested by Ripu for true estimation of gas composition is now an accepted practice in the coal industry of Australia.

**STRATA MANAGEMENT SYSTEMS**

Most studies on the strength of rock are based upon small samples tested in the laboratory where, neither the field conditions, nor the size effect can be taken into consideration. As such, the results of laboratory tests cannot be applied directly in the prediction of field behaviour. For the application of laboratory results to field behaviour certain reduction factors are used. Ripu felt that this concept is not necessarily applicable.

The concepts he developed in this area include the following:

1. For any geomechanical data to be reliable, tests must be conducted in the field or on sample sizes which can adequately represent the field samples e.g. behaviour of jointed rocks.
2. In brittle rocks, the behaviour is related to the development of cracks and classical theories representing field behaviour and material softening are not applicable. This is true for intact and fractured rock e.g. time dependent deformation of rock.
3. In the design of underground structures, it is essential that data be collected using back analyses of such parameters which have most effect on the design e.g. angle of internal friction in pillar design.
4. Each case must be analysed taking into account the geology which plays an important role e.g. massive beds in subsidence.

In the area of basic geomechanics, Dr. Lama conducted research in the following areas:

1. Mechanical properties of intact and jointed rocks.
2. Time dependent behaviour of jointed rocks.

In the area of applied geomechanics, research was conducted in the following areas:

1. Pillar design under high horizontal stresses and yield pillar design.
2. Roof bolting design.
3. Early strength packing materials and stability of wide headings.
4. Ground movement of narrow and subsidence over wide openings.
Summary of the research findings of Dr Lama:

- Research conducted on testing of large samples in the field and comparison of the results on smaller samples in the laboratory showed that when field data is required for mine design, laboratory results are not applicable particularly in fractured rocks with high density of joints such as coal. The data is not only quantitatively but also qualitatively unreliable. The compressive strength of the samples is dependent upon the stress distribution on the bearing surfaces of the samples and this shows up in the power relationship of strength and (height/diameter) dimensions. If the stress distribution is uniform, as is the case with brush platens, then the (height/diameter) effect vanishes. The volumetric effect is important and this is related to the number of defects and the type of defects in the rock. For a sample to be representative of the rock mass, it must contain at least 100 – 150 such defects.

- The dynamic failure of rock is a result of the relative stiffness of the roof and floor rocks as well as the change in the loading conditions of the rock. When the state of stress suddenly changes from triaxial to uniaxial, the extra energy is suddenly released in the form of rock bursts and also in the form of outbursts of gas and coal.

- Stress distribution in jointed rocks is determined by the mobility of the joints and this determines the strength of the jointed system. The closeness of the joints and the angle of orientation of the joints, joint continuity are all important. The post-failure behaviour is determined by the joint density. Rocks with high joint density have much smaller post-stiffness and fail gradually.

- The time-dependent behaviour of intact rocks is the result of crack propagation. The amount of strain before failure that the material will undergo is determined by the post-failure curve of the rock. The material must deform so much such that the time-dependent strain induced approaches the post-failure curve for the stress conditions imposed. For this reason, the intact rock will undergo a large amount of deformation. Fractured rock, on the other hand, already lies close to the post-failure curve and hence the fractured rock will show very small time-dependent deformation. This is also supported by the fact that the effect of rate of loading on intact samples of rock is considerable, but the effect of displacement rate on frictional behaviour of joints is very small. This research was awarded the Heico Gold Medal by the Indian Geotechnical Society in 1978.

- Research on prediction of failure using micro-seismic studies showed that the higher frequency noise increases at a faster rate as the ultimate failure approaches, though the energy associated with lower frequencies is higher. Using theoretical modelling and experimental results, it was shown that as the fractures develop in a model or a system, the resonance frequency of the system moves towards the lower frequency spectrum.

- When rock joints are filled with clay, the thickness of the fill plays an important role. When the thickness of the fill is small, there is some dilation and displacement values at peak decrease with increase in thickness, but as the thickness of fill increases, there is consolidation of the joint followed by a continued increase in displacement at peak shear. The effect of this is that the behaviour of the filled joint is not governed by the behaviour of the material filling the joint. Even at a fill thickness of twice the asperity height, the joint is stronger than the fill material.

- Research based on an industry wide survey of roof conditions on the stability of roadways particularly under high stress conditions showed the direction of drivage of the heading with respect to the principal horizontal stress is important. Roadways driven at angles of 30 – 90 degrees to the major stress undergo greater displacement and greater damage as the angle increases. Analysis showed the drivage of the first roadway can relieve the stresses in the next roadway to be driven. The distance to which the relief can be expected is linearly related to the height of the caving. In the design of the longwall layouts, this is the most important consideration in Australian mines.

- It is important that the roof bolting design must take into consideration not only the strength and stress parameters but also the actual deformation values that the roadway undergoes.
Keeping this in mind a computer program was developed which incorporates the field measurements and optimises the rock parameters to arrive at the optimum values of bolt requirements. Research using this program clearly indicated that in cases where the roadway has to be placed at very high angles to the major stress or where the depth of excavation is large, it is essential that the roof bolting system be optimised by allowing some displacement of the rock and using yielding roof bolts. Otherwise the roof bolting density will be too large and uneconomical. The concept of immediate support will not be appropriate under these conditions.

- Under a NERDDC Project, he conducted studies in roof bolting with a view to increase development drivage rates. He demonstrated that by reducing bolting cycle time through application of high pre-tension to full-column grouted bolts. For this work, he had the 1995 ACARP Award of Excellence in the Underground Category.

- The design of pillars particularly under conditions of high horizontal stresses must take into account the behaviour of the rock in-situ as the pillar behaviour and fracturing gets modified under such conditions and pillar design using classical methods, based upon vertical stress as the major principal stress, are not applicable as these give much higher values. A special technique was developed which is based upon the measurement of fractured zones using air permeability. This allows the calculation of the in-situ friction angle of the coal pillars which gets modified depending upon the roof, floor and stress condition. This allowed design of 10 m wide stable pillars for depths of 465 m for a 3-heading development and were introduced in KCC mines. The results also showed that the width of the pillar will depend upon the direction of drivage. This result cannot be deduced from classical pillar design theories.

- The results of a major tapering pillar experiment also showed that the width of a yielding pillar for the layout of a 3-heading gate road development to combat high stresses during the mining of a retreat longwall panel is about 8 m. The results of this research were applied to the design of longwalls 12 – 14 at West Cliff Mine and these resulted in an increase in productivity by a factor of two compared to longwall 11 mined in the same area. This technique was consistently used in the set-up heading when stresses are high in all longwalls at West Cliff and Tahmoor mines and was adopted in other mines.

- Studies on subsidence under massive sandstone beds showed that the effect of chain pillars on the surface is absent. The surface behaviour is governed by the bending of the massive beam the stable thickness of which develops as the critical width of the excavations is reached. This thickness under Southern coal field (NSW) conditions was found to be about 100 m. This has eliminated the requirement for modification of longwall layout to control subsidence damage.

- A new system of gate road development based upon Shortwall mining with a central pump pack was proposed. This system can develop roadways faster than any other existing systems and is cost effective. A major study under this project conducted showed that the coal wash reject, 10 mm, can be pumped up to 2 km at solid density of 82% and can be placed in position with a compressive strength of 0.25 MPa within 2 hours. At 450 m depth this system with a roadway width of 15 m is stable. It was suggested that this may be the best method where 3-heading development is needed due to high stress and high gas conditions.
CONTRIBUTIONS OF RIPU DAMAN LAMA
PROFESSIONAL CONTRIBUTIONS

Dr. R. D. Lama graduated from Punjab University (India) in 1957 majoring in Physics and Chemistry and obtained his B.Sc. Mining Engineering from Banaras Hindu University (India) in 1961, with a first class. He was awarded a Government of India Scholarship for studies in Poland where, in 1966, he obtained his PhD. from the Academy of Mining and Metallurgy in Kraków. Based on his research, he obtained DSc. degree from the Indian School of Mines in 1994 and also DSc. Mining degree from the Kemorovo Mining Research Institute of the Russian Academy of Sciences, Siberian Branch in 1995.

On his return from Poland, he joined as Reader in Coal Mining at Banaras Hindu University where he served till 1971. From December 1971 to December 1974 he was Senior Research Scientist at the Institute of Soil and Rock Mechanics, University of Karlsruhe, Germany. In 1975 he joined the Division of Applied Geomechanics, Commonwealth Scientific and Industrial Research Organisation (CSIRO), Australia, first as Senior Research Scientist and then as Principal Research Scientist (1976) and Section Head of Coal Mining Geotechnology (1979). In 1981 he was offered a position as Manager, Mining Technology, Kembla Coal and Coke Pty. Ltd. (RTZ-CRA Group); the position he occupied until his death.

Since his graduation, Ripu worked in universities, teaching and performing research in India, Poland and Germany (13 years), research in CSIRO (7 years) and in the Coal Mining Industry (14 years). His technical expertise can be divided into the following areas:

- Design of coal mines (from exploration to execution, with particular reference to geologically disturbed areas and deep mining)
- Mine evaluation, economic analysis, feasibility reports for coal mines
- Geomechanics applied to coal mining and civil engineering in tunnelling, slope stability and large underground chambers
- Gas from coal seams, gas drainage and utilisation and gas outburst in coal mines
- Dust control in mines

He has written 128 papers and 91 consulting reports in the area of geomechanics, gas control and outbursts in coalmines, and mine planning and design. He has co-authored five books in geomechanics and mine ventilation which are standard references and these have been translated into Chinese and Japanese. The four volume book *Handbook on Mechanical Properties of Rocks* published by Trans Tech (Germany) is a standard reference book on the subject and the fifth book *Environmental Engineering in Mines*, published by Cambridge University press, is a standard text book for undergraduate and post-graduate mining students. These books have been translated into Japanese and into Chinese. He was also the editor of the proceedings of the *International Symposium cum Workshop on Management and Control of Outbursts in Underground Coal Mines* which was held in Wollongong in 1995.

He has been a consultant to mining companies in many countries including USA, Australia, New Zealand, China, Germany, India, Poland and Greece. He has run advanced courses in the areas of his expertise and lectured in many countries in Europe, Asia and North America. He is also an Adjunct Professor of the Beijing Graduate School of Mining and Metallurgy, Beijing, China. Ripu was co-supervisor of five students who successfully completed PhD Thesis. He was also been an examiner of MSc and PhD Theses submitted at the Universities of NSW, Wollongong and Monash.

Dr. Lama was a member of many national and international bodies on which he represented Australia. He was a recipient of the Robertson Medal of MGMI (India), B.H.U. Gold Medal and Nand Lal Gold Medal from Banaras Hindu University (India), Heico Gold Medal from the Geotechnical Society (India), Engineering Excellence Award from the Institution of Engineers, NSW (Australia) and Research Excellence Award from the Australian Coal Research Association. In 1991 the International Bureau of Rock Mechanics nominated him for the Japan Prize, the highest award in engineering. He was listed in *Who is Who in Engineering* published by The Association of Engineering Societies, USA. In 1995 Dr. Lama was elected to the Corresponding Member of the Russian Academy of Natural Sciences.
**PERSONAL TRIBUTES**

Ripu Lama demonstrated the true value of good science to the improvement of both the safety and economics of the mining process. He was a person of prodigious intellectual capacity and generous disposition whose contribution to the industry cannot be adequately measured by conventional metrics. I am a better Engineer and Manager for having worked with Ripu. I have no doubt there are many individuals in our industry that will reflect on his contribution to their own personal development and feel the same. While we mourn his loss, we should celebrate his life and his achievements.

*Mark Cutifani, Managing Director, Sons of Gwalia Limited*

Ripu was a person of innovative ideas, tremendous enthusiasm and energy. He was an early advocate and researcher of many current practices in gas drainage and strata control. I remember, that on many occasions, when an underground mine was experiencing ground control problems, Ripu would be called to advise on the causes and possible solutions. He was a good colleague, greatly missed.

*Colin Seaborn, formerly General Manager Technical Development, Kembla Coal & Coke*

I have had the privilege of working with Ripu over a number of years – trying to assist him in unravelling the complexity, and mystery coals porosity and certain coals’ gas holding capacity – not to mention the phenomena gas-desorption. Unashamedly, I confess to my envy and admiration for Ripu’s ability to solve a mathematically based hypothesis in his head, without having to put numbers on paper while I was fumbling with my advanced HP-Scientific calculator to get to the same answer.

However, much more important and memorable to me was the pleasure to work with Ripu for his inescapable kindness and readiness to listen to anyone’s problems (of which there were plenty) and offer his immediate help – not just promises. Such irrepressible kindness was of particularly great help and encouragement to the numerous post graduate students whom he supervised, and visiting overseas professionals with whom Ripu had collaborated.

With fond memories – that will be with me – as with your other friends and associates in the “Sciences and Mysteries of Coal Mining”

*Michael Pretor, Austral Coal*

I first met Dr Ripu Lama when he attended a conference in the mid 1970s. He was always dedicated to his work and to research. He was a driver and was a very efficient person who had the ability to analyse a problem and put it into simple perspective.

*Lew Griffiths*

Ripu or “Rip” as he was affectionately known on the minesite possessed the ability to relate his practical knowledge equally well to the machineman on the face, as those esmerized by his technical presentations at world class symposiums. His record of enlisting the enthusiasm and support of managers in applying the results of his research in providing practical solutions to a broad range of mining problems gained him unparalleled respect and admiration.

Ripu’s dedication to the improvement of the industry and its people, rather than reflection upon his past achievements remains a measure of the calibre of the man - we miss him.

*Bob Miller, General Manager, Springvale Coal Pty Ltd*
I have known and worked with Ripu Lama since the late 1970’s, when he first came to work at West Cliff from the CSIRO.

During the next 20 years Ripu, was personally worked at the leading edge in the development of gas drainage and rock mechanics technologies. He had the exceptional ability to not only carry out detailed and comprehensive pure research, but was able to translate this information into practical solutions and applications that were well accepted by operational personnel.

One of his key skills was that he was able to communicate equally well with all levels within the industry, from academic researchers, to business leaders and technical personnel right through to the miner at the coal face.

In my opinion, his greatest contribution to the coal industry was the technical support given to the elimination of the risk of gas outbursts in the Bulli Seam by the development and implementation of outburst management plans. Without this, the future of many mines in the Southern District would have been less than secure.

Ripu Lama will always be well remembered and respected by all those with whom he worked. We all miss him.

_Ian Sheppard, Manager Engineering Services, Tahmoor Colliery_

"Ripu at Kembla Coal & Coke Office, Wollongong"
Ripu at CSIRO Syndal, Victoria

Ripu at CSIRO Syndal, Victoria
Underground visit by Ripu

Ripu visiting China 1990 with Ken Cram

Ripu and Barbara at NSW Blue Mountains, 1992
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INDUSTRY IMPACTS
OUTBURST AND GAS MANAGEMENT

Bruce Allan

Outbursts of gas, coal and rock are not a new phenomenon and have seriously impacted the safety of men and equipment within mines and the ultimate viability of the operations.

Operational and management techniques have been applied to mines in the Illawarra as a means of working with outburst issues, but a complete understanding of the phenomenon at the different mines and locations is still lacking.

Over the years specialists have championed the studies and investigations and significant advancements have been made in the safety of the mines. Extraction of quality reserves in otherwise unmineable domains has been made possible.

With the economic pressure now on the coal industry, studies and research into outbursts of coal and gas has declined worldwide, and we are at a crossroad for the future.

Outbursts have occurred in literally all the major coal producing countries of the world and over 35,000 outbursts have been noted worldwide in the past one hundred and fifty years, some with very serious consequences, and which on occasions resulted in loss of lives.

The outburst of gas, coal and rock is neither new nor isolated to any particular coalfield in Australia. This phenomenon has forced mine management to seek to develop an understanding of outbursts and develop methods for the management of gas within mines.

Initially outbursts were managed by practical experience but over the last sixty years scientific research and experimentation has led to a better understanding of the phenomenon.

A large number of gas outbursts have been recorded in the Illawarra coal fields from the 1890s to the present. All have been within the Bulli Coal Seam from Metropolitan Colliery in the North to Kemira Colliery in the south to Oakdale and Brimstone Collieries in the west.

Coal gas outbursts types have varied from pure CH₄ to Co² and have ranged from small to quite large in intensity often liberating large quantities of coal.

Dr Alan Hargraves and Dr Ripu Lama had championed focussed research in the Illawarra from the 1960s to the late 1980s. Their work which was applied at a number of Illawarra mines led to the greatest advancement and understanding of the coal mine gas outburst phenomenon. Unfortunately this did not prevent the outbursts which resulted in the tragic loss of life at South Bulli (1991) and West Cliff (1994).

Outburst management plans were introduced into NSW and the Illawarra region following the South Bulli incident. The outburst management plans developed by mine management in 1991, tended to be based on the respective experiences of the mine to that date, lacked formal approval processes and “hard barriers”.

In 1994 the Department of Mineral resources (DMR) called together coal mining industry operators from the Illawarra, and researchers together with the inspectorate to draft an outburst mining guideline which formed the basis for MDG 1004 – Mining Guidelines (July 1995). These guidelines now form the basis of outburst management plans at coal mines within the Illawarra and the state of New South Wales and have created a whole new approach to hazard management of coal gas outbursts, putting “hard barriers” into place in the management plans for coal mines.

1 BHP Billiton, Illawarra Coal
Significant documented material and experience is available worldwide for present and future mine planners and operators to understand the potential and liability for a coal seam to outburst. This can be broken down into the following key areas.

- Presence of faulting
- Structural related thrust or horizontal movement
- Seam variations and coal strengths
- Coal cleating and jointing
- Presence of dykes
- Seam permeability
- Seam gas content
- Gas pressure
- Seam gas type, i.e. CH$_4$ or CO$_2$
- Seam gas composition

The important issue in all this is to understand the broader picture of the resource to be mined and to accurately manage and understand the coal seam physical changes, be they large or subtle.

The present management and control of outbursts of gas and coal is based on the two following broad concepts:

- Develop methods, which reduce the likelihood of outbursting.
- Develop systems such that men and equipment can be protected from the effects should such an outburst event occur.

which can be simply expressed as Prediction, Prevention and Protection.

**Prediction**

- This requires detailed geological understanding of the source and interpretation of known Geological anomalies.
- In-seam exploration and seam gas contents
- Measurement of coal seam stress conditions
- Identification and training of employees

**Prevention**

- Application of the various types of gas drainage techniques
- Induced shot firing
- Water injection
- Remote mining

**Protection**

- Outburst gas management plans that are functional and auditable
- Control of ventilation and gas monitoring
- Provision of self-contained self-rescuers linked to self-escape systems
- Training and development of people in relation to outburst management

The prime objective of Illawarra coal mine outburst management plans is to facilitate exploratory inseam drilling and gas drainage aimed at reducing seam gas contents to agreed threshold levels in all areas of the mine, where development and longwall operations are to be carried out. This drives a “permit to work” philosophy.

Resultant gas content levels ensure that the risk of an outburst, or other release of dangerous quantities of noxious or flammable gas, is minimised to allow normal mining operations to be carried out.

In exceptional instances where this objective is unattainable, the plan can make provision for alternative mining procedures to be used. These alternative mining procedures would be under strictly controlled and considered circumstances, which maximise protection to employees and the operation.

Outburst management plans in NSW are formulated in accordance with the Department of Mineral Resources (DMR) Outburst Mining Guidelines (MDG 1004) July 1995. Plans are reviewed and audited at intervals not exceeding two years.

Threshold levels for operating collieries mining the Bulli Seam vary from $5 - 9.5$ m$^3$/t and is dependent upon the percentage of CH$_4$ or CO$_2$ in the sample.

To date, when mining below or within these agreed threshold values and subject to the presence of geological anomalies, no outbursts of coal or gas have been reported.
In a recent case at Tower Colliery where the gas threshold could not be reduced in the time available, it was agreed to remote mine the area. Development was slow, cautious and costly in order to protect men and equipment. This remote mining development driveage induced an outburst using an ABM20, continuous miner, whilst cutting through a known geological structure.

The current gas threshold levels in use in the Bulli coal seam do have a safety factor. Without greater factual and scientific information being available and understood, there is too high a risk to vary them.

Another issue that impacts the true threshold values is the reliability and accuracy of the sampling and analysis of the seam cores yielding the total gas content. The capture, sampling and testing for total seam gas analysis has wide and varied limits of accuracy and until they can be narrowed and precise measurements fully understood, the present threshold limits used in NSW have no supporting basis for change.

The drainage and removal of gas into a pipe range as practiced at most major mines operating in the Bulli coal seam is not without significant additional cost to the operation.

On average, for BHP Billiton Illawarra Coal mine gas drainage costs average $1.20 per run of mine tonne. This excludes the costs of owning and operating a surface gas extraction plant and excludes the cost of capital.

In the case of a predominant CH\(_4\) environment some of this cost may be offset through the ability for power generation or steam raising, but in the case of CO\(_2\) gases no viable alternate uses are in play.

In an area were gas drainage has not been effective or has been delayed due to particular circumstances or events, costly delays in development or longwall production can have major and lasting economic impacts on a mining operation. This can ultimately impact on the viability of the mine and Company.

Over 700 outbursts which have impacted on mine safety and production have been recorded in Australia over the past 100 years. During this period research has been somewhat variable in Australia. Overseas countries, which provided a large source of outburst related research in the past, have now closed either most or all of their coal mining industry.

In Australia, Alan Hargraves, applied some predictive techniques to outbursts, but it was not until the late 1970s that Ripu Lama, then working for the CSIRO, joined the effort by conducting micro seismic tests at West Cliff in an attempt to trace and predict outbursts of gas.

From the early 1980s a significant amount of in-house research and development (R&D) was undertaken by both AIS Collieries Pty Ltd and Kembla Coal and Coke Pty Ltd, to try to understand and manage the phenomenon of gas outbursts in the Bulli Seam. Both organisations along with external research groups utilised the support of the Federal Government NERDC funded programmes.

Many successful and valued projects grew out of the NERDDC research, one in particular being the West Cliff CH\(_4\) gas extraction and 13 megawatt mine site CH\(_4\) gas power project.

Today NERDDC has been replaced by Australian Coal Association Research Programme (ACARP) which continues to be highly supportive of projects related to gas drainage, drilling and outburst related research.

Since the early 1990’s ACARP funded research into CH\(_4\) gas drainage and outbursts has declined in Australia.

The focus and structure of research and development has changed today, from Research Institutes to industry funded collaborative research. The days of large centralised Research Institutions practicing research for the sake of research have gone.

Industry is looking for a highly applied focus to gas and outburst related research, but we lack the Champions of the past.

ACARP is trying to address some of these concerns by moving to “landmark” projects to stimulate research but projects are often awarded by priority.
CONCLUSION

As the coal mining industry today strives to find safe and economic methods to understand and manage the phenomenon of gas and coal outbursts we lack the drivers of this work from the past.

We appear to lack the ability to attract, support and house potential Champions whether it is in industry, research organisations or academia. Unless we move forward and develop our understanding and management of gas and outbursts by developing greater knowledge and quantifiable research we will be forced to turn away from coal reserves that have this phenomenon. If these resources are seen as too difficult to mine, there will be limited recovery of our declining high quality coal reserves.
HISTORY OF OUTBURSTS IN AUSTRALIA
AND CURRENT MANAGEMENT CONTROLS

Chris Harvey

ABSTRACT: Outbursts have been recognized as an inherent, world wide mining related phenomenon since the 1850’s. The level of understanding has grown and developed as Mining Engineers and Geologists have been able to gain more understanding of coal seam characteristics and measure or test various coal or seam parameters such as seam gas content, seam gas pressure, coal strength and depth of cover. This paper outlines some of the concepts associated with understanding the factors, which contribute to outbursts and details more specifically the nature of outbursts experienced in Australia, especially for the Bulli Coal Seam. A number of key outburst incidents, which have had a distinct bearing on outbursts management concepts, are considered along with the current outburst management approach.

INTRODUCTION

The identification of the specific set of circumstances attributed to “outbursts” is believed to have followed from the observations of Taylor (1852-53). This study based upon experience in British mines identified three types of gas emissions, the first form of gas emission being characterized as the free gas, which is emitted from the coal to atmosphere and is in equilibrium with the atmosphere. The second form of gas emission was identified as being associated with highly compressible gas being present within the coal at high pressure. A particular characteristic of this gas emission is its slow release through the natural structure and pathways within the coal, creating the cracking, dislodgement and “bursting” of small pieces of coal as the gas is emitted. The third form of gas emission was identified as a variation of the second form and was associated with changes in the coal structure, which have disturbed or stopped the flow of the higher compressible gas. It was identified that the presence of faults and basalt intrusions caused the gas or the migration of gas through the coal seam to become irregular or non-uniform, resulting in “gas pockets” in certain areas. Taylor postulated that the mining of these areas would subsequently result in sudden emissions of gas and its sudden dispersion into the mine workings. Hence an early and accurate definition of outbursts was developed, which identified the role of geological structures, folds and dykes, changes in structure of the coal and the existence of high gas pressure.

The interaction of seam gas, (gas content in m³/tonne) and gas pressure, in combination with tectonic characteristics such as seam structures, dykes and faults has been identified as determining the outburst potential. Nekrashovski (1951) identified possibly for the first time that outbursts are not the result of a single factor but rather a multiplicity of factors acting together. It was identified that gravity has a role to play particularly for steeply dipping seams where the potential for outbursts to be initiated in roadways driven to the rise, is greater than for roadways driven to the dip.

CURRENT OUTBURST CONCEPTS

In general terms, it is universally recognized that an outburst is the sudden release of a large quantity of gas in conjunction with the ejection of coal and associated rock, into the working face or mine workings. The violent and unexpected nature of these events enhanced the risk and danger to mine workers. It is also recognized that numerous factors can contribute to the specific nature or characteristics associated with outbursts in a particular coalfield with the three primary factors being:

- Intense stress within the coal seam
- High gas content and high gas desorbability
- Low coal strength

1 NSW Department of Mineral Resources
Following from this and work by prominent industry researchers in Australia such as Alan Hargraves and Ripu Lama, the current information and understanding of outbursts has tended to involve the following factors.

- An inherently high in situ gas content for the coal seam associated with a rapid rate of desorption and greater depth of cover.
- Geological structures have a major link with the incidence of outbursts particularly strike slip faults.
- Compression type structures, such as reverse or thrust faults and strike slip faults have a greater potential to create mylonite, sheared and crushed coal than tensional structures such as normal faults.
- Mylonite and crushed coal has the potential to desorb very rapidly and when combined with localised stress conditions, which greatly reduce permeability, there is the potential for pockets of pressurised gas to be associated with structures.

This tends to reflect the general belief and assumption, based upon previous outburst incidents, that Bulli seam outbursts are largely considered to be a gas-dynamic phenomenon, rather than geo-dynamic. The necessary high strata stress component is related to geological anomalies rather than being a normal in situ seam condition induced by depth of burial, high lateral stress and mining induced stress. The latter being of particular note for outburst incidents at Leichhardt Colliery in Queensland. Hence any outbursts management strategy must reflect the multiplicity of facets or components, which characterise outbursts.

**AUSTRALIAN EXPERIENCE**

Outbursts have occurred in the Sydney Basin, Illawarra Coalfield and the Bowen Basin, Northern and Central Queensland. These two Basins are linked geologically and cover an area of 220,000 km², but have been defined separately upon regional and or local characteristics. Over 700 outbursts have been recorded within Australia over 106 years.

The Bulli Coal Seam, located within the Illawarra Coalfield to the south of Sydney, has the dubious honour of having the first recorded outburst in Australia at Metropolitan Colliery on September 1895. The various characteristics and peculiarities of Bulli seam outbursts are discussed in more detail below.

With regard to the Bowen Basin in Queensland, three collieries have experienced outburst incidents, these being mines in the Collinsville area along with Leichhardt and Moura Collieries. Both methane and carbon dioxide outbursts have been recorded with the primary characteristics being the presence of geological structures and a higher level of in situ seam gas. The physiological effects and characteristics associated with carbon dioxide have received particular attention especially in reviewing outburst related fatalities.

**Queensland Outbursts**

**Outbursts in the Collinsville Area**

Instantaneous outbursts in the Collinsville area were referenced by Hargraves (1958) and most recently by Lama and Bodziony, (1996). The seam outcrops on the northern rim of the Bowen Basin and has been subjected to various levels of deformation as identified in the nature and frequency of faulting, the rank of the coal and the wide spread occurrence of igneous intrusions. Low reverse angle thrust faults, in conjunction with bedding plane and slip strike faults are common. The Western extent of mining within the No.2 State Mine was limited or controlled by the Three Mile Creek Fault, which is a thrust fault with 400 metres of throw.

The first outburst incident for the Collinsville area occurred at the State Mine in 1954 at a depth of 250 metres. It resulted in the death of seven men and three horses, with carbon dioxide being the prominent gas. A royal Commission was held in 1956 following these fatalities. Since 1954 there have been nineteen outburst incidents recorded for the Collinsville area as documented by Williams and Rogis (1980). It would appear that as mining operations developed within the Collinsville area and it became more difficult to manage or control the outbursts, the mines were closed and re-started at shallower depths, near the outcrop. Consequently four separate mining operations have been undertaken in the Bowen coal seam, over a 50 year period, with operations being closed or restarting at shallow cover as the depth of workings approached the critical depth, ranging from 250 metres to 280 metres.

**Outbursts at Leichhardt Colliery**
Leichhardt Colliery is located within the Bowen Basin, to the South of the central Queensland town of Blackwater. As described by Hanes (1995) and Lama and Bodziony (1996), the colliery reported its first outburst incident in 1974 while mining the Gemini coal seam at an overall seam thickness of 6 metres (with an average working or extraction thickness of 2.5 to 2.8 metres) and depth of cover ranging from 350 metres to 410 metres. The local geology at the mine was regarded as structurally complex due to a number of faults of less than 10 metres throw, striking to the North West. The colliery workings were intersected by three prominent faults, two being steeply dipping normal faults of throw greater than 10 metres and the third being a shallow dipping reverse fault of 3.5 metres throws. Gas content of the Gemini coal seam was measured at 16 m³/tonne, being predominantly CH₄.

The outburst in 1974 occurred when mine workings had only developed 175 metres from the seam inset of the No. 2 shaft. In all, more than 200 outburst incidents were recorded at Leichhardt Colliery with three dislodging 300 tonnes or more of material. The largest outburst occurred on the shallow dipping reverse fault with minor displacement (~200 mm). The outburst which was associated with sheared and brecciated coal in the vicinity of the fault, ejected 500 tonnes of coal and approximately the same amount of rock.

Coal mining operations were undertaken using either a Joy 10CM or a Voest Alpine road header (used to cut a profiled roadway) until 1978 when following a major outburst, shot-firing was used. It was believed that outbursts at Leichhardt Colliery were stress controlled and gas induced. This was further complicated by mining induced cleavage and stress within the coal, which when combined with a directional permeability tended to create a higher than normal gas pressure gradient, based primarily upon the direction of orientation of the mine workings. The interplay between horizontal stress, (identified as being the maximum principal stress), vertical stress and mining induced stress was best described by Hanes (1995). The nature of mining conditions, gas content and ultimately the inability to successfully manage or control outbursts, all contributed to the mine being closed in 1982.

Outbursts at Moura Colliery

Moura No. 4 Colliery was an underground mine developed off the high wall of an open cut. The seam being mined had a total height of 5.2 m with the lower 2.5 metres being mined and a shale/sooty coal band, reported to be mylonitised coal, in the roof. Gas content for the seam was measured at between 8–9 m³/tonne with a maximum seam gas pressure of 1.03 Mpa (gauge) for a vertical depth of 135 metres (Troung et al, 1983).

Three outbursts have been recorded for Moura No. 4 Colliery with two of the three incidents being related to a major joint zone. The third outburst was believed to be associated with a zone of weak coal, uncharacteristic of normal condition in the mine. Generally outbursts were not considered to be a significant mining related problem due to the high strength of the coal, the low seam gas content and the slow desorption rate. In 1995 the mine was closed following a methane gas explosion, which resulted in nine fatalities.

Bulli Seam Outbursts

The first reported outburst incident in the Illawarra Coalfield occurred on 30 September 1895, according to Hargraves (1965). Details as to the size and intensity of this and other early incidents are very sketchy, however it would appear that all the early incidents (1895 to 1911) were associated with faults, dykes or zones of fractured coal, the discharge of both CO₂ and CH₄.

Table 1 gives an indication as to the number of outburst incidents attributed to each mine working the Bulli Coal Seam and the next section, provides some insight into specific details of outburst incidents, for particular mines. The outburst phenomenon was not treated with the same degree of importance as that exhibited in some coalfields throughout Europe and elsewhere in the world, due primarily to the comparative low fatality rate. To date there have been twelve outburst related fatalities for the Bulli seam and these have predominantly been associated with carbon dioxide, and seam structures.
Table 1  Bulli Seam Outbursts

<table>
<thead>
<tr>
<th>Colliery</th>
<th>No. of Outbursts</th>
<th>Size in tonnes</th>
<th>Gas</th>
<th>Geological Structure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Appin</td>
<td>26</td>
<td>2 - 88</td>
<td>mainly CH₄ with CO₂ on dykes.</td>
<td>Predominantly strike slip faults; mylonite zones.</td>
</tr>
<tr>
<td>Brimstone (closed)</td>
<td>2</td>
<td>30</td>
<td>CO₂</td>
<td>Mainly dyke related structures with strike slip movement.</td>
</tr>
<tr>
<td>Corrimal (closed)</td>
<td>4</td>
<td>12</td>
<td>CH₄ &amp; CO₂</td>
<td>Shear zone associated with minor faulting &amp; dykes.</td>
</tr>
<tr>
<td>Kemira (closed)</td>
<td>2</td>
<td>60 - 100</td>
<td>CO₂</td>
<td>normal fault with mylonite.</td>
</tr>
<tr>
<td>Metropolitan</td>
<td>154</td>
<td>Up to 250</td>
<td>mainly CO₂ with minor amounts of CH₄</td>
<td>Predominantly with dykes &amp; faults that exhibit slicken sides &amp; mylonite.</td>
</tr>
<tr>
<td>Bellambi West (South Bulli)</td>
<td>13</td>
<td>1 - 300</td>
<td>mainly CO₂</td>
<td>Strike slip faults with mylonite; dyke zones &amp; thrust faults.</td>
</tr>
<tr>
<td>Tahmoor</td>
<td>90</td>
<td>5 - 400</td>
<td>mainly CO₂</td>
<td>Mainly strike slip faults; with dykes (110° - 135°) &amp; thrust faults: mylonite usually present.</td>
</tr>
<tr>
<td>Tower</td>
<td>19</td>
<td>1 - 80</td>
<td>mainly CH₄</td>
<td>Mainly strike slip faults with dykes.</td>
</tr>
<tr>
<td>West Cliff</td>
<td>254</td>
<td>4 - 320</td>
<td>mainly CH₄ with CO₂ to the NE development</td>
<td>Predominantly strike slip faults (100° - 110°) with slicken sides &amp; mylonite; dykes and thrust faults have been associated with outbursts.</td>
</tr>
</tbody>
</table>

Mine by Mine Experiences

Appin Colliery
Appin Colliery has recorded 26 outbursts varying in size from less than 2 tonnes through to a reported 88 tonnes. The first outburst occurred in May 1966. It ejected 50 tonnes of coal along with an unknown but significant amount of CH₄ and was related to a zone of joints that were evident in the immediate roof. Five small outbursts mainly less than 8 tonne but one up to 20 tonne have occurred with no prominent geological structure. Strike slip faults tend to account for the majority of the outbursts along with one occurring adjacent to a dyke and associated with cindered coal. The largest recorded outburst occurred in July 1969, ejecting 88 tonnes of coal and a large amount of CH₄. Some 2 hours after the event 4% CH₄ was measured in the general body. This outburst was associated with a strike slip fault and a readily identifiable mylonite zone 0.05 metres wide.

Gas content at Appin is in the order of 13 m³/tonne and an extensive gas drainage system is used to prevent or minimise the risk of outbursts and manage gas liberated during mining. Composition of the gas is predominantly CH₄, however high CO₂ has been recorded adjacent to faults and dykes.

Corrimal Colliery
Corrimal Colliery, now part of Cordeaux Colliery, recorded four outbursts associated with a north easterly trending shear zone, minor faulting and dykes. The first outburst occurred in October 1967, ejecting 5 tonnes of coal and an unknown amount of gas. It was associated with a shear zone exhibiting strike slip faulting, crushed coal and mylonite as well as two thin dykes, less than 1m in thickness. The largest outburst occurred in November 1967 discharging up to 12 tonnes of coal with both methane and carbon dioxide in unknown quantities being given off. All the Corrimal outbursts were associated with a shear zone that bisected the colliery with outbursts being reported in the central and southern extent of the zone. The outburst prone areas were related to intense jointing with mylonite being present in lateral bands and within the cleat near the roof. This shear zone is a prominent geological structure and is recognisable on aerial photographs.
Kemira Colliery
Kemira Colliery, now closed, recorded two outbursts in May 1980 and May 1981 on a single normal fault of 0.4 to 0.7 metres vertical displacement. Mylonite was identified in bands and within the cleats near the roof at the outburst sites. Carbon dioxide was the predominant gas with 60 and 100 tonnes of coal being discharged.

Bellambi West Colliery
Bellambi West Colliery, formerly known as South Bulli Colliery, had its first outburst on a 110° "shear zone", located in the northern part of the mine. A small hole or cone near the roof was associated with the outburst. Further small outbursts have occurred along the same shear zone with 5-30 cm of mylonite being identified. Also in the northern part of the mine the dip of the seam changes from 1° to 10° with accompanying bedding plane shears in a claystone band located 10-15 cm from the roof. This acts as a preferred shear band and as many of the outbursts occurred at this level the sheared claystone area was regarded as having outburst potential. In the southern part of the mine an outburst occurred associated with a dyke zone. This had a breccia pipe inside the dyke zone and cindered coal. Significant quantities of carbon dioxide were liberated.

The fatal outburst at South Bulli Colliery on 25 July 1991 was also the largest recorded for the colliery and related to a low angle thrust fault with a 35 cm band of powdered coal. Approximately 2 metres of the face collapsed with the outburst and a cavity was formed in the right hand side of the face resulting in 300 tonne of coal being discharged. The gas liberated was predominantly CO₂ with high gas pressures being noted from drill holes used to prove the fault after the outburst incident.

Metropolitan Colliery
This mine has the longest history of outbursts in the Bulli Seam. Going back to 1895, it has recorded a total of 154 outburst incidents and has been associated with the greatest number of outburst related fatalities, (seven lives lost). A review of relevant reports and information indicates that the majority of the outbursts occurred on structures, especially a zone known at the mine as the “soft outburst zone”. Gas composition and gas content varies greatly throughout the mine, with the presence of faults and dykes being considered the primary cause.

Work undertaken by Hargraves (1965) showed a correlation between mining method, advance rate and outbursts. Inducer shot firing was used at Metropolitan Colliery as a means of initiating outbursts. The largest recorded outburst ejected 250 tonnes of coal and was induced by shot firing. Between 1961 and 1968 some 100 outburst incidents were believed to be related to inducer shot firing.

Zones of outburst potential can be plotted on the colliery plan based upon previous experience. With the last fatal outburst at Metropolitan being in 1954 it could be argued that management of the outburst risk was satisfactory. However, the most recent outburst in September 1992 had the potential to endanger mineworkers especially as the various warning signs while being clearly evident were not recognised by workers and supervisors.

Tahmoor Colliery
Tahmoor Colliery has had 90 outbursts since 1981 with the majority of them being associated with east south easterly structures, mainly dykes and strike-slip faults, with an orientation of 110° to 135°. It is believed that the dykes have been reworked with strike slip fault movement. A series of north easterly reverse faults have been associated with four outburst incidents and these structures have been difficult to drill by conventional rotary methods. The following is a summary of outbursts at Tahmoor:

<table>
<thead>
<tr>
<th>Structure</th>
<th>N° of Outbursts</th>
<th>Violent</th>
<th>Size (tonne discharged)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Across Dyke</td>
<td>3</td>
<td>3</td>
<td>5-400</td>
</tr>
<tr>
<td>Strike slip/dyke</td>
<td>28</td>
<td>17</td>
<td>5-120</td>
</tr>
<tr>
<td>Strike slip fault</td>
<td>55</td>
<td>18</td>
<td>5-100</td>
</tr>
<tr>
<td>Reverse fault</td>
<td>4</td>
<td>1</td>
<td>5-40</td>
</tr>
</tbody>
</table>

The largest outburst incident at Tahmoor colliery occurred in June 1985, ejecting 400 tonnes of coal and an estimated 4,500 m³ of CO₂ into the development heading, burying both the continuous mining machine and the shuttle car. In recent times gas drainage has been used and where gas content cannot be reduced quickly enough shot firing is the preferred approach.
Tower Colliery
Tower Colliery has recorded 19 outbursts, with the first incident occurring in July 1981 and so far this has also been the largest outburst. The size of outbursts has varied from less than 1 tonne to 80 tonne with unknown amounts of CH₄ being liberated. These have predominantly occurred in the south western part of the mine, against a dyke with associated strike slip faulting. Low intensity bumping and slumping has been experienced with outburst type conditions where seam gas content of 10-12 m³/tonne was recorded with gas composition being predominantly methane. A gas drainage system is currently utilised to reduce the in seam gas content and control gas during mining operations.

West Cliff Colliery
West Cliff Colliery had its first outburst in December 1976 and since that time 252 outburst incidents have been recorded. This first incident ejected 120 tonnes of coal and was related to a shear zone, with strike slip faulting and mylonite and this zone proved to be the site and focus for a number of subsequent outbursts. The size of the outbursts has varied from 4 tonnes to over 320 tonnes with the majority being related to zones of strike slip faulting having a strike approximately 100°-110°. The largest outburst, some 320 tonne of coal occurred at the northwest end of a normal fault where the gas drainage holes had not penetrated. There was a major joint zone 3-4 m wide in the roof associated with this outburst site and a mylonite band some 30 mm thick.

The gas composition has been predominantly methane but several outburst events in the north eastern part of the mine have involved very high levels of gas (>16m³/tonne) and >95% CO₂. Mining operations at this colliery have only been possible through the use of gas drainage and specified outburst mining procedures and a purpose built continuous mining machine to afford protection to the miner driver. This has minimised the risk of injury to the mineworkers and permitted mine development through many outburst zones.

West Cliff Colliery has the dubious distinction of recording the only outburst so far to have occurred on a retreating longwall face within the Illawarra coalfield. On 3rd of April 1998 two outburst incidents of low intensity occurred on the longwall face 23. The area where this occurred had not been adequately covered by gas drainage as the take off point for the face had been relocated to give an additional 45 metres of longwall coal. The outbursts were identified by 2 cavities or cones in the face at the roof extending about 1 metre into the coal. Gas samples taken after the incident but in close proximity recorded 98% CO₂ at a seam gas content of up to 21 m³/tonne. There was no apparent structure at the face and it was believed that the outburst occurred due to the high gas content, the localised stress conditions (including mining induced stress) and the extremely low permeability of the coal (Piper, 1998; Walsh, 1999). The West Cliff Outburst Management Plan has since been amended to ensure that all longwall panels are effectively pre-drained.

CONTROL AND MANAGEMENT OF OUTBURSTS

Early Concepts
Initially outbursts were considered just one of a number of factors or problematic conditions inherently related to mining coal at greater depth. The unexpected nature of outbursts was of concern and while the maintenance of boreholes in advance of the face was considered to be desirable, the primary and most effective early method of controlling outbursts proved to be inducer shot firing. In support of this type of approach special shot-firing controls or more appropriately regulations were established, especially following on from the outburst incidents experienced at Metropolitan Colliery and at Collinsville. To some degree the major benefit of this approach was to remove people form the immediate face area, limit the number of holes to be fired at any one time, limiting the detonator delay and stipulating a minimum level of ventilation.

As mining technologies changed the concept of predicting outbursts via advanced drilling, plotting geological structures and monitoring seismic activity were developed. These approaches related to recording or assessing one particular outburst component as an indicator followed by the timely removal of workmen to limit danger. Following from this approach was the provision of purpose built mining machinery to afford protection to the continuous miner driver. While certain aspects of these types of outburst control may still be evident in current day outburst management plans they generally have a different focus and are not the only control mechanism utilized at the mine.
Outburst Management

Outburst management differs from outburst control in that management relates to managing the outburst risk. This risk management approach will utilize outburst prediction and prevention techniques with the ultimate “fall back” being the protection of mine workers, from the consequence of outburst incidents. As such, it has a human focus rather than technological and would therefore involve procedures and processes all aimed at and achieving the management aim. The inter-relationship of outburst prediction, prevention and protection is explained diagrammatically in Figure 1. This type of approach was first brought to the notice of mines operating in the Bulli coal seam after the triple fatality at South Bulli coal mine in July 1991. It involved the concept that the management of risk associated with outbursts requires the establishment and maintenance of specific “barriers” (as identified in an Energy/Barrier Chart), which prevents the “energy” from being released into the mine environment by an outburst, and endangering mine workers.

In more general terms, it was identified that there was no one specific technology or mining technique, which could be used to guarantee safety in outburst prone mines. The effective management of outbursts to ensure safe working conditions, involved a number of techniques and technologies. These technologies including measurement of seam gas content and composition, identification of geological structures, use of gas drainage techniques, identifying in situ and mining-induced stress regimes when put together in a particular format or management system could effectively manage the outburst risk. The background concepts of management systems and more specifically, quality management concepts were fundamental to the development of outburst management plans.

Outburst Management Plans

The various characteristics, which lead to identifying the outburst potential, and subsequently to the outburst risk, have clearly been shown as site specific. Geological conditions and the mining methods utilised at any one particular mine site will necessitate the development of a specialised outburst management plan for that mine. This will not only reflect the techniques and technologies used at that mine to manage the outburst risk, but must reflect the “culture” at that mine, the way work is performed and the way mine management relates and interacts with the workforce. In this regard the human and organisational aspects of the plan can be considered as equal in importance to the technologies utilised, as the acceptance of an outburst management plan and its implementation is just as important as any other aspect of outburst prevention.
Fig. 1 Diagrammatic Representation Of Outburst Management
Fig. 2  Outburst Management Plans as a Safety Management System
While it is of primary importance that any outburst management plan is developed for each specific site with due regard for geological and mining conditions, for the plan to be considered as acceptable it must have a number of key elements. These elements are contained within guidance documentation as provided by the Department of Mineral Resources, Coal Mining Inspectorate, MDG 1004 (DMR, 1995). The manner in which the key components of an Outburst Management Plan fit within a Safety Management System is shown in Figure 2. This also shows how the General Requirements, Mandatory Elements and Process Requirements for each management plan inter-relate to ensure a safe mining outcome. The inclusion of the Corrective Action, Review and Audit processes are fundamental to the plan achieving its purpose and being successfully implemented.

In conjunction with the development of outburst management plans has been the enforcement via regulation, of gas threshold levels for all Bulli seam mines. While this has supported a systematic approach to reviewing outburst factors and assessing the outburst potential via the measurement of gas content, the most significant development has been the wide spread adoption of gas drainage to manage gas within the mining environment. There have been a number of additional “spin offs” resulting from this approach along with the attitude of that outburst mining is not an option and normal mining procedures will only be used.

The success of the Management Plan approach in combination with gas threshold values for Bulli seam operations is clearly exemplified in Fig. 3. This shows the major reduction in outburst incidents for the Bulli seam since the management mechanism was established for outbursts.

Fig. 3 Recorded Outburst Incidents Since 1990

CONCLUSIONS

It is a sad but understandable fact that the study and analysis of outburst events and characteristics has been directly related to outburst fatalities, especially multiple fatality incidents. Outbursts are a multi faceted mining phenomena and there is no single universal predictive tool. Similarly there is no one mechanism for the prevention of outbursts. Gas drainage has been the most successful outburst prevention technique, especially for the Bulli seam; though this is not universally the case and most probably would not have been of assistance in the case of coal with very low permeability (such as at Mt. Davy, NZ) and high in-situ stress.

The current approach, which utilizes management plans and directed gas threshold values, has greatly reduced the number of outburst incidents. The key benefits being the collection of data such as seam geology, gas content, gas composition, potential geological structures, lead time for gas drainage and correlation against gas threshold values. All these components are successfully combined to give a total image/picture of the coal seam immediately before it is mined. The most suitable mining method is then selected in the interests of safety and the
mine manager or appropriate senior mining official then signs off on the collected information and the mining method. However ongoing training in outburst awareness must continue to be the ultimate and fundamental protection for all face workers. The ability to identify key outburst warning signs and related changes in mining conditions can save lives.

It would appear that over and above the effectiveness of the current outburst management plans for Bulli seam mines, there is an undue reliance on one outburst indicator, namely seam gas content, as the primary standard or determinant for outburst risk. For the plans to become more adaptive and comprehensive, other factors, such as seam gas pressure, coal strength and permeability, need to be considered and incorporated within the outburst management plans. This reflects the overall belief and assumption that outbursts in the Bulli seam are largely a “gas-dynamic” phenomenon and as such, this does not easily accommodate variations within the seam and the mining environment that could cause Bulli seam outbursts to become “geo-dynamic” phenomena.

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GAS CONTENT TESTING FOR OUTBURST MANAGEMENT COMPLIANCE

Ray Williams¹

ABSTRACT: It is fundamentally important to make sure that compliance for mining is not issued on the basis of gas content tests that are inadequately located, deficient in frequency or in error. This paper covers some of the issues in relation to location and frequency of cores and validation of gas content test results. Core locations are discussed in the context of recent findings on the reduction of gas drainage efficiency towards the end of in-seam, cross panel boreholes. Some guidelines for sample frequency are given, with increased frequency being required in circumstances involving departure from “normal” conditions as defined through analysis of past data. The gas content validation methodology is set up to provide rulings on gas content test results at the time of reporting, and is intended to aid mine staff in better evaluating test data as part of the process of issuing management plan conformance notices.

INTRODUCTION

Routine gas content testing is the prime means of protection against instantaneous outbursts of coal and gas. In applying gas content results to mining, the following elements are important:

- The gas content threshold
- The area around a roadway to which the threshold applies (the “barrier”)
- Taking sufficient samples and in the right localities
- Minimising the chances that an error has been made in the gas content test

Gas content thresholds were initially determined for the Bulli seam on the New South Wales South Coast Lama (1995). They have been extended to the Hunter Valley, Bowen Basin and New Zealand (Mt Davy Mine) using GeoGAS’s Desorption Rate Index (DRI) approach (Williams and Weissman 1995, Williams 1997).

In the DRI approach, outburst proneness is regarded as being directly related to the desorption rate of the coal. Bowen Basin coals (Goonyella Middle and German Creek seams) have higher gas desorption rates than the Bulli seam and accordingly, the gas content thresholds are lower. For CH₄, the Bulli seam gas content threshold is 9.5 m³/t at 20°C and 1013 hPa m³/t at a DRI of 900. For the same DRI, the Goonyella Middle seam has a gas content of 7.0 m³/t and the German Creek seam (Middlemount/Tieri) a gas content of 7.7 m³/t.

After a period of 20 years without outbursts² in the Bowen Basin, an outburst occurred at Central Colliery (German Creek seam) on 20th July 2001 followed by one at North Goonyella mine (Goonyella Middle seam) on 22nd October 2001. While the outbursts were small, they were significant in being the first to occur in these seams and in areas deemed safe, according to locally applied gas and outburst management plans.

Investigations into the incidents included coverage of gas content threshold, barrier width, sample location, sample frequency, and gas content test reliability. At this stage there has been no modification of gas content thresholds away from the 900 DRI basis and for this paper, there is little point in reiterating the DRI approach which has been given wide coverage. Advances in outburst mechanism understanding and the definition of an improved basis for the setting of thresholds/triggers are the subject of current research by Xavier Choi and Mike Wold, CSIRO Petroleum to be reported on in papers at this conference and workshop. The width of the barrier around a roadway is relevant, but is currently the subject of client confidential studies by Strata Control Technology Pty. Ltd. and GeoGAS.

¹ GeoGAS Systems Pty Ltd
² Prior to 2001 the last outburst was at Collinsville No.2 Mine on 16th April 1981
The scope for this paper covers developments in two areas, these being:

- Taking sufficient samples and in the right locations
- Minimising the chances that an error has been made in the gas content test

**GAS CONTENT SAMPLING**

Within any area being evaluated for compliance, subjective decisions are made concerning sample location and frequency. Samples need to be taken in the area where the gas content is likely to be highest, based on gas drainage borehole geometry and borehole gas flow rates. Ideally, the sampling regime should be prescribed as a minimum standard to be met, with operator discretion only being employed to take more samples than the minimum.

**Sample Location**

Gate roads are normally developed after cross block drilling and gas drainage. The regions where gas drainage is least effective is toward the ends of boreholes. Here, gas recharge results in a reduction in gas drainage efficiency toward the end of boreholes as evidenced in gas reservoir modelling and corroborated to varying degrees, by field measurements. Current GeoGAS-ACARP research Project C10008 entitled “Improved application of gas reservoir parameters” has shown this “end-hole-effect” to be sensitive to borehole spacing (Fig. 1), directional permeability (Fig. 2) and virgin gas content magnitude. This can cause the heading on the virgin coal side which is B Heading in Fig. 1 to be exposed to higher gas contents, depending upon the amount of over-drilling of the cross block borehole and if down dip, the depth and location of perforations in the dewatering tubing. In Fig. 1 the “X” direction is parallel to the borehole and at right angles to the gate-road drivage. The “Y” direction is at right angles to the borehole and parallel to the gate-road drivage. It should be noted that “End hole effect” is insensitive to the magnitude of permeability.

Experience of the “end hole effect” is variable. For example, North Goonyella mine appears to be sensitive to borehole “end hole effect” whereas Central Colliery in normal drainage conditions appears to have less sensitivity. Given similar borehole spacings, this may be a reflection of differences in directional permeability.

For cross panel drilling, cores for gas content testing should be taken midway between the two boreholes and along the line of the barrier on the virgin side (Fig. 2). It should not be necessary to take samples below the barrier line (ie closer to A Heading), unless failure to comply occurs. In that case, additional samples can be taken along the line of a barrier defined around A Heading to validate A Heading alone.
The most important design requirement is to match the length of borehole over-drill beyond B Heading, with the barrier width and the end-hole-effect on gas content reduction, taking into account any additional deadening effect caused by the final location of dewatering tubing and the position of perforated sections. The end-hole-effect will probably be a function of borehole spacing and directional permeability. If over drilling is insufficient, there will be difficulty in achieving compliance along the barrier to B Heading.

Sample Frequency

Arriving at an optimum sample frequency requires examination of the past history, assessing the uniformity of results from test to test and linking that to environmental differences such as the time on gas drainage, ease of gas drainage and magnitude of gas content results in relation to the threshold value. Changes from the norm need to be carefully assessed.

Suppose compliance cores are routinely taken along a gate-road development with consistent placement with respect to the gas drainage-drilling pattern. Results are characteristically between 3 m$^3$/t and 5 m$^3$/t in an environment where the virgin gas content is 11 m$^3$/t and the gas content threshold is 8 m$^3$/t. What if the next sample gives a result of 7.5 m$^3$/t, which is below the gas content threshold? This result would be a departure from the norm and as such, should be a trigger to increase the sample frequency, in addition to seeking to understand what is happening.

Taking supplementary samples to check an abnormal result can be difficult. By the time the result is posted, the drill rig has probably moved to a new location. To minimise disruption it is important to plan the sample program to take account of potential problems identified during drilling and gas drainage. The consistency of borehole gas flows together with material balance calculations is a guide to the uniformity of gas drainage. Borehole gas drainage rates are shown in Fig. 3

In a fan of cross panel boreholes, additional samples can be planned for areas where gas drainage variability is above or below the norm for that area. For example, when a fan of boreholes behaves similarly, there is increased confidence that the drainage is proceeding according to plan (Fig. 3a). This can be backed up by material budget calculations and compared to expectations from modelling. Anomalous gas drainage requires more attention. For example, in Fig. 3b, there are three poor performing boreholes, two reasonable boreholes and one anomalously high flowing borehole. Apart from checking that gas drainage has been optimised by ensuring that there is no blockages or water accumulation an assessment of this area would require additional coring to confirm the state of drainage. This would hopefully be carried out early enough to enable timely remedial action. The high flowing borehole is probably the result of structural enhancement, so faulting should be suspected suggesting a review of drilling records.
RELIABILITY OF GAS CONTENT TESTING

A reliable gas content test result is probably the most essential ingredient in assessing an area for compliance with the management plan. Test failures do occur, and it is important that when they do, they are recognised prior to using the results. The most problematic part of the gas content test is connected with the field, “lost gas” (Q1) component. The main problem is leakage of canisters caused by not achieving a gas tight seal after placing core in the canister and/or failure of valves and fittings.

Persons who undertake the field Q1 testing are usually drillers or geologists. These persons need to be trained and certified as competent to carry out this important task. The training includes handling and final testing of canisters prior to use, avoiding contamination of the “O” ring seals and preparation for transport to the laboratory.

Validation of the gas content test results involves checking that leakage has not occurred during transport from the field to the laboratory.

The desorption rate or IDR30 which is the quantity of gas per unit mass desorbed in the first 30 minutes determined from the “Q1” measurements in the field and is related to the final reported gas content value Qm (Fig. 4). Normally, the faster the desorption rate, the higher the final reported gas content (Qm). Highly fractured core can produce a rate of desorption that is high compared to Qm. Other checks covered below are required to differentiate this condition. If a canister leaks between sealing in the field and receipt at the laboratory, Qm will be low compared to the IDR30, and the value will plot well down in the region of abnormally high desorption rate or potential leakage in Fig. 4.
When canisters are received in the laboratory, they are immersed in water for signs of leakage, then the pressure build up in the canister is measured. The canister is then depressurised and the volume of gas released called “initial Q2” or “Q2init” is measured.

Both the canister pressure and the volume of gas immediately desorbed (Q2init) upon release of pressure are a function of the –

- Mass of coaly material in the canister
- Void space in the canister
- The rate of gas desorption
- The time taken between sealing the canister in the field and relieving the pressure in the laboratory

These parameters are defined and are calculated for each test and a multivariate analysis undertaken. Equations defining initial Q2 and canister pressure are:

\[
Q2init = A_1 \times TCP + B_1 \times VS + C_1 \times M + D_1 \times DesRate + E_1
\]

\[
CanP = A_2 \times TCP + B_2 \times VS + C_2 \times M + D_2 \times DesRate + E_2
\]

\[
DesRate = \frac{1}{1 + k \times IDR30}
\]

Where:

- \(Q2init\) is the measured volume of gas desorbed upon depressurisation of the canister on receipt in the laboratory (ml).
- \(CanP\) is the measured canister pressure on receipt in the laboratory (kPa).
- TCP – Time in days that canister is pressurised between sealing in the field and receiving in the laboratory.
- VS – void space (ml)
- M – mass coaly material (g)
- IDR30 – initial desorption rate after 30 minutes from time zero (m^3/t)
- A, B, C, D, E and k are constants

Using the above formulae, a comparison of calculated and measured “Q2init” and canister pressure is made Figs. 5 and 6.

The tests whose values are denoted by “red” squares in Figs. 4, 5 and 6 have failed all three tests, according to where the threshold boundaries have been placed. “Failed” tests are can be supported by other evidence, such as visible signs of coal debris on the “O” ring seals of the canister.
The main point to undertaking these checks is to provide the mine with a reliability evaluated test at the time of first receiving the gas content results. A test valid that fails all three checks may still be, but it is definitely abnormal, and as such, warrants closer consideration.

The equations for each check are specific to each mine and are embedded into the spreadsheets for that mine, so that the rulings are made automatically as each test result is calculated. To date, only near pure CH₄ Bowen Basin coals have been assessed.

Additional reliability information is added to the gas content test report, covering core condition, evidence of any abnormalities and correction to a determined density. Density correction is especially important in thick seams, where there is a chance of reporting a low gas content due to a high ash sample being inadvertently taken.

CONCLUSIONS

While it is generally accepted that cores need to be taken in areas of suspected worst gas drainage, understanding the borehole end-hole-effect provides a basis for the sample location rationale. In addition to the direct use of core results for compliance, they should significantly aid in the better design of borehole length and spacing combinations.

Barriers against outbursting have historically involved differentiating normal from abnormal conditions. This approach is similar, involving identification of normal and abnormal conditions in relation to sample location, frequency and gas content test characteristics.

It is suggested that routine gas content compliance results be augmented by the validation methods described.

ACKNOWLEDGEMENTS

Gas reservoir modelling of end hole effects was undertaken by Dr. Eugene Yurakov of GeoGAS Systems Pty. Ltd. as part of ACARP project C10008 Improved application of gas reservoir parameters.

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GEOLOGICAL STRUCTURES IN RELATION TO OUTBURST EVENTS

Rod Doyle

ABSTRACT: In Australia irrespective of coal seam or coalfield, geological structures play a crucial role in the experience of having Instantaneous Outbursts. Identification of geological structures through concerted exploration activities, including drilling and remote techniques, or through the observations of both operators and geological personnel can allow for hazards to be identified and for precautionary measures to be implemented. This paper reviews; geological structures known to be associated with outbursts, some techniques used in the definition of such structures, and touches on the current procedures in place at mines that experience outburst phenomena or are concerned about such risks.

INTRODUCTION

In reviewing this topic geological structures that should be of interest are briefly explained. An excellent summary of geological structures is presented in Lama and Bodziony (1996). The authors provide information of interest from the global database.

Anyone undertaking underground investigations cannot help but be concerned about the serious effects of outbursts; the loss of life, the impact that fatalities have on the immediate families, the workmates and the local community.

UNDERGROUND GEOLOGICAL MAPPING

Every geologist knows that it can be a difficult task to appropriately and routinely map roadways in the underground environment. Thousands of linear metres may need to be reviewed in a short space of time, often under adverse conditions and sometimes estimates on the significance of structures are rapidly made - these assessments can at times be arbitrary. In the case of an outburst investigation, the reverse is usually true; a detailed investigation can be undertaken for a small area known to be of great significance. It should also come as no surprise that different people put different emphasis on different structures that are observed. (This is probably true of any scientific endeavour.) Nevertheless, whilst geologists may disagree over some of the minor features, the majority would generally agree about the major aspects.

However, trying to assess the risk of outburst potential prior to an event, from underground mapping alone, is a big ask. In hindsight, it is easy to say that the difference should have been picked up and mining operations ceased. If the reader accepts this as a reasonable proposition, then we have to ask ourselves, ‘Can we rely on mining operators to observe potentially dangerous situations associated with outbursts?’ The answer to this is equivocal.

Without appropriate training could miners have much hope of identifying such structures? Yet with a modest level of training and appropriate underground experience, coupled together with a good sense of being aware of the mining environment in which they work, there is a real opportunity to give forewarning of ‘changing conditions’. This aspect is without doubt one of the most important messages to get across to mining operators – the issue of changing mining conditions. What is different about our workplace today compared with yesterday?

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1 Dartbrook Mine, Anglo Coal
GEOLOGICAL STRUCTURES

The nature of geological structures that can have an impact on outbursting are varied. They range from the obvious structures such as faults and dykes to the not so obvious, including stress impacts, folding, shear zones. While it may and seem elementary there can be some benefit to the reader in having a simple description of structures in one small paper. Many of the deputy and undermanager exams often have such questions.

FAULTING

**Normal**
A fault is a planar structure with variable throw along its axis. It is helpful to visualise a fault in the following way – take a broadsheet of a newspaper and pinching it near the centre tear the sheet about 10cms in length so that the sides are still intact. This gives a 3D perspective of a normal fault plane. In the centre of the tear is the maximum throw of the fault and the throw of the fault diminishes towards its extremities.

In the 2D diagram below the Foot Wall (Block B) has moved or been thrown upward with respect to Block A (the Hanging Wall). The fault plane indicates that the two blocks have moved apart under tension. This fault is described as a normal fault.

![Normal Fault Diagram](image)

**Reverse**
In the 2D diagram below the Hanging Wall is upthrown with respect to Block A (Foot Wall). However, the angle of the fault plane indicates that the two blocks have moved towards each other under compression. If the angle of this fault plane is 30° or less with respect to the horizontal plane, the fault is termed a ‘thrust fault’.

![Reverse Fault Diagram](image)

**Strike Slip**
In the 3D diagram below the two blocks shown indicate that the vertical movement is zero, while the horizontal movement could be significant.

![Strike Slip Diagram](image)
Plate 1. A normal fault in the Bulli seam – Southern Coalfield - normally 2-3m thick. The effect of the fault has been to reduce the mining height to about 1m. The trace of the fault plane can be seen as the heavy black line (centre) with sandstone in both the roof and floor.

The level of horizontal movement may only be metres, but could be much more. At various collieries in the Sydney Basin movement at a scale of a few metres across dykes has been observed. At Dartbrook in the Hunter Tunnel there is also strike slip movement of a few centimetres across a dyke.

Mylonite, or more correctly fault gouge, is often associated with strike slip faults. Mylonite is the product of one block sliding against the other. This sliding movement naturally produces friction, which in turn produces the fault gouge material, which often displays slickensides. This crushed coal can easily be broken in the palm of an observer’s hand. Generally, it is darker than the coal and stands out well in unstonedusted ribs. It varies in thickness from millimetres to decimetres. It can contain more moisture than the normal level of water inherent in the coal.

Mylonite tends to form a natural barrier to the migration of gas and under confinement this ‘barrier’ remains in place. When mining takes place and the confinement is reduced, the barrier, being weak, is readily ejected releasing the ‘free gas’ that is in its structure together with whatever gas has built up behind that barrier and is then available for rapid movement. In many respects shear zones are effectively equivalent to strike slip faulting and mylonite zones.

Anticline / Synclines
The 2D diagram below indicates the nature of anticlines – hill like, and synclines – valley like. Here the naturally flat sediments are ‘folded’ to represent hills and valleys. Seam rolls and steep dips can be associated with such structures. These structures can be formed from a compressional event of large magnitude, for example, basin wide tectonic events, but can also be associated with smaller localised events.
When mined these folded structures can exhibit high stress and cause difficult conditions requiring extra support. This is particularly true in and around the points of inflexion (bending). An analogy would be to take a wooden ruler and bend it to the point of failure – it would look like an anticline or syncline up to the point of failure, but prior to failure, stresses develop in the ruler at the points of inflexion and in a simple sense the same stress events occur in a coal seam.

**Jointing – Fractures**

Jointing is usually described as being associated with contraction of a rock mass during the process known as diagenesis or for coal, coalification. What these words simple refer to is the process of how sediments change from the state that they were originally deposited e.g. sands or muds, changing to the solid rocks that we see. Jointing can also form as a response to tectonic activity such as folding. In one sense joints are like the cracks we see in concrete, when there are inadequately spaced or no expansion joints.

Jointing often increases in frequency towards a geological structure. In particular not only can more joints be observed, but the orientation of these joints tends to be near parallel to the structure rather than following the direction or orientation of the normal joint pattern. This can be one of the most significant tell tale signs that operators have at their disposal to note that mining conditions have changed.

More should be done to explore for geological structures but where presentation of outburst incidents is dependent on assessment by mining operators they should be trained in methods of observation.

Why do joints increase in frequency and change in direction? This comes about because the joints are formed by two different processes. As discussed above the normal joint sets are formed during the process of diagenesis, i.e. early on. It is not until much later when a dyke intrudes or a fault occurs that this extra jointing is formed in association with the major geological structure. The new joints can develop nearly parallel to the structure as the ground accommodates the event. The frequency of the joints reduces away from the structure. This is why noticing changing ground conditions is so critical in trying to identify outburst potential.

**Cleat**

Jointing and cleat are pretty much formed by the same processes. Cleat refers to the ‘fracture patterns’ present in coal which are generally formed during coalification. People often talk about face and butt cleats. The face cleat being the dominant structure in coal, which to some extent is pervasive – i.e. extends for some distance, often throughout the coal seam. Whereas the butt cleat tends to only extend in length between two main/face cleats i.e. limited distance. Cleat can also be formed during tectonic activity.

**INTRUSIONS**

**Dykes**

Dykes form a vertical barrier in a coal seam much like a dam wall holding back a reservoir of gas. A dyke forms when there is movement of magma from within the earth’s crust towards the surface. It often forms a wall like structure of varying thickness. The breadknife in the Warrumbungle National Park is a great example of a dyke.

During the intrusion stage the hot gases and fluids that precede the magma act like a fracturing device and either push the country rock apart in a hydrofrac manner or ingest some of the country rock. Igneous activity has long been associated with the presence of carbon dioxide. During this process the coal is often coked to a moderate thickness away from the dyke material itself. Coked coal is clearly a very tell-tale sign of igneous activity.
Plate 2. An igneous dyke (white) on some 2-3m thick running near parallel to a longwall face line, AFC at base and chocks to right.

Sills
Silling is another form of igneous intrusion that intersec

ts strata in the horizontal plane. Silling can cause severe deterioration of coal and often leads to areas of coal being abandoned. Silling is common throughout the various coalfields in New South Wales, but silling has not been associated with outbursting.

Igneous Plugs and Diatremes
These geological structures are associated with igneous intrusive events. They are vertical in nature and are generally cylindrical in shape. While several have been identified in underground workings, none have been associated with outbursting. For a detailed account of diatremes the reader is referred to Crawford et al 1980.

FINDING GEOLOGICAL STRUCTURES

Surface drilling investigations allow for an overview of stratigraphy and seam continuity and can indicate major structures. However small scale structures, e.g. dykes and faults <2.0m are very difficult to identify with strike slip faulting all but impossible. This is particularly true in a moderately deep underground scenario where budgets may only allow for 250m grid spacing of boreholes which would generally be regarded as closely spaced.

In delineating geological structures in any mining area there are several tools that are of major use, in the first instance. These would be supplemented with an overview of existing structures in the near vicinity (Regional Tectonic Setting) and a literature survey. Examination of information from adjacent mines, if available is also essential. Ward (1984) provides an excellent overview of the geological investigations that should be routinely undertaken to assist in defining geological structures and coal reserves.

Gravity surveys and satellite imagery are other tools that can prove useful.

Geophysical logging tools are an absolute must for the majority of surface drilling activities. Resistivity can be used down hole as well as cross-country. The so-called acoustic sonic tool is making a slow introduction into the industry, but it can be a very useful tool for identifying small-scale structures. Green (2001) reviewed the success and value of this tool.

Magnetics is an essential tool for locating igneous activity. It has, also been used with some success in determining faulting. Either aero- or surface magnetics can identify areas of high magnetic susceptibility in the rocks close to the surface and at shallow depths. This information can then be transferred into high-resolution colour plots that identify the nature of structure. Moloney and Doyle (1996) identify the success of such techniques).

Further reviews using this approach and applying a detailed interrogation of the data is presented in Munroe et al (2001). Magnetics affords the opportunity to utilize drilling with specific targets in mind.

Seismic surveys have been of great value in interpreting faulted ground and can sometimes identify dykes and silling along with synclines etc. This type of survey can range from the ‘wacker packer’ style to 2D or 3D high-resolution dynamite surveys. Much has been written of the success and Peters and Hearn (2001) have reported on...
recent finding in the Bowen Basin. Results from this type of work allow for specific targets to be focussed on to
gauge the accuracy of the interpretation.

The benefit of any of these techniques is in defining the presence, location and magnitude of geological structures. These structures can then be placed on a Hazard Map to be used when determining the likely impacts of both
development and longwall extraction.

**UNDERGROUND EXPLORATION ACTIVITIES**

The impact of in-seam drilling has increased markedly during the last decade. Some of this improvement results
from high quality survey tools giving rise to high-level confidence in the location of boreholes and their influence
on surrounding strata.

The downside is that despite the obvious benefits of having geophysical logs run in these holes, the development
of this technology for in-seam boreholes has not developed at an adequate pace. What also needs to go hand in
hand with the use of this technology is the realisation by mining operators of how important is the information that
the geophysical logs provide in locating geological structures. Ironically, for surface exploration boreholes, it is
unlikely that any geologist would ever choose not to run geophysical logs – nor would they allow the drillers to
write the borelogs for the hole, which is common practise throughout the in-seam drilling industry. I believe that
gеophysical techniques for in-seam work must be improved, further that it is critical in mines that have a specific
outburst risk.

Remote techniques such as Radio Imaging (RIM) have a role to play. RIM can give varied results in different
coal seams and in different coalfields. It is clear that in areas where it has been found to work, it can be employed
to identify; geological structures, clean coal, areas of high moisture and potential zones of high gas content.

**MANAGEMENT PLANS**

Many mines have identified the need to implement Outburst Management Plans (OMP). The plans, if adequate, if
followed and if audited, ensure that a specific process is in place to manage activities to avert the risk of having
outbursts.

Most OMP rely upon a combination of determining the in-seam gas content and identifying geological structures.
If the gas content falls below certain cut-offs, mining can progress without further work. If the gas content is
excessive than further in-seam drainage is required to reduce the level of gas prior to mining commencing. If a
structure is present then further drainage may be required.

Training of miners to identify geological structures, whilst essential, does not necessarily mitigate against an
outburst event taking place. It is simply a means of detection and a limited one at that.

It is also important to assess the level of risk with a risk assessment of the potential at any site. A statistical
analysis of the available gas data to assess the likely variability and determine the adequacy of the data should be

**CONCLUSIONS**

Geological structures have a critical role in the outbursting events. In Australia outburst events without geological
structures are extremely rare.

The role of structures appears to be twofold; providing disturbed ground and effectively creating a barrier to gas
migration or drainage.

Ongoing research of outbursting, should be encouraged.
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CURRENT IN-SEAM DRILLING TECHNIQUES

John Hanes

INTRODUCTION

In-seam drilling has two main purposes: gas drainage and exploration. Each hole can yield information on the geological structure of the ground drilled, but in the drilling of some gas drainage holes, valuable geological data is lost. Drilling technology has changed in some ways over the last ten years, but still has some way to go. There is room for more improvements to reduce the costs, to reduce the risks and to increase the information gathered.

DRILLING EQUIPMENT AND TECHNIQUES

Drill rigs have undergone some changes for the better in the last ten years. They have been increased in power and maneuverability for drilling longer holes. Examples of these rigs are the Boart Longyear LMC75 and the Cram RamTrak Diamec 262 with automated rod handling. Drill rods have been improved with the introduction of the Boart-Longyear NRQHP which has a totally new thread and better strength properties.

Hole surveying has also advanced considerably over the last 10 years. Currently the Advanced Mining Technologies' (AMT) survey tool, the DDM is the norm. Since 1994, AMT has sold approximately thirty-five directional drilling survey instruments to companies in Australia and approximately twenty to overseas countries, such as USA, Japan, China and Republic of South Africa. The majority of these tools have been DDM MECCA instruments. AMT recently developed the Drill Guidance System (DGS) and trialled it at Tower Colliery. The DGS allows for the addition of other geophysical tools when they are developed and approved. A profiler or indicator of proximity to roof/floor is required by industry. Sigra’s torque-thrust tool should also be a useful add-on when its output can be interpreted. A non-IS version of the DGS can be used for surface to in-seam. A major problem in getting new in-hole surveying and logging technology into the industry is the long delays in getting IS approvals. IS approval has been obtained for NSW for the DGS, but there have been considerable delays getting approval for Queensland. Why do we have two different approval systems especially for equipment manufactured to Australian standards?

Guided drilling is mainly conducted as “flip-flop” drilling: 6 m is drilled to the right then 6 m to the left. Gray (1998) advised that the main limit to hole length was the strength of the drill rod joints under tension during pulling of the rod string. Hole friction was a major factor. To reduce hole friction, he recommended that holes be drilled straighter using the oilfield technique of rotary-slide. This technique has been routinely used at Appin Colliery since early 1988.

At Appin drilling is conducted with a down hole motor (DMH) which is slowly rotated at 200 RPM. A 92 mm bit and a 1.25 degree bend are used. The bit stays central and does not wave about. The result is a gun barrel spiraled hole. The AMT Mecca system is used for survey and has not been affected by rotation. There is less friction in the hole and there is better flushing of cuttings. Hydraulic pressures are reduced leading to the ability to achieve longer holes if required. The maximum drillable lengths previously obtained with the Diamec drill rig were around 500 m to 600 m, but with rotary drilling with DHM, +700 m has been achieved. Penetration rates are better, having increased from 40 to 70 quality metres per shift. Without rig moves, an extra 15 m to 20 m are achieved. There have been no detrimental effects on the survey tool. Rotation speeds are 150 to 200 rpm, ie about half of normal rotary drilling rotation. There is very little lateral deviation of the holes compared with holes drilled by the flip flop method. Some drilling contractors refuse to use the technique stating that it subjects the downhole motor to too much vibration.

CMTE are currently working on an ACARP funded project, Project C 9020, to develop a non-rotating high pressure drill string used to advance a pure waterjet cutting head. The technique will use a conventional bent sub for directional control.

1 Coordinator of ACARP In-seam Drilling and Gas Research
Conventional rotary holes are still used, mainly for infill drilling and gas content core collection. Although BHP developed a monitored ProRam drill rig (Danell, 1999) and showed that it could detect outburst prone structures, the drill has not been used for detecting structures ahead of advancing faces, nor have other drills been fitted with monitoring equipment. This is a case of good research ignored by industry.

Drilling contractors and mine drillers are continually reviewing drilling methods to improve their methodologies and to reduce risks.

**DRILLERS**

The information which is gained from any in-seam hole is still completely dependent on the vigilance of the driller. An experienced and dedicated driller can detect even small structures through minor changes in drilling characteristics. The mine can only use this information if it is accurately recorded then properly interpreted. The Australian coal industry now has many experienced in-seam drillers. Tahmoor Colliery and BHP-Billiton’s South Coast mines employ their own drillers and equipment for most of their drilling requirements. Other mines use drilling contractors. The industry is well serviced, but better communication appears to be required to improve results and satisfaction.

**EXPLORATION VERSUS DRAINAGE**

Although most drilling is conducted for pre-drainage of gas, each hole can yield information on geological structures critical to mining continuity. Some holes are purpose drilled for exploration. Different drilling and data gathering techniques are required for each type of hole and there are different risks involved. Drainage holes are usually limited to across block distances up to 300 m length. There is generally little risk in these holes and they are drilled quickly, one recent example was the drilling of a 611m hole in a 10 hour shift. Exploration holes typically involve greater hole lengths (the record to date is 1602 m), slower drilling and delays for cuttings collection, and branching. Exploration drilling has a higher risk of equipment loss, especially in very long holes drilled into structured ground where up to $700,000 of equipment is in the hole.

To reduce the risk and cost of exploration drilling, an accurate prediction of seam structure and drilling conditions should be provided to the drillers prior to contract agreement. A competent geologist who can make decisions according to the information gathered during drilling should supervise the drilling. Drilling contractors report that most mining companies provide only sketchy information on predicted geology and leave the decision making and eventual blame if things go wrong, to the driller. These same companies would not allow surface exploration programs to be left up to the drilling contractors. During the 1980’s, in a longwall mine, several drainage holes drilled across the block bogged. The drilling was overseen by the mine engineers as part of the production process. The geologist was not involved or provided any information from the holes until a fault was later intersected by the shearer in the block, but not in the gateroads. A review of the drilling plan showed that the drainage holes had bogged on the fault and provided valuable information that was not interpreted. Several weeks of mining were lost while the face was relocated beyond the fault. This was a costly way of learning that successful mining comes from teamwork involving several disciplines.

**STRUCTURE DETECTION**

When most drilling was conducted with rotary drilling, structures were detected by bogging of the rods. With downhole motor drilling, the more powerful drill rigs allow drilling through smaller structures without bogging. A vigilant driller is required to detect these zones. The industry still requires automated logging during drilling to detect structures. The brightest hope comes from the Sigra torque and thrust tool which is being developed under ACARP funding (Project C7023). In laboratory tests, it has successfully detected minor differences in coal strength during drilling. Another potential is the borehole dielectric probe (Murray et al, 1999) which detects changes in moisture in the coal. To get such tools into the mines will require mine support for initial field proving then financial support for the construction and approvals. Mining personnel need to champion research projects or progress will be slow.
STICKY DRILLING

Although drilling technology has advanced, sticky drilling zones still challenge the best of drillers. These zones are typically associated with geological structures or stress concentrations. When the drill bit enters such a zone, the rods, through an uncertain mechanism, become stuck. Perhaps the stressed coal tightens about the bit and motor. Perhaps the soft coal caves and before being fully cleared by the circulating water, blocks the hole behind the motor. The result is hundreds of thousands of dollars worth of equipment left down the hole. Recovery attempts are time consuming and costly. If the equipment cannot be recovered, it does not take many losses to bankrupt a drilling contractor. Tower Colliery has a few such drill rod graveyards.

The Sigra borehole pressurization system, developed under ACARP funding (Gray, 1998) and currently seeking a trial site for proving, offers a possible solution to drilling through sticky zones. It allows drilling to be conducted under applied fluid pressure which internally supports the hole wall. This technique is successfully used in surface drilling. ACARP are currently funding CMTE research (Project C10016) into better methods for drilling through sticky zones and impermeable coal.

One contractor reported how he encountered sticky zones which were passed by drilling in the roof. A stressed zone was intersected which had not been predicted by the mine personnel. When mined, the coal in the stressed zone leaped out from the face and ribs (was this an outburst?). No faulting or other structure was obvious. The intended drainage hole became an exploration hole.

LOST GEAR RECOVERY

When drilling equipment is stuck in the hole, there are a few ways of attempting to recover it. Stories abound from the past of trying to pull the rods with a shuttle car or Eimco with varying success. Today, it is accepted that the best method is to overcore the rods and bottom-hole-assembly to free them. Recently, a set of gear was stuck at 960 m and then recovered by overcoring; this was a record.

CONCLUSIONS

In-seam drilling has advanced considerably in the last ten years in respect of the equipment used for drilling and surveying the holes and the expertise of the drillers. There is still much to be done to improve the collection of data from the hole for identification of geological structures and reporting of the data. The development of downhole probes for the detection of structures while drilling has been frustratingly slow.

ACKNOWLEDGEMENTS

The information summarized in this paper originated from my numerous colleagues who are in some way associated with in-seam drilling and who regularly share their knowledge and experience at the ACARP In-seam Drilling and Gas meetings. They identify the problems in drilling and then solve them, sharing the benefits of their knowledge throughout the industry.

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OUTBURST RESEARCH

Mike Wold

INTRODUCTION

The two books published by Dr. Ripu Lama (1995) and Dr. Lama and Dr. Jakob Bodziony (1996) are outstanding collections of literature, observations and original work in the field of gas outburst research and management. They provide a very solid base for further effort and developments; but they also serve to indicate that the problem is one of extreme complexity for which no ‘silver bullet’ solution is likely to be found. Since 1996, research has continued in several locations in Australia and overseas, generally stimulated by outburst events, or the perceived risk of occurrence in new areas or as mines go deeper.

This short presentation includes a bibliography of various publications in the recent outburst research literature (Appendix 1). These publications indicate that some good progress has recently been made both in Australia and elsewhere in developing a better understanding of outburst mechanisms, and in the author’s own experience, in developing quantitative models of those mechanisms. It is hoped that the bibliography provides a reasonably up-to-date introduction for the enquiring reader. It does not cover gas drainage, alleviation and outburst management. More comprehensive information can be found in the reference lists of the catalogued papers, for example in the ACARP report by Wold and Choi (1999) which contains a detailed assessment of research on modelling of outburst mechanisms.

The main body of this presentation comprises an overview of a research project that CSIRO Petroleum has recently commenced with support by ACARP. The project represents an attempt to provide a practical advance for the assessment of safe mining criteria for outburst prone conditions.

VARIABILITY OF COAL SEAM PARAMETERS FOR IMPROVED RISK ASSESSMENT FOR GAS OUTBURST IN COAL MINES

This project is directed to the ACARP Underground Health and Safety Program objective, Strengthen gas control systems – outburst.

The aim of the research is to:

- Improve the basis for gas outburst risk assessment.
- Provide a rational basis for extension of criteria for safe mining under gas outburst conditions, beyond the current criteria of gas content and gas composition.

This will be done by:

- Development of a statistical model of spatial variability of permeability and coal strength using new methods developed by CSIRO for the petroleum industry.
- Application of the new CSIRO outburst model to assessment of outburst conditions using the above parameters, and their variability distributions and correlations.

The major benefit of success in this work will be an improved ability to understand mechanisms, conduct risk analyses, and specify outburst threshold values (THV’s) that:

- build incrementally on the experience and safety record achieved over a number of years of mining in the Southern coalfields,
- are specifically designed for the particular conditions that prevail at the target coalfield or mine,
- assist in the provision of safe working conditions at increasing depth in the Bowen Basin,
- allow optimum mining development and production speeds,

1 CSIRO Petroleum
• provides a more quantitative basis for outburst management decision in the face of the new regulatory and legal framework governing safety and accidents.

The work will also demonstrate methods that will provide a blueprint for ongoing collection of data for outburst risk management.

In addition to the paramount issue of worker safety, and liabilities and costs of accidents, there is potential for substantial economic gains through defining optimised safe operating conditions.

The project will tackle the problem by collection and measurement of coal strength and permeability data, development of quantitative statistical models that account for their variability, application of the new CSIRO outburst model to determine sensitivity to these variables, and development of a quantitative risk analysis approach which considers these sensitivity measures.

DEFINITION OF THE PROBLEM

Mining of gassy coal seams such as the Bulli seam in the Southern Coalfield is subject to the hazard of gas outburst, and safe mining is governed by statutory criteria of gas content and gas composition. Application of these criteria has been successful in greatly reducing the incidence of outburst hazard in the Bulli seam, but it is not known whether the criteria might be overly restrictive when applied to some mining conditions, particularly when CO₂ is present.

Furthermore, in other areas such as the Hunter Coalfield and in the Bowen Basin, mines are now operating at increasing depth, with increasing gas contents, and in some cases with high CO₂ composition. The determination of safe working criteria for these mines is becoming a critical issue.

It is important to recognise that improvements in risk assessment must be done incrementally, based on the solid position established by the current methods. This is particularly so under the new regulatory and legal framework governing safety and accidents. Nevertheless, the development of new quantitative approaches to risk assessment appears to be the most promising way forward.

STATE OF THE ART

Efforts to understand and manage the outburst problem have been hindered by the complexity of the physical mechanisms involved, the difficulty in determining the various contributing factors and how they interact, and the need to continuously measure and monitor underground conditions as mining progresses. Nevertheless, in the last decade outburst risk has been brought under control in Australia by the introduction of in-seam gas drainage ahead of mine development and production. Under strict government regulation, drainage to meet safe gas content THV's, is carried out in all mines assessed as at risk. This concept is based on the work of Lama particularly with respect to the outburst problems at West Cliff Colliery (Lama, 1995). The regulations also mandate the use of outburst management plans, prepared and implemented at each mine. These are based on the recognition that sound management has a major and essential role to play, in conjunction with technical and operational procedures (McKensey, 1995).

In the management of outburst risk, the two prime issues are safety and productivity. Worker safety is an essential requirement with which there is no compromise. This underpins the setting of THV levels, and provides a fixed reference for discussion of mine productivity issues. Application of the THV criteria has been successful in virtually eliminating outbursts. However, the criteria are limited to the factors of gas content and gas composition, whether CH₄ or CO₂, with modified mining methods required in the close presence of major coal structures.

It is widely recognised that other physical factors have the potential to modify the risk, but a much-improved understanding of how these factors interact and contribute to the evolution of outburst conditions is required before they could be taken into account. The THV criteria were developed based on West Cliff mine experience with the Bulli Seam, and are applied to other mines such as Appin, operating under similar conditions (Lama, 1996). Technical arguments may be put to modify the criteria on the basis of comparative conditions, but the high safety level must be maintained.

If it is considered that a gassy mine might operate safely with increased THV's, there is the potential for increased development rate and reduced gas drainage costs. In marginal economic operating conditions, this could impact on
total mine viability. In undertaking a risk analysis for this purpose, both operational and geomechanical-reservoir factors must be considered, with weighting factors being assigned to a number of variables in a decision tree process. Methods that improve the quantitative basis for weighting factors, and broaden the range of variables that can be quantified, may therefore contribute to maintenance of safety standards while increasing productivity and viability.

**CSIRO OUTBURST MODEL**

A new model that can quantitatively simulate the evolution of outburst initiation has been developed by CSIRO, supported by ACARP (Wold and Choi, 1999; Choi and Wold, 2001a,b).

The initiation of outburst depends on the complex interaction of some important processes and factors. These control the evolution of a range of reservoir, geomechanics and fluid-dynamics field variables. The main processes and factors may be broadly categorised under the headings of

- gas desorption and two-phase fluid flow,
- effective stress and poroelastic effects,
- mechanical strength and geological structure,
- time-rates of mining and drainage,
- energy in free gas,
- coal fragmentation and fluid-particle interaction.

The current model development was undertaken to produce an improved understanding of outburst mechanisms, and to quantitatively model the influence on risk of outburst of a range of geomechanical, reservoir and operational factors. The method adopted was to couple a geomechanical model and a CBM reservoir model.

Model development and applications have been matched where possible to insitu data, observations and operations, taking into account broad natural variability. The main model variables are as follows:

- opening geometry and mining advance rate,
- vertical stress based on depth; horizontal stresses based on field measurement and depth,
- intrinsic permeability and permeability anisotropy based on field measurement,
- desorption isotherms from laboratory measurement; initial reservoir pressure based on depth, or reduced to various pressures to represent drainage or under-saturated conditions,
- sorption times based on production well history matching,
- CH$_4$/CO$_2$ composition in the range 0-100%; and coal strengths from laboratory measurements.

**RESERVOIR HETEROGENEITY AND GEOSTATISTICAL MODELS**

Permeability and strength values in coal can vary strongly over short distances, undergoing step changes often associated with presence of features such as bright and dull bands, cleat and fractures at various length scales, and mineralisation in the fractures. This heterogeneity poses problems in trying to estimate the permeability and strength properties that take effect at various length scales. The question of how to upscale measurements made on core to represent the behaviour of the coal face with dimensions of metres involves the application of statistical methods which account for the abrupt changes in properties, and their distribution in space. This has obvious importance when trying to understand mechanisms of outburst failure and the risk of its occurrence, particularly when considering one set of seam conditions compared with those at another site. Experience with the CSIRO outburst model has shown that permeability and strength are major determinants in outburst mechanisms. Therefore, in seeking to incorporate consideration of permeability and strength in assessing outburst risk, this project aims to quantify the variability at a scale suitable for the outburst model to handle, typically at a scale of 1-2m.

Geostatistics offers a collection of deterministic and statistical tools aimed at understanding and modelling spatial variability. Generating stochastic realisations of reservoir and geomechanics properties with a suitable level of spatial correlation in the values of permeability and strength is one of the more difficult challenges in petroleum statistical modelling. CSIRO Petroleum has had success in quantifying permeability variability using both
standard geostatistical models and by developing non-conventional models which better account for high levels of heterogeneity, in particular using Levy Fractal models (Liu, et al. 1996).

The field measurement components of this project are designed to provide data that can be analysed using geostatistical tools that appropriately model the spatial variability. If valid models can be built, comparative studies between different mine conditions may provide better risk assessment for safe mining conditions.

FIELD AND LABORATORY MEASUREMENTS

Permeability of coal can be measured in situ using well testing methods familiar to the petroleum industry, and in laboratory on core samples, but examples from the coal mining industry are fairly sparse. CSIRO has had more than decade of experience in the development and application of field and laboratory methods for measuring permeability, strength and stress in coal and porous sedimentary rocks. A recent example is provided by Wold and Jeffrey (1999). This study included a well interference test from which permeability anisotropy was quantified. However, strength and permeability in particular may have high spatial variability, and no evidence has been found in the literature of attempts to quantify this variability for coal seams.

In this project, the majority of the laboratory tests will be done in-house by CSIRO using standard techniques and equipment. Of particular interest will be the application of the rock strength device to measure strength parameters. This a portable device that could be readily deployed at a mine site to provide rapid turnaround of results from core with no special preparation required. This aspect may be of particular importance in practice, for ongoing outburst risk management by mine operators at a future time. The field well tests will be done in collaboration with a consultant. The reservoir model SIMED will be used in interpretation of results from the well tests, which are of novel configuration.

RISK ANALYSIS

Broadly speaking, the goal of risk analysis is to help the decision-maker choose a course of action, given better understanding of the outcomes that could occur. Risk analysis provides some qualitative and/or quantitative methods for assessing the impacts of risk on decision situations, and there are many different approaches that might be taken. Conceptually, sensitivities determined from the outburst model results could contribute to the assessment of probability of initiation of outburst event as a function of the permeability and strength variables. However, the scope of these considerations will be limited to comparative and incremental risk from the base position of the existing THV’s that result from a decade of experience in outburst management and control.

CSIRO Petroleum has formed a specialist risk analysis group that has commenced working across a range of industry problems.

SAFETY IMPLICATIONS

The ACARP Program recognises the need to strengthen gas control methods with respect to gas outburst, within the underground health and safety program. The core issue of this project is healthy and safe working conditions in gassy coal seams. It is implicit that the proposed methods developed will incorporate safety standards to at least the same level as currently exist. It seeks to promote increased certainty in setting safety levels for mining under increasingly gassy conditions, in newer areas.

BENEFITS TO COAL PRODUCERS AND DELIVERABLES

The major benefit of success in this work will be an improved ability to understand mechanisms, conduct risk analyses, and specify outburst THV’s that:

- build incrementally on the experience and safety record achieved over a number of years of mining in the Southern coalfields,
- are specifically designed for the particular conditions that prevail at the target coalfield or mine,
- assist in the provision of safe working conditions at increasing depth in the Bowen Basin,
allow optimum mining development and production speeds,
provide a more quantitative basis for outburst management decision in the face of the
new regulatory and legal framework governing safety and accidents.

The work will also demonstrate methods for the measurement and analysis of permeability and strength, which
will provide a blueprint for ongoing collection of data for outburst risk management.

Estimating the value of benefits to be gained depends on the scenario chosen, and those with knowledge and experience of the issue could readily do this. However, a simple example is indicative. If at a mine, development rates are slowed by gas drainage to meet excessively stringent THV requirements, such that overall longwall production is impeded by a total of five days in a year; then an estimated 50,000 tonnes of production would be lost at a net value of greater than $1,000,000. Conversely, a fatality caused by outburst where too lenient THV requirements applied is completely unacceptable.

DELIVERABLES

The project will produce a number of strategic deliverables and a number of technical deliverables.

Strategic deliverables:

• Rational basis for extension of existing criteria for safe mining under gas outburst conditions
to include other key variables.
• Improved basis for outburst risk assessment using quantitative measures.

Technical deliverables:

• Method demonstrated on site for measuring permeability at a length scale appropriate to the
outburst problem.
• Demonstration of portable test equipment for rapid and inexpensive measurement of coal
strength.
• Improved quantitative understanding of spatial variability of permeability and strength.
• Improved quantitative understanding of the sensitivity of outburst mechanisms to variations in
coal permeability and strength.
• Improved approach to quantitative risk assessment.

CONCLUSIONS

The work in outburst research of Dr Ripu Lama and others has provided a very sound basis for ongoing
developments and applications of research in gas outburst management. Australian and international researchers
are continuing to work in the field, and several projects are currently being sponsored by ACARP. Work by
CSIRO Petroleum is directed towards improving the technical foundation for gas outburst risk management, based
on a program of field measurement, numerical modeling and risk analysis.

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APPENDIX I
Bibliography of Recent Outburst Research


HYDRAULIC FRACTURING APPLIED TO STIMULATION OF GAS DRAINAGE FROM COAL

Rob Jeffrey

INTRODUCTION

Hydraulic fracturing is routinely applied to stimulation of oil, gas, and coalbed methane wells around the world. The stimulation effect is achieved in coal seams as in other reservoirs, by producing a conductive fracture, connecting the well to the coal reservoir. The conductivity of the fracture is usually maintained by placing a round and sieved sand proppant in the fracture channel. The proppant prevents the fracture faces from closing back completely on one another after the treatment. The design of the fracture treatment, therefore, centers on selecting fluids, injection rates, and slurry concentrations that will produce the desired propped fracture channel.

HYDRAULIC FRACTURE GROWTH IN COAL

A hydraulic fracture is produced by first isolating a section of the wellbore using either perforations through selected intervals of the well casing or some sort of packers. The fracturing fluid is then pumped through an injection string into the isolated section, causing the pressure to increase until a fracture opens at the borehole wall. Continued pumping then forces fluid into the fracture, which pressurizes it, causing it to open and extend deeper into the coal. Initially the fracture grows quite quickly, but as the treatment continues, more and more of the fluid injected at the wellbore is lost from the hydraulic fracture into the surrounding coal. This fluid loss is one of the most important processes that controls how fast and how far the fracture will grow. The amount of fluid that leaks off can be controlled, in part, by selecting different fracturing fluid, water, gel, or foam and by varying the injection rate. Because coal permeability is stress dependent, the leakoff process in coal is non-linear. As fluid is lost from the hydraulic fracture, the pore pressure in the coal around the fracture increases, which results in an increase in the permeability of this coal, contributing to additional leakoff.

Non-linear leakoff arises because of the naturally fractured nature of the coal seam. A second important aspect of hydraulic fracture growth in coal also results from the natural fractures in the coal. As the hydraulic fracture grows through a naturally fractured rock, it propagates along and across the natural fractures. The hydraulic fracture channel formed then develops branches and offsets along its extent. This complex hydraulic fracture geometry in coal has been documented by mining and mapping the propped fracture formed by the treatments (e.g. Figure 1). T-shaped branched fracture geometries, which often form at material property boundaries, are also commonly produced, but not often designed for.

In contrast to the multiple branched fracture shown in Figure 1, hydraulic fractures that are relatively planar may form, as shown in Photo 1. The nature of the fracture formed by a treatment is strongly dependent on properties of the coal seam being treated such as the existence of natural fractures, faults and shears, and the in-situ stress conditions. Fracture geometry near the borehole or well will depend on the orientation of the borehole with respect to the natural fractures and the in-situ far field stress. Use of thicker, more viscous fluids, such as crosslinked gels and foams, are believed to reduce the amount of fracture complexity compared to using less viscous fluids like water. However, coal is chemically active and exposure to organic polymers in gels may cause damage to the coal permeability. Correctly formulated and tested fracturing fluids and fluid breaker systems are required to avoid damage to permeability and fracture conductivity.

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1 CSIRO Petroleum
Fig. 1: Vertical section through a propped hydraulic fracture in the Great Northern seam at Munmorah Colliery, NSW.

HYDRAULIC FRACTURE STIMULATION

Smith and Shlyapobersky (2000) list three reasons for carrying out hydraulic fracture stimulations:

1. to bypass near-wellbore damage,
2. to form a conductive channel in the reservoir,
3. to change the fluid flow path in the reservoir.

Near-wellbore damage is caused by a number of factors, which include stress concentrations around the well and drilling induced damage such as cuttings and drilling fluids plugging the permeability pathways around the borehole. In addition, the permeability of the coal may be damaged during production if fines migrate in the seam or if precipitants form near the borehole because of pressure and associated water chemistry changes. This damage is characterised by what is called the wellbore skin effect and is one of the parameters measured by injection or production well testing.
By placing sufficient proppant in a hydraulic fracture that extends a sufficient distance, a conductive channel is formed in the coal seam that acts as a conduit for the water and gas to travel along. The fracture faces expose a large area of the seam to the lower producing pressure, allowing the water and gas to drain directly into the propped fracture at an accelerated rate. Hydraulic fracture treatments are designed to place a propped conductive fracture in the coal seam that will efficiently stimulate production from the seam. The stimulation effect achieved depends both on the conductivity and size in length and height of the fracture and on the permeability and thickness of the coal seam. Effective stimulation of a low-permeability seam requires longer moderate conductivity hydraulic fractures, while stimulation of a high-permeability seam requires shorter high-conductivity fractures.

The fractures which are formed drain fluid and reduce the pressure in the seam around them. Fewer wells or boreholes are then required to drain the seam or, alternatively, the seam can be drained more quickly using the same number of holes. In addition, fractures placed in horizontal holes serve to connect the borehole with over and underlying coal by forming conductive channels through stone or dull unfractured coal layers.

FRACTURING VERTICAL WELLS FOR COAL SEAM GAS DRAINAGE

Vertical wells drilled in advance of mining to drain seam gas require stimulation to accelerate the drainage process and to allow fewer wells to effectively drain the area targeted. A typical distance between wells might be 200 to 400 m. Hydraulically fractured wells at this spacing might require five years or more to drain 50 percent of the gas in place. Closer spaced wells drain the gas more quickly, but the total costs of drilling, completion and operating rapidly increase. Therefore, using vertical wells to drain gas before mining requires significant lead-time and upfront investment. There is good scope for mines to partner with a coal seam methane producer to reduce the cost to the mine significantly. Hydraulic fracturing is routinely used to stimulate coal seam methane wells and experience here and in the U.S. indicates that the effect of the hydraulic fracture on eventual mining of the seam is negligible (Jeffrey et al., 1997, Jeffrey et al., 1998, Diamond and Oyler, 1987).
FRACTURING HORIZONTAL DRAIN HOLES

Horizontal wells are drilled and hydraulically fractured in oil and gas reservoirs. The fracture treatments are undertaken to stimulate production and connect the horizontal well into layered reservoir formation. The horizontal layering in the reservoir invariably imparts a permeability anisotropy to the rock. The vertical permeability is typically significantly lower than the horizontal permeability. In addition, hydraulic fractures bypass the near wellbore damage zone, which can be a significant factor in reducing the productivity of any horizontal well or drainage borehole.

Hydraulic fractures can be placed in horizontal drain holes by running inflatable straddle packers on an injection string. Fluid bypass or even fracturing of the coal under the packers may occur (Jeffrey, 1999). Several trials of placing hydraulic fractures in coal seams have been carried out (Croft, 1980, Kravits, 1993, Jeffrey, 1999) with some success reported by Kravits. Special pumps and blenders are needed if sand is included in the treatment, but some stimulation effect can be achieved using only water. If proppant is pumped, the treatments must be large enough to extend and open a fracture to a width sufficient to accept the sand before the slurry stage is pumped. The benefit of including sand is in the larger stimulation effect or higher conductivity that can be achieved and the potential to reduce the number of horizontal holes needed to drain a volume of coal. The potential for losing the straddle packer system in the hole will vary with seam and borehole conditions, but can be a significant cost in some cases. Alternative methods of fracturing horizontal holes have been developed in the petroleum industry and might be adapted to fracturing horizontal drain holes in coal.

CONCLUSIONS

Hydraulic fracturing can be used to place a high conductivity channel in the coal seam. The conductive channel stimulates gas and water drainage rates by bypassing near borehole damage and forming a low pressure drain in the coal. As a result, gas drainage rates are increased.

REFERENCES

OVERVIEW OF GAS OUTBURSTS AND UNUSUAL EMISSIONS

Ripu Lama¹ and A Saghafi²

Abstract: Gas and rock outbursts are unwanted complications of underground coal mining, which have occurred over the last 150 years of underground coal mining worldwide and are still occurring. ‘Outburst’ is a dynamic phenomenon that causes the sudden concurrent release of gas and strata energy. The released energy causes pulverization of large amounts of coal and rocks, which are then ejected into the working areas during mining of the outburst prone zone. This paper discusses some of the 30,000 outburst events recorded worldwide and suggests indices to identify the outburst zones as well as methods of management of outbursts. The conclusions are based on overseas and Australian experiences particularly research carried out by the authors in the early 1990s in coalfields of the Illawara area.

INTRODUCTION

Outbursts and abnormal gas emissions in mines are a manifestation of conditions associated with high gas contents of the coal seam mined and the seams surrounding it. An outburst is an event where coal and/or other rocks are ejected from an advancing face together with the emission of large amounts of gas. The phenomenon manifests itself over a time, ranging from a fraction of a second to a few minutes. It consists of a series of events occurring in succession. The amount of coal and rock material ejected can vary from a fraction of a tonne to several thousand tonnes. The largest outburst in the world occurred at a Gagarin Colliery in the Donetsk coalfield (Russia) where 14,500 tonnes of coal was ejected together with 60,000 m³ of methane (Stepanovich et al, 1976).

There is a long history of outbursts of gas and coal in underground coal mining and they have occurred in most coal producing countries of the world (Table 1). In the last 150 years, almost 30,000 outbursts have been recorded, with the largest number, almost one half, occurring in the People's Republic of China.

High gas emissions occur without the ejection of coal or rock when the coal is permeable and when stress levels are low and strength is high. Coal seams, that at lower stress levels show high gas emissions, invariably experience outbursts when the strength of coal is low or stress levels are high.

Gas outbursts are associated not only with methane gas, but also with carbon dioxide. Outbursts associated with carbon dioxide are more violent, more difficult to control and more dangerous because of the greater sorption capacity for carbon dioxide. Most outbursts in the world, however, are associated with methane, which is formed during coalification process. Outbursts occur more in seams of high rank (bituminous, anthracites and semi-anthracites), because high rank coals have, greater capacity to adsorb gas at a given pressure, higher internal surface areas, lower porosities and lower permeabilities. A few outbursts have occurred in some lignite mines such as the Valenia mine in Slovenia.

Whilst methane present in a coal seam is either generated during coalification or by microbial processes or both, carbon dioxide is usually derived from an outside source such as magmatic activity. Carbon dioxide permeates into the coal seams, together with the circulating fluids through faults, intrusions dykes and major joint systems. At places, this may completely displace the inherent methane. Gas compositions in the vicinity of these structures can thus vary from almost 100% methane to almost 100% carbon dioxide if a coal field has been affected by igneous intrusions. So far outbursts of carbon dioxide have been experienced only in Australia, Poland, Canada, Czech Republic and France.

¹ This paper was prepared during the last months of 1996. Ripu passed away in January 1997.
² CSIRO Energy Technology
Table 1 - Occurrence of outbursts in various countries (Bodziony and Lama, 1996)

<table>
<thead>
<tr>
<th>Country</th>
<th>Mine field/s</th>
<th>Coal/ Rock burst</th>
<th>Gas type</th>
<th>No. of outbursts experienced</th>
</tr>
</thead>
<tbody>
<tr>
<td>Australia</td>
<td>Sydney basin</td>
<td>Coal</td>
<td>CH$_4$ + CO$_2$</td>
<td>&gt; 669</td>
</tr>
<tr>
<td>Belgium</td>
<td>Southern coalfield</td>
<td>Coal</td>
<td>CH$_4$</td>
<td>487</td>
</tr>
<tr>
<td>Bulgaria</td>
<td>Balkan</td>
<td>Coal</td>
<td>CH$_4$</td>
<td>250</td>
</tr>
<tr>
<td>Canada</td>
<td>Crows Nest</td>
<td>Coal + rock</td>
<td>CH$_4$ + CO$_2$</td>
<td>411</td>
</tr>
<tr>
<td>China</td>
<td>Large number of coal fields</td>
<td>Coal + rock</td>
<td>CH$_4$</td>
<td>&gt; 14,297</td>
</tr>
<tr>
<td>Czech Republic</td>
<td>Ostrava Slany</td>
<td>Coal + rock</td>
<td>CH$_4$ + CO$_2$</td>
<td>482</td>
</tr>
<tr>
<td>France</td>
<td>Various</td>
<td>Coal and other rocks</td>
<td>CH$_4$ + CO$_2$</td>
<td>&gt; 6,814</td>
</tr>
<tr>
<td>Germany</td>
<td>Ruhr Ibbenbüren</td>
<td>Coal and other rocks</td>
<td>CH$_4$</td>
<td>359</td>
</tr>
<tr>
<td>Hungary</td>
<td>Mecsek</td>
<td>Coal</td>
<td>CH$_4$</td>
<td>~ 600</td>
</tr>
<tr>
<td>Japan</td>
<td>Hokkaido + Kyushu</td>
<td>Coal</td>
<td>CH$_4$</td>
<td>920</td>
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<td>CH$_4$ + CO$_2$</td>
<td>1,738</td>
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<td>Anima-Resica</td>
<td>Coal</td>
<td>CH$_4$</td>
<td>20</td>
</tr>
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<td>South Africa</td>
<td>Main Karoo</td>
<td>Coal</td>
<td>CH$_4$</td>
<td>5</td>
</tr>
<tr>
<td>Russia *</td>
<td>Various</td>
<td>Coal and other rocks</td>
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<td>UK</td>
<td>Various</td>
<td>Coal</td>
<td>CH$_4$</td>
<td>&gt; 219</td>
</tr>
</tbody>
</table>

There is no clear relationship between the frequency and size of an outburst and various parameters that influence outbursts (Cyrul, 1992). The size of an outburst however is related to the size of a geological structure on which it occurs. All other factors remaining constant, depth (and stress) increases the size of an outburst but at a relatively low rate (Bodziony and Lama, 1996). The minimum depth at which an outburst occurs depends on specific local conditions (Table 2). At Ibbenburen Colliery which is the deepest anthracite coal mine of Germany, located in the Ruhr coalfield outbursts were recorded from depths of 1150 m and more.
Table 2 - Outburst depth in various countries (Bodzinoy and Lama, 1996)

<table>
<thead>
<tr>
<th>Country</th>
<th>Largest quantity of material ejected</th>
<th>Minimum depth at which outburst started, m</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Rock + coal, t</td>
<td>Gas, m³</td>
</tr>
<tr>
<td>Australia</td>
<td>1,000</td>
<td>14,000</td>
</tr>
<tr>
<td>Belgium</td>
<td>1,600</td>
<td>34,000</td>
</tr>
<tr>
<td>Bulgaria</td>
<td>350</td>
<td>12,000 - 19,000</td>
</tr>
<tr>
<td>Canada</td>
<td>3,500</td>
<td>60 - 140,000</td>
</tr>
<tr>
<td>China</td>
<td>12,780</td>
<td>3.5 x 10⁶</td>
</tr>
<tr>
<td>Czech Republic</td>
<td>4,310</td>
<td>96,000</td>
</tr>
<tr>
<td>France</td>
<td>330</td>
<td>400,000</td>
</tr>
<tr>
<td>Germany</td>
<td>2,500</td>
<td>66,000</td>
</tr>
<tr>
<td>Hungary</td>
<td>1,800</td>
<td>27,000</td>
</tr>
<tr>
<td>Japan</td>
<td>5,200</td>
<td>600,000</td>
</tr>
<tr>
<td>Kazakhstan</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Poland</td>
<td>5,000</td>
<td>750,000</td>
</tr>
<tr>
<td>Rumania</td>
<td>500</td>
<td>-</td>
</tr>
<tr>
<td>South Africa</td>
<td>200</td>
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<tr>
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<tr>
<td>Ukraine</td>
<td>14,500</td>
<td>600,000</td>
</tr>
<tr>
<td>UK</td>
<td>400</td>
<td>60,000</td>
</tr>
</tbody>
</table>

While a large fall or a fracturing of a pillar or a part of the coal face may give rise to large emissions of gas, together with displacement of coal, it must be distinguished from an outburst. A typical outburst is associated with a violent ejection of coal and emission of gas. The cavity formed from which the coal is ejected has a typical shape of a cone or follows the contours of a structure with which the outburst is associated.

A vast majority of outbursts occur in development roadways when driven in virgin ground in coal or when opening a seam from a cross-cut driven in rock. In mines which operate at depth, outbursts have occurred on longwall faces, particularly where these faces pass below remnants where high stresses result in decreases in permeability and crushing of the coal. Outbursts rarely occur in coal seams which have been destressed and degassed as a result of mining of neighbouring seams. The three most important parameters that characterise a coal seam liable to outbursts are high gas contents, low permeabilities of the coal seam and high rates of gas emission when the coal is crushed. Seams with the following characteristics can be classified as outburst prone:

- gas content (methane) > 8 m³/t,
- permeability < 2 mD, and
- mechanical strength < the lowest principal stress.

In mines with carbon dioxide as the main seam gas, lower gas contents than required for methane can give rise to outbursts.

The presence of a geological structure or discontinuity in the coal seam is very commonly associated with outbursts. Structures such as strike-slip faults, reverse faults and shear zones reduce the strength of coal within and in the vicinity of the structure and cause changes in the stress gradient and gas pressure gradients thus facilitating conditions that favour an outburst if sufficient gas is available. Gas itself does not necessarily initiate an outburst, though some investigations show that the presence of gas reduces strength (Josien et al, 1983; Lama, 1995) and causes high stresses to develop (Ettinger, 1977). It also results in body forces, which can initiate an outburst.
Added to the factors described above, advances in mining technology and the consequent high rate of roadway development and face advances in longwall mining, contribute to the occurrence of gas outbursts as a result of significant reduction in minimal time for the strata and the coal seam to reach a new state of equilibrium.

MECHANISM OF OUTBURSTS

Because of the wide variety of conditions under which outbursts occur, there is no single theory that can explain the phenomenon.

Some of the earliest concepts on the nature of gas emissions, gas pressure and properties of coal was presented by Halbaum (1989-1900) who outlined the basic theory of gas pressure to describe sudden emissions of gas and outbursts. Later researchers such as Telfer (1911-12), Rowan (1911-12), Ruff (1930) developed theories of outburst followed by Caufield (1931), Jarlier (1936), Belov (1931) and Pechuk (1933) who introduced the role of stress and mechanical energy in outburst theory. Since the early 1950’s the most extensive work in this area is reported by Russian investigators (Khodot, 1951, Ettinger, 1952, Kristianovich, 1953 a, 1953b) who have considered mechanisms of sorption/desorption of gas and stress, in the generation of outbursts. Skochinski (1954 a, 1954b) synthesised the concepts, his analysis being based upon experiences in the former USSR and results of investigations of a team of about 300 researchers during 1952-54. According to him, outbursts occur as a result of mutual interaction of numerous factors including the followings:

1. Rock pressure, which is associated with,
   • Development of cracking and crushing of coal at the edges of the excavation, its destressing and decrease in strength.
   • Changes in permeability of coal seams, re-distribution of gas pressure and emission of large gas volumes.
   • Transfer of rock pressure from the static phase into a dynamic phase as a result of destruction of the coal seam close to the face or due to loss of the resistance offered to the roof. This results in development of further cracking and crushing and as a consequence, creation of fresh surfaces which increases desorption. This leads to formation of gas transportation paths resulting in a drop in gas pressure and release of its potential energy. Releasing of the elastic energy of the coal together with the gravitational energy, which converts to the dynamic energy of coal in movement and increasing the intensity of sudden outbursts.

2. Gas present in coal:
   • Static and dynamic gas pressure in coal under normal gas pressure gradients cannot initiate an outburst unless the following three conditions are fulfilled:
     (a) Sufficiently high gas contents of coal.
     (b) Fast rate of crack development and disintegration of coal as a consequence of mining, with the formation of a large number of new surfaces that can ensure intensification of desorption and filtration of gas.
     (c) Formation of cracks of sufficient length and volume in fractured or crushed coal that will ensure distressing, flow of gas into the excavation and decrease in gas pressure between the excavation and the coal where gas is desorbing.
   • A sudden drop of gas pressure, of the order of 2 MPa or more over a distance of 1 mm alone is enough to crush the coal, throw it and ensure propagation of the crushing wave to a certain distance into the rock mass.
   • In an outburst, the gas present in coal particles disintegrates the particles into fine dust and intensifies the flow of gas and coal particles into the excavation.
   • The gas liberated from the fractured and crushed coal is sufficient to cause an outburst if the gas pressure is at least in the range of 0.3 - 0.6 MPa.

3. The physical and mechanical properties of coal and micro and macrostructure of the coal seam. The structure of coal defines the following:
   • the strength of coal and its resistance to stress,
   • the rate of emission of gas and work exerted by gas during its emission, and
   • the amount of gas in coal and total potential energy that may be available in an outburst.

4. Gravitational force becomes effective in steeply dipping seams and excavations after an outburst is initiated. The kinetic energy increases under the effect of gravity and approaches an amount equal to that of gas, in the case of outburst caverns formed in line with the rise of the seam (Nekrasovski, 1951).
According to Skochinski (1954a, 1954b), rock pressure is not the basic factor causing an outburst, rather it is gas that is responsible for its development and sustenance. Skochinski does not consider the role of tectonic forces in the process of an outburst because of the following:

- There are no methods that can define residual tectonic stresses.
- Very often two coal seams lying 20 - 30 m from each other differ in their susceptibility to outbursts in spite of the use of the same method of mining.
- Coal seams and the surrounding rocks are intersected by a number of micro and macro cracks.
- Rock pressure alone is insufficient to cause an outburst.

Since the work of Skochinski (1954a, 1954b) and his colleagues the concepts have been extended and mathematical theories have been developed. Khristianovich (1953a, 1953b) developed the crushing wave theory and has treated the outburst process as a complex function of natural tectonic and induced stress which causes initiation of an outburst; and free gas present in the pore space transports the broken materials. The crushing wave travels from the face into the solid, destroying successive layers of coal in the direction opposite to the direction of movement of the broken mass. These disturbances are destressing waves and receive their energy from the compressed gas in the pore space. According to Khristianovich, differential gas pressure, at the face of the crushing wave is, equal to or greater than the tensile strength of coal. This results in splitting the coal into small layers (discs). His theory is supported by a number of investigators who have reproduced outbursts in the laboratory (Yartsev, 1958, Ujihira et al., 1989, Ujihira and Nakajima, 1991, Gawor et al., 1991).

More recently, coal under high gas pressure has been treated as a retrograde material (Litwiniszyn, 1983) with gas as a solid solution in the coal matrix. The gas undergoes a phase change as a result of changes in thermodynamic conditions. The model presented by Lihvinisyn has been validated in the laboratory (Bodziony & Kraj, 1995).

A number of authors have associated outbursts and rockbursts as one single phenomenon with the difference that gas may be absent or that gas is a secondary factor for rockbursts (Josse, 1957, Budryk, 1951, 65, Coeuillet, 1959, Szirtes, 1966, Lama, 1995). Numerical models which use tensile strength as a criteria of failure, have been used to predict outbursts (Paterson, 1986, Barron & Kullmann, 1990; Chen et al., 1995).

In general, there is convergence of views that the following factors play a dominant role in outbursts:

- Geological structures particularly faults, contact zones of coal with volcanics and deformation of coal.
- Static and dynamic stresses in the neighbourhood of other excavations.
- Lower strength of the coal seam in relation to the stress levels.
- Gas pressure and gas content of the coal seam.
- Rate of gas desorption.
- Sudden exposure of the coal seam.
- Part of the excavation that forms steep faces.

### PREDICTION OF OUTBURSTS

The necessary conditions for an outburst to occur vary from mine to mine, but the four most important and widely accepted conditions are:

- gas content,
- geological structures,
- stress regime, and
- material properties.

All the four factors work together in producing an outburst. Geological structures determine the location of the outbursts. Stress plays a role in initiating an outburst. Gas content determines the amount of energy that is available for an outburst and transfer the material.

When predicting outbursts, the important thing is consideration should be given to the factor that plays a major role in a particular situation. Gas content is always an important factor. Without the presence of a certain critical gas content value, outbursts of gas, coal and rocks will not manifest themselves at all.
Geological conditions such as faults, dykes and shear zones, play a very dominant role in shallow mines and even in deeper mines where the size of the outburst increases where a geological structures are present. In the absence of a geological structure, outbursts depths down to 1,000 m depth are small.

Stress plays an important role in deep mines. The role of stress must be judged in association with the strength of the coal/rock. Stress levels that are sufficient to fracture rock to almost a state of pulverisation cause intense outbursts. While gas content can be measured or estimated fairly reliably and geological structures may be predicted or are evident at the places where outbursts have occurred, measurement of stress is not easy and is almost impossible on a regular basis under operating conditions in mines. The effects of stress however can be measured indirectly.

The methods that have been developed to predict outburst conditions can be divided into groups based upon various factors influencing the method (Figure 1). It is difficult to categorise each method precisely. For example, gas content which is a function of gas pressure influences the properties of coal and rock. Discing is a phenomenon which describes the status of stress but it is also influenced by gas pressure and rate of drilling. Radio imaging is based upon the dielectric resistance of coal, but its value is highly dependent upon the moisture levels and in mines with low moisture coal, its use in prediction of structure is dependent upon moisture level anomalies.

The type of method used depends upon local conditions. Some mines may use more than one method for continuous prediction. For local and regional prediction invariably more than one method is used.

CONTROL OF OUTBURSTS

Control of outbursts of gas and coal and rock is based upon two broad concepts. The first approach is to develop and use methods so that outbursts do not occur at all. The second approach, is to develop systems so that the miners and equipment can be protected from outburst effects. The nature of such methods or systems will depend upon the dominant factors that can be changed or controlled most easily. Accordingly, classification is based upon control factors such as gas or stress or locational factors. Two examples of the classification system are given in Figures 2 and 3 (Bodziony and Lama, 1996).

In stress control methods, short holes (40 to 80 m long) of large diameter (up to 300 mm) are drilled ahead of the face (Yu-Bufan, 1985; Anon, 1964). The number and length of holes drilled depends upon local conditions and the technology available. Large diameter drilling presents dangers of an outburst initiation during the drilling process and many times requires either remotely operated equipment (not yet available) or drilling behind specially constructed barriers. Destressing of a coal seam is possible by extracting a neighbouring seam above or below. This lowers the stress as well as gas content levels.

The important point to consider is complete extraction without leaving any barrier pillars. When this is not possible, camouflet blasting, hydraulic washing of the coal face or slitting of the roof/floor rock is adopted. Camouflet blasting was the first and most commonly used method in Europe the first half of the 1900s. Although this method reduces the danger of unexpected outbursts, the frequency increases. Hydraulic washing has been used mainly in Hungary when the seam is very weak and drill holes cannot be maintained even for camouflet blasting. Slitting has been tried in Russia and Ukraine successfully, but the method is highly labour intensive and there are no machines available that can produce deep slits efficiently. Blasting using relief holes has been suggested but has not been clearly demonstrated at this stage.

Gas drainage prior to mining is the most common method presently being used in Australia, Poland, China and Russia. Holes of up to 300 m have been drilled ahead of the face for degassing, to bring the gas levels down to safe threshold values. Technology exists for drilling holes within coal seams to depths of up to 1,000 m. Developments in seam gas drainage from the surface, using surface boreholes and hydraulic fracturing has been successful in the USA, but its applicability to drain gas economically from seams susceptible to outbursts is yet to be demonstrated. Directional drilling from the surface presents a distinct possibility (Oyler and Diamond, 1982). Underground hydraulic fracturing has been successful in Russia (Lidin, 1987).

Chemical treatment of coal seams by injecting water with 2% hydrochloric acid has been tried in Russia successfully to increase permeability of the coal seams where a high percentage of calcite is present (Airuni, 1981). Modern underground longwall technology requires high rates of advance of development headings. Longhole advance drainage seems to be the only technology at this stage that is capable of meeting the demand.
Remote machine mining is a distinct possibility for outburst control in the near future (Wynne and Case, 1995). Face cutting machines such as continuous miners have been successfully equipped for the protection of the drivers.

Prediction of structures which are the loci of the vast majority of outbursts is the key element in control of outbursts. A number of methods have been tried. Unfortunately, no method has been proven completely successful. Geophysical techniques such as micro-seismic and seismic wave analysis have been reported to be successful in the Donetsk basin of Ukraine, China, Japan and Australia (Hatherly et al, 1995; Styles, 1995; Zhang et al, 1987; Kolesov et al, 1995) and geophysical techniques such as radar imaging are being tried in Australia (Murray, 1995).

Sonic probes (Hatherly et al, 1995) have been developed and are being tested. In-seam seismic techniques are successful in predicting small faults and high moisture zones underground (Thomson et al, 1995). Filtration properties of coal are shown to help in locating zones prone to outbursts (Lama, 1983). Fracture density changes show that the shear zones may be detected up 100 m away (Shepherd, Rixon and Creasey, 1980). More effort is required to ensure that all structures can be delineated prior to their intersection.

MANAGEMENT SYSTEMS FOR CONTROL OF OUTBURSTS

Mining of seams liable to outbursts requires the development of special procedures to ensure that the risk to miners and equipment is eliminated or reduced. Most mines have laid out basic mining conditions which, when achieved, can make mining of outburst-prone seams safer. These conditions may be based upon defining critical threshold values for gas content or gas desorption rate. The methods to reach the threshold values depend upon local conditions. The purpose of the management systems is to ensure that the procedures are in place and are precisely followed and to ensure that under no circumstances can mining proceed if it endangers operations.

A management system thus relies on checks and balances to achieve the desired result. It also ensures that the system operates independent of the people who developed it. The desired results can be efficiently achieved by defining the methodologies and when the desired results are not achieved, the procedures need to be varied without sacrificing safety. Most countries around the world have developed guidelines for mining under outburst conditions. The basic outline of a management plan is given in Fig. 4.

The key features of a good management system are:

- Definition of the problem which clearly states what is the most important parameter (e.g. gas, structure, desorption index).
- Management plan that outlines the standard to be achieved pertaining to all technical parameters.
- A clear outline of the standard operating procedures.
- A decision making process that clearly allows the direction to be taken when the results of the operating procedures applied become available.
- Organisation, responsibility and authority of each person collecting information and those responsible for decision making.
- Flexibility so that in case of failure of the procedure, new methodologies can be introduced reasonably and quickly.
- Participation of all involved in the data collection, operation and decision making at various levels.
- Training of each member and updating the skills so that the "best practice" is always followed.
- Auditing of the system so that it conforms to the standard set in the plan.
- Information and control documentation is kept and available to all involved.
- Procedures available for corrective action if shortcomings are found in the management plan.
- Outburst mining procedures if normal mining is not possible.

Quite independent of the method used, the management system is the key to safe mining.

UNUSUAL EMISSIONS OF GAS

Very high emissions of gas are commonly noticed in mines particularly on longwall operations and in development headings. Penetration of old workings has been very common in the first half of 1900s. With better
knowledge of the old mine geometry records and development of advance boring, these events have been largely eliminated.

High gas emissions in development workings occur in seams of shallower depth, high permeability and in areas with open jointing. These emissions are basically due to intersection of high conductivity zones which allow very high flow into the mine workings. In mines with low in-situ gas contents (4 to 5 m$^3$/t), large gas emissions, with ignition at the face have been encountered. Sudden emissions also occur when highly gassy seams lying close to the seam under development, results in the fracture envelope penetrating the adjacent seam or bursting of the floor/roof under high gas pressures.

On the longwall face, sudden emissions of gas occur from the floor when strong sandstone beds separate the seam under extraction and the underlying seam. The adsorbed gas present in the lower seam accumulates below the strong beds in a free state. Strong beds break at intervals and high conductivity cracks allow flow of gas from the floor into the mine workings. The higher the delay in the fracturing of the inter-lying strata, the greater the volume of flow into the workings. Figure 5, which gives methane emissions into a longwall face as a function of time, shows the cyclic nature of the phenomenon as observed in the Bulli seam on the South Coast of New South Wales, Australia. In one example the gas is released from the lower seam lying about 10 m below the seam under extraction. A sandstone bed approximately 1 m to 1.5 m thick lies between the seams almost immediately above the lower seam. Gas emission rates up to 1.5 m$^3$/s have been recorded. The largest floor outburst delivered 330,000 m$^3$ of gas. These are more frequent in the first longwall and decrease when the neighbouring longwall has been mined. The phenomenon has been termed by some investigators as floor bursts.

Hinderfeld (1994) reports the occurrence of a similar phenomenon in the Ruhr coalfield. During the period 1969-84, 18 large floor bursts occurred in the Ruhr district with an average flow of 15.6 m$^3$/min. Large flow rates on the longwalls released between 4,240 to 200,000 m$^3$ of gas. In the roadways, it released 25,000 - 85,000 m$^3$ of gas and in the drill holes 2,000 - 16,350 m$^3$.

Studies have indicated that the factors which influence sudden large emissions are:

• Small tectonic zones of high gas content and high permeability,
• High gas content of coal seams / unit area,
• Strata sequence with thick strong beds, between the seam mined and surrounding seams,
• Coal face geometry where the extracted longwall panels form more or less a square shape.

Control of such events is possible only by pre-drainage of the source of gas emission and gas accumulation.

CONCLUSIONS

Outbursts of gas, coal and rock have increased in frequency as mines become deeper. To avoid the cost of running mines prone to outburst, older coal mining countries have closed their mines where the mining needs to go to deeper seams. In Australia however the gas drainage experiences over the last two decades have shown that the sudden emission of gas and outbursts can be controlled by pre-drainage of the gassy and outburst prone areas. Definition of gas content thresholds and draining seams to levels below the threshold have been effective in reducing the number of outbursts. The main initiative in outburst control is therefore pre-drainage of the coal seams and detection of geological structures.

Modern fast mining increases the risk of outbursts, management systems have been developed to allow safe and outburst free application of rapid mining technology. Remote operation of equipment is needed so that unexpected outbursts do not cause any physical damage to the operators by applying various safety measures in areas where there is a danger of outbursts and where safe threshold levels cannot be achieved can be mined safely.
Fig. 1 Classification of outburst prediction method (Bodziony and Lama, 1996)
Fig. 2  Classification of outburst control methods based upon most important factors (Bodziony and Lama, 1996)
Fig. 3 Classification of outburst control methods based upon locational factors (Bodzony and Lama, 1996)
Outburst management plan

General requirements

Plan elements

Processes

Procedural part

Technical standard part

Internal standards

External standards

Fig. 4 Basic outline of management plan
Fig. 5 - Occurrence of high gas emission due to floor bumps on longwall
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CASE STUDY
CENTRAL COLLIER QLD

Dieter Bruggemann1

ABSTRACT: A coal mine outburst occurred on the 20th of July 2001, in ‘B’ heading of 310 panel. It is the first such incident to have occurred at the mine in its 17 year history. Mining has since the initial days progressed to deeper levels where now the workings are at some 425m below the surface. The issue of greater gas contents with depth is recognised where inseam cross block drainage hole patterns are employed to drain and therefore reduce gas levels in the developing panels and in the longwall. The outburst incident was intimately related with a geological structure, which was covered by inseam drainage boreholes. Subsequent to the outburst further training of the workforce and a full review of the coal mine outburst management plan was undertaken. Enormous experience was gained by all involved in the investigation process.

INTRODUCTION

Central colliery is situated in Central Queensland approximately 250km north west of Rockhampton and is one of three mines comprising the German Creek Coal Mining operation plus one in the project phase. Central Colliery was the first mechanised longwall coal mine in Queensland with underground mining commencing in January 1984 and the first longwall coal produced in 1986. Fig. 1 shows the lease areas.

This paper aims to summarise the events of the 20th July 2001. It is also intended to describe the outcomes of the investigation process and what lessons have been learned as a result of the incident.

Fig. 1 CAPCOAL Leases

1 Capricorn Coal Management
GEOLOGY

The geology of the German Creek operation is based on the reserves of the German Creek Formation and the Rangal Coal Measures. The former contains economic coal in the Pleiades, Aquila, Tieri, Corvus and German Creek seams. In the latter only the Middlemount seam is the seam of principal economic significance.

Stratigraphy

Central Colliery is situated in the centre of the Bowen Basin and the operation is worked over a 12km-strike length. The 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 sedimentary strata is well jointed with the primary joint set trending northeast and a well defined secondary set trending southeast. In the mine area, sediments were deposited in a fluvio-deltaic environment. The massive sandstone units found in the area have been attributed to beach bar deposition. Coal seams worked range in thickness from 0.5 to 4.0m (Figure 2).

A rider seam, the German Creek Upper, of approximately 0.3m thickness, splits away from the main German Creek seam in the southern side of the main lease to a height of greater that 8m to the north. The immediate roof to the south of the seam split comprises a thin mudstone unit which gives way to interlaminated fine grained micaceous sandstone and siltstone averaging 50Mpa UCS. To the north of the split line the roof comprises a carbonaceous siltstone interburden, averaging 60Mpa UCS which is overlain to the north by laminated sandstone and siltstone. The immediate floor is comprised of a dark grey carbonaceous siltstone averaging 50Mpa UCS, although in places this floor is substantially weaker.

![Fig. 2 Bowen Basin North – South Stratigraphic Relationship](image)
IGNEOUS ACTIVITY AND STRUCTURE

Structures in the form of folds, faults, shears and jointing are present throughout the mining lease. Intrusions in the form of dykes and sills span the lease area. As can be seen in Figure 3, these nominally follow a northeast – southwest trend, and are common to distinct structural domains, which often gives rise to adverse roof conditions when mining. Current depth of mining is about 430m with the principal horizontal stress oriented in a north-northeast direction at approximately 24Mpa. The horizontal to vertical stress ratio is some of 2 to 2.5 to 1.

SEAM GAS

At Central Colliery, gas levels, which are at present in the order of 12m$^3$/t to 15m$^3$/t, of mainly methane gas increase with depth. Gas chromatography data indicates that CO$_2$ is a minor contributor to the total gas make.

A gas drainage system has been in place since the 306-longwall block where face parallel drainage holes were employed, having an average length of 260m. From longwall block 307 to the current 311 longwall blocks, a fan pattern of inseam drainage holes was used, to aid drainage efficiency and minimise the problem of relocating the rigs.

The spacing was progressively reduced from an average of 50m in LW306 to LW310, to a spacing of 40m for longwall block 311. This is in recognition of the fact that increased virgin gas contents are present at depth and thus the frequency of drainage holes would have to be increased in order to achieve a post-drainage gas content of less than 7.5m$^3$/t within the time period available for effective drainage. The spacing of each fan pattern of 5 holes has also been progressively reduced so that these occur at every second cut-through (Fig. 3). In addition, these boreholes have been oriented through the subsequent panel so that their intersection is at right angles to the development direction, which in turn has alleviated the problem of having gas leakage into the development workings during development. The German Creek seam at Central Colliery has a permeability of between 3 – 10mD.

Fig. 3 Central Colliery Workings and General Seam Gas Drainage Hole Patterns
THE OUTBURST

An outburst occurred on the 20th July 2001 in ‘B’ heading of 310-maingate development panel, where initial reports suggest that the amount of coal expelled was in the vicinity of 50 tonnes. Later after some of the broken coal was removed, it was estimate that 80 to 90 tonnes was displaced. No injuries resulted from the incident with the exception of a few minor bruises and scrapes.

Background

Events leading up to the outburst were that the 2-entry panel was to be extended to 28 cut-through, with normal mining proceeding at that particular stage. Inseam drilling was in place to determine the gas content levels. The area had been drained by the standard cross panel drainage holes drilled to a spacing of 40m. Drainage is normally quick and efficient. The 12 to 15m³/tonne are usually reduced to 3 to 4m³/tonne within 6 to 8 month’s time. A core taken from a vertical hole prior to mining, just outby of the outburst site had a gas content of 6.8m³/tonne. The next test core, some 140m inbye had a content of 7.8m³/tonne. The gas content threshold is 8m³/tonne.

The face was driven about a pillar length inbye of 27 cut-through. The outburst occurred from the right hand rib / face junction. The drainage holes on either side of the outburst were checked two shifts prior to the outburst, where the flow had considerably reduced but no blockages were indicated. All holes were on suction.

Just prior to the outburst, a number of events occurred. The first event was a loud “bang” which caused the continuous miner driver to put the miner into reverse high tram. The noise itself appeared to be “deep and appeared to emanate from the roof”. Then cracking in the rib was noticed and the rib was noted to be fretting as a result. The second event took place some seconds later, when another loud “bang”occurred which was described as being louder than the first. At that particular stage the miner had traveled backwards some 2 metres, and some small pieces of coal was being thrown towards the miner. A distinct pressure change had resulted in the ambient atmosphere, where personnel reported the ‘popping of ears’. The third event was described as a type of “suction towards the face” with the majority of the face coal having been thrown out by that stage.

As these events were occurring, the personnel in the adjacent heading also heard these three waves of noise, they also trammed backwards, thinking it was a roof fall about to happen or a coal mine outburst.

Most of the displaced coal was of blocky consistency. A structural change in the local geology had been noticed over the last 30m of the development. The normal cleat is first intersected by the left rib, but additionally, another three cleat orientations were noticed as shown in Fig. 4. Some of these additional cleat orientations have been present at times elsewhere in the mine, but at this location it is remarkable how consistently they occur in their attitude (strike and dip), persistence and spacing. In addition, the roof joint pattern appeared to alter, in terms of spacing from a normal one every three to six metres to approximately one every metre. In the outburst cavity itself these joints appear at a spacing of only 30cm semi-parallel to the identified outburst structure.

The main structure associated with the outburst is interpreted to be a strike slip fault, with a nominal dip slip component. The axis of the outburst was perpendicular to the structure, and it could be said that some nine metres of the seam outbye of this main feature was affected by the geological disturbance. Subsequent to the outburst drilling using Pro Ram holes ahead of the two faces bogged in the approximate locations of where the main structure is located. In one particular instance in one of these boreholes, a gas push, through the drill string was recorded. Subsequent testing of this area indicated a content of 11.68m³/tonne of mainly methane gas in the down dip side of ‘B’ heading.

Information from further investigations using a ‘Fault Tree Analysis” system post the outburst event, indicated that a greater knowledge of the gas regime in the area was required.
Fig. 4  310 Panel Outburst Cavity – Geological Features

Additional holes were drilled between the headings to establish gas contents and the location of any structures. The threshold for the area was temporarily reduced to 6m³/t, and if the core content from this area was greater than 5m³/t, then additional cores were to be taken at 25metre intervals. Fig. 5 shows the general arrangement of inseam borehole locations pre – outburst, whereas Fig. 6, is the inseam borehole and sampling arrangement post – outburst. Positive outcomes from the work were that the “geologically disturbed zone” was positively identified from three inseam intersections and seam gas contents dropped markedly away from this disturbance.
INTERPRETATION OF THE EVENTS

It is important to analyse the mechanism by which an outburst has occurred. It is suggested that the outburst mechanism for the 310-panel event can be divided into four stages.

1. The area ahead of the face behaves as a confined solid, whereby free water is present in the pore spaces and all gas is chemically adsorbed onto the coal maceral. The water that is present limits the free desorption of gas. At this stage the mining process in ‘B’ heading of 310 panel was too distant to affect this free desorption stage. That is, further than nine metres as indicated in the geological mapping of the outburst cavity with respect to where the continuous miner was positioned at the time of the event. Within the nine metres though, the geology appeared to be disturbed, with the inbye most three metres being particularly affected.

2. As the face advanced, confinement on the coal was reduced, loading becomes biaxial, tensile failure commenced in tandem with a reduction in pore fluid pressure around the tensile failure. At this stage gas desorption began with free gas accumulating at high pressure. The fluid pressure at any distance into the face is in equilibrium with the leakage / desorption rate. At the particular stage, when the loud roof noises were heard, failure of the roof inbye of the face area can be interpreted. This may have been due to some horizontal de-stressing, resulting in depressurisation of the area ahead of the miner position, and therefore somewhere in the outburst cavity there would have been a volume change. As this occurred, the fluid pressure lowered to below the gas desorption pressure of the coal during mining.

3. As the face advanced close to the high-pressure free gas accumulation, being essentially the highly fractured and somewhat mylonitised coal, mass movement may have taken place. The reservoir of expanding gas provided the stored energy required to propel the fractured coal into the opening. This can be related to a massive change in confinement which promotes a sudden large increase in gas desorption rate, to the extent that the gas desorption rate is fast enough to maintain a high pressure in the fracture network, thereby causing coal mass failure. This can be considered the final event during the 310-panel outburst. The severity of the expulsion of the coal particles depend on the steepness of the gas pressure gradient, the free storage capacity...
of the coal prior to the event, the desorption rate of gas from coal and the depth of the tensile failure into the face.

4. The gas emitted during the event and the outburst cavity size are dependant on the total surface area exposed and the permeability of the coal mass. Emissions continue until the gas pressure gradient migrates into the solid coal and equilibrium is attained with the permeability. This may take some time particularly with geologically disturbed areas, as the permeabilities are lower and the gas flow paths are essentially anisotropic.

These steps are fairly typical of other coal mine outbursts which have occurred. In an overall sense, the amount of gas liberated as recorded by the American Mining Research (AMR) is in the normal range, and comparatively, the amount of coal dislocated is relatively small when compared to the known previous occurrences.

**LEARNING OUTCOMES**

A number of initiatives were instigated after the outburst event. These include updated outburst awareness training to all the crews. The training package involved the explanation of what a “coal mine outburst” is. An attempt is made to heighten awareness of such phenomenon and explain that numerous such outbursts have occurred in the past, with common dominators with respect to their parameters and geological setting. An explanation of the association of structure, stress magnitude and direction and finally coal strength is explained. Finally the physical signs as a recognition tool, as employed in other coal mining districts in Australia, are given, namely spitting of the face, face bumps, calcite stringers, abnormal orientations of coal cleat or jointing to just name a few. In this way the crews are able to “see” potential problems and them accordingly for the Geologist to inspect, very much in the same vein as roof support issues.

Inseam drilling crews have also been further updated in the detection of outburst prone structures. Some of the items to be recognised include the recognition of the inability to penetrate a known area, gas surging, bogginess, rods binding up and red-brown colouration of drill returns. As inseam drainage is the main weapon in combating gas levels, which may be too high for safe mining progress, this is of ultimate importance for the safe and efficient progress of the development cycle.

Inseam drilling does provide an initial identification of structural disturbances in training and potential outburst prone structures. It can give confirmation of the existence of such structures as are identified from remote detection techniques such as surface seismic and geological projections from adjacent panels. In addition the surveying of such boreholes has dramatically increased the confidence in the location of these structures, when in comparison, boreholes drilled using programs have little or no confidence assigned to them, as their drilling direction cannot be controlled. The in-seam gas test-sampling program is made very difficult, because of tight operational constraints.

The inseam borehole lengths were increased to run well past the target maingate, in an effort to minimise the drainage end effects. These boreholes are extended nominally 20 metres past the virgin side rib. In addition, as has been discussed above, the distance between these boreholes has been tightened to 40 metres, from a spacing of 50 metres in the past.

The core sampling strategy now involves the taking of compliance cores from “flank holes” Fig. 7. The logic revolves around the recognition of the regional to local geological environment in which Central Colliery operates. It can be said that a major proportion of all known structures including vertical igneous intrusions occurring in the Central Colliery mining area strike in a northeast – southwest orientation. The exception to date has been the intersected structure in ‘B’ heading of 310 panel, where the outburst had occurred. Its orientation is in an east-west orientation, thus using flank holes, it is expected that all other orientation of geological structures that may occur will be intersected. In addition, the sampling strategy has now a new focus, with the samples being taken in the worst possible location, between existing cross-block drainage holes and on the near virgin side of the adjacent longwall block.
The sampling strategy also involves the taking of such sample within 15 metres of the solid rib off ‘B’ heading in areas of known outburst-prone structures. A barrier distance of 10 metres is in existence for areas of no known geological disturbances.

Finally the gas content threshold has been revised, based on a more conservative desorption rate index of 900 (Williams, 2001) to a maximum gas threshold of 7.5 m³/tonne based on 100% methane gas.

Fig. 7 Inseam Drainage Flanking Hole Design

Photo 1 Outburst Area
CONCLUSIONS

As can be seen from the forgoing, a great deal of work and learning has occurred as a result of the coal mine outburst at Central Colliery. Fortunately no one was seriously injured and the mine has benefited from the experience and lessons learned. As the mine gets deeper, new challenges will be faced. An understanding of all seam gas aspects of the local geological environment is essential in planning a safe approach to new mining areas. This is achievable if a rigorous and systematic approach to risk is adopted, and mine operators maintain awareness of emerging technologies and an increasing understanding of outburst mechanisms.

REFERENCES

CASE STUDY

MANAGEMENT OF OUTBURST RISK AT TAHMOOR COLLIERY

Peter Wynne

INTRODUCTION

Tahmoor Colliery mines the Bulli Seam at a depth of 400-440m and has had to manage the risk of outbursts since it commenced production in the early 1980’s. Of necessity, it has been at the forefront of the development of systems to reduce outburst risks and has made many significant contributions to the industry in this regard. In the 1980’s, the general strategy for addressing these risks was that outbursts were “inevitable” and so most effort went into protection against their consequences. By the 1990’s, this approach was no longer acceptable and there was a major redirection of effort towards the prevention of outbursts. This effort has been highly successful and there has not been an outburst at Tahmoor since 1992, when predrainage of development commenced. Prior to 1992, outbursts were occurring at an average rate of about 10 per year.

SUMMARY OF TAHMOOR OUTBURST HISTORY

Critical events and stages of outburst management at Tahmoor include:

- 1981 – first recorded outburst
- 1985 – continuous miner driver killed by outburst whilst cutting dyke
- 1985 – encapsulated continuous miner introduced for cutting outburst structures
- 1982 to 1992 – averaging 10 outbursts per year crossing structures
- 1992 – introduced ABM20 continuous miner
- 1992 – commenced predrainage of coal around structures
- 1992 – draft Outburst Management Plan
- 1992 – remote mining through fault, last prepared a recorded outburst
- 1992 to 1997 – ongoing refinement of drilling techniques
- 1994 – Outburst Management Plan formalised
- 1999 to 2001 – grunching through “tight” coal zones

TAHMOOR OUTBURST ENVIRONMENT

The Bulli Seam mined at Tahmoor is between 1.8 and 2.3m thick. The incidence of geological anomalies such as faults and dykes is probably about average for current Bulli Seam mines. The virgin gas content is typically 12 to 13m³/tonne. The relatively unique feature of Tahmoor is the high proportion of carbon dioxide in the Bulli Seam gas. For all the coal mined to date, the Bulli Seam has probably averaged about 65% CO₂ and 35% CH₄. In the last two years, this ratio has typically been 90% CO₂ to 10% CH₄. For the Tahmoor North area some four kilometres to the north of current mine workings, where longwall extraction will commence in 2004, the ratio gradually changes to become methane-rich, at 30% CO₂ to 70% CH₄. This high proportion of CO₂ makes the gas drainage task harder for several reasons. Firstly, the gas threshold to which to drain is around 6.5m³/tonne, whereas if the gas was 100% CH₄ it would be 9.5m³/tonne. Secondly, CO₂ is inherently harder to drain because the coal has a greater affinity for CO₂ than for CH₄. Consequently, it is necessary at Tahmoor for the drainage holes to be more closely spaced and left on suction for a significantly longer time, compared to mines where the seam gas is predominantly CH₄.

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1 Austral Coal
Several common factors have been associated with all outbursts that have occurred at Tahmoor:

- They were invariably associated with the actual cutting of coal. None occurred when the face was being bolted or on non-production activities.
- They were invariably associated with either a fault or a dyk. It is not known for them to occur because of variations in the coal texture, eg mylonite zones, as occurred at some other Bulli Seam mines.
- High gas content.

In the last three years, some zones of coal which are abnormally difficult to drain have been encountered in the Southwest portion of the lease. This change in drainage characteristics does not appear to be related to any increase in outburst proneness, but results in the need for much-increased drainage effort and/or special mining techniques. This “tight” coal is the subject of ongoing research.

**SIGNIFICANT TAHMOOR CONTRIBUTIONS**

Because Tahmoor had a significant outburst problem from the early 1980’s and they were few systems available to the industry at this time, systems had to be developed locally to enable mining to continue with reasonable levels of risk. Of necessity, Tahmoor has been responsible for many innovations including:

- Design and development of the encapsulated continuous miner concept.
- Introduction of Ripu Lama’s gas content thresholds in conjunction with Westcliff.
- Introduction of the first outburst management plans in conjunction with Westcliff.
- The first remote mining of outburst structures and the first televised outburst.
- First large-scale use of grunching, as an outburst mining technique.

**OUTBURST RISK MANAGEMENT IN THE 1980’S**

The first outburst occurred not long after mining of the seam commenced. It soon became apparent that outbursts were to be expected whenever a fault or dyke was encountered. They could be anywhere from a bucketful to many tonnes of coal and came to be accepted as just part of working at Tahmoor. All this changed in 1985 when a fatality occurred whilst a dyke was being cut by a continuous miner. This involved over 200 tonnes of ejected coal and was violent enough to move the 45-tonne continuous miner back more than a metre. This event caused a rapid rethink about the outburst risk. The encapsulated continuous miner was quickly developed as a tool to enable structures to be safely mined. The key features of this system were:

- It was limited to only the miner driver at the face when cutting the structure.
- Radio communications to the miner driver.
- Independent filtered compressed air supply to the miner cabin and backup air supply in bottles.
- Physical protection of the cabin with Lexan screen and steel door.
- A standby squad of rescue personnel at the fresh air base, wearing breathing apparatus, ready to assist anyone at the face, should an outburst occur.

The encapsulated miners were used to traverse scores of faults and dykes and withstood many actual outbursts, without injury to personnel. The concept was soon copied by other Bulli Seam mines and “bomb squad” conditions became the standard way of handling perceived zones of high outburst risk. Whilst proving to be a reasonably safe technique for known zones, there was still no defence against a previously unidentified structure encountered without warning during coal cutting. This risk to be later highlighted by the multiple fatality at South Bulli, in 1992.

At the time, the encapsulated continuous miner was a significant advancement. The underlying philosophy was that outbursts were not preventable and so the best risk management strategy was to provide protection against their consequences. This was made somewhat more palatable to the crews involved by the payment of outburst allowances. Whilst these were often dressed up as allowances for extra skills, they were in fact “danger money” and rewarded people for taking risks. Meanwhile, outbursts continued to occur at the rate of about 10 per year at Tahmoor and the frequency at other mines was increasing, as they steadily progressed westwards into deeper coal with higher gas content.
OUTBURST RISK MANAGEMENT IN THE 1990’S

By the early 1990’s, risk assessment techniques from outside industries started to be applied to coal mining and Tahmoor conducted a risk assessment for the introduction of a new type of continuous miner. The ABM20 was radically different in the fact that it was actually continuous, coal cutting and roof support took place simultaneously. There were to be four operators installing bolts from a platform located about 1m back from the cutting head. It soon became evident these operators would be exposed to an unacceptable risk of an outburst resulting from an unexpected occurrence of a structure in the coal face. Systems had to be introduced to address this risk. It would not be feasible to protect these operators from the consequences of an outburst. The strategy had to change immediately to preventing the outburst occurring. We were fortunately able to draw on the work of Dr Ripu Lama, who had put massive effort into reviewing the worldwide experience with outbursts and their control. Probably the most significant single conclusion from Dr Lama’s work was that if you remove sufficient of the gas from the coal, an outburst cannot occur.

Trials were conducted as soon as the necessary drilling equipment could be acquired. Gas drainage was initially done around known faults and dykes, as these were perceived to constitute the biggest risk. This original drilling and drainage was rather hit-and-miss, as there was a lack of ability to accurately steer the hole. Even so, it appeared effective from the very early stages, as structures which had previously outburst on nearly every encounter were now being mined without outbursts occurring.

Combining Dr Lama’s work with local experience, Tahmoor developed the first draft of our Outburst Management Plan. To have an outburst at Tahmoor two things were necessary, high gas, plus the existence of a fault or dyke. Because both these were necessary for an outburst, controlling either should be sufficient to prevent an outburst. A “belt and braces” approach was adopted to address both, to allow a margin for error. The prime objectives of the Plan should thus be – reduce the gas content and find the structures (so that they can be avoided or mined with special techniques). Concurrently with Westcliff, we drafted and implemented Outburst Management Plans. These draft plans soon became the template for the mandatory Plans required by the DMR in the Bulli Seam. The Tahmoor Outburst Plan remains essentially in the same format today, with minor amendments and refinements.

Throughout the 1990’s there was continual improvement in drilling techniques for predrainage. Initially, any gas drained was a plus, but by the mid-1990’s, predrainage of all development was a necessity. The most critical factor in this improvement was the capability to survey and steer the hole. Drilling was originally rotary, with single-shot survey. As more and more drilling was undertaken, the limitation of these techniques became evident. This was because single-shot survey severely reduced drilling availability as holes became longer and, because it was not real-time, it could not be used to actively steer the hole. It only recorded where the hole had been. There was a progressive transition to down-the-hole motors with electronic real-time survey tools. These have enabled faster, more accurate and more consistent drilling. Given the right circumstances, roadways can now be predrained up to 800m away from the drill site. The progressive improvement in drilling techniques is well illustrated by Fig.1. It can be seen that in 1994 (right hand side of diagram), some areas were overdrilled, whilst others were underdrilled and thus drainage was less than optimum. By 2001 (left hand side), the patterns are very evenly spaced, giving almost perfect coverage of the areas to be drained. The accuracy achieved in the drilling is impressive as drillholes are typically encountered during mining to be within 2m of where the survey indicates. Annual drainage hole drilling typically exceeds 60km and requires some 12 employees.

Another interesting development during the 1990’s was the development of remote mining. With the introduction of the radio controlled ABM20’s, it was suggested that they could be automated, to enable a be mined with nobody at the face during actual cutting. Tahmoor staff developed a system to drive the ABM20 remotely and it was used to traverse a 3m fault in the Eastern part of the mine. The system involved a lot of complex gear including video cameras and signal systems, that had never been applied to underground coal. Successful mining was carried out part of the way through the fault, but then, as anticipated, an outburst of about 60 tonnes occurred, damaging the ABM20. This outburst was viewed remotely by the operators, but could unfortunately not be recorded. We believe this to be the world’s first televised outburst. This project was successful in proving that remote mining of outburst prone zones was possible. In 1993, an ACARP grant was received to fund the further enhancement of remote mining. The major addition was the “separately-ventilated” control room. This was a caravan-like trailer from which the face equipment was controlled. By drawing fresh air from well outbye, the need to use special flameproof equipment was avoided and also the operators were relieved of the need to continuously wear breathing apparatus. This system was successfully applied to the crossing of a dyke in the Longwall 14 gateroads.
Fig. 1  Drilling Techniques
THE TAHMOOR OUTBURST MANAGEMENT PLAN

The essential elements of the Plan are:

1. Drill and predrain the coal which typically requires holes at about 20m spacing on suction for 3 to 6 months. The location of every hole must be surveyed, so that there are no patches of coal left undrained.
2. Monitor borehole flow. If these are less than expected additional measures may be required, eg redrill at closer spacing.
3. Drill “scout” hole as a final check for structures, then take and analyse core sample.
4. If samples pass and scout holes are accurate and clear, authorise each pillar length’s advance for each panel. This involves at least three individuals, to minimise the chance of error.
5. Quality system issues such as audits, training, reviews and document control are included.

The requirements of the Outburst Management Plan are onerous. I would estimate that it requires several hours involvement per week for the manager, development superintendent, mine geologist and the drilling superintendent. Of all the systems in place at Tahmoor, it is probably adhered to most rigorously, as it is literally a life-and-death matter. A typical Authorisation plan is shown in Fig 2. On the back of this, and as part of the Authorisation, is the drilling and drainage data, shown as Fig 3. This diagram shows where samples were taken, location of “scout” holes and where intersections of drillholes should be expected. This gives some idea of the background work necessary for each authorisation.

The last outburst was in 1992. There have been no outbursts since the initial Outburst Management Plan was introduced. As there has been no overall change in the outburst potential of the coal mined, this success can only be attributed to the effectiveness of the Plan. Had we not had the Plan, with its associated drainage, outbursts would still almost certainly be occurring at the rate of 10 per year.

Figure 2 – Typical Outburst Authorisation
GRUNCHING

By the late 1990’s the Outburst Risk problem appeared to be largely resolved. Drilling patterns were close to perfect with very evenly spaced holes and no undrilled patches. Drilling was getting progressively further ahead, giving increased drainage times and enabling uninterrupted development. However Mother Nature had a surprise in store to prevent things getting too easy. Patches of coal that would just not give up gas, despite being well drilled were encountered. Initially, in the Longwall 18 gateroads, these patches were relatively small and infrequent. At first, it was not possible to overcome this “tightness” by closer hole spacing and/or longer drainage time. However, in the Longwall 19 gateroads, this “tight” coal became more widespread and would not drain even after nine months at a 6m hole spacing. A method had to be devised to enable safe mining through coal that was above the threshold otherwise, longwall extraction would soon be halted. The use of high-pressure water to cut “slots” around the drillholes was trialed, as a research project. It was considered these could lead to stress relaxation in the proximity of the hole, opening up the coal and thus enabling increased gas flow. Whilst slots were successfully cut to about 1m from holes, no increased gas flow was stimulated.

The D.M.R. Section 63 of 1994 decreed that development through “above-threshold” coal could only occur by means of remote mining. Tahmoor had previously used remotely controlled mining equipment and it was certainly technically possible. However, it was not considered practical to utilise this extensively, as it was just too complicated to keep operating on an ongoing basis. Grunching (excavation of coal with explosives) was proposed as an alternative. It was considered as remote because the shot is initiated from a remote mining location and so nobody is at the face when the coal is excavated. Grunching had the advantage that the gear is simple and the change from normal mining can be effected quickly. It was also expected to give more than the 1 to 2m advance per shift achieved with the previously used remote mining system. Its principal disadvantage was that it was labour intensive and hence would have a low rate of advance, compared to normal mining. Grunching was previously only allowed with P5 explosives. However, these were not available at the time and it was decided to use P1 explosives. This generated considerable discussion, but for once our high CO₂ content worked in our favour. The CO₂:CH₄ ratio exceeded 4:1, and so it would be impossible to ignite the seam gas. On this basis, the DMR did not stop the use of P1 explosives.

Grunching proceeded and did enable Longwall 19 to be developed on time. About 1600m of grunching, about 500 shots was done in all. This was achieved without incident. Grunching proved to be an effective, reliable but slow (3m/shift) means to traverse “tight” coal and will be utilised again in the future if necessary. Relocation to a new
longwall domain, which is draining normally is currently occurring and hence widespread grunching in the near future is unlikely.

CONCLUSIONS

The outburst risk is now minimal, but this requires constant vigilance and considerable effort to maintain standards. As was experienced with the Longwall 19 “tight” coal, subtle variations in the geology can require refinement of techniques and there can never be surety that outburst problem is solved. Issues that will be pursued in the near future include:

“Tight” coal
- What are the characteristics that enable it to be identified well in advance of mining? (This is one aspect of a current research project).
- Once its characteristics are understood, what techniques can be applied to drain it with minimal impact on mining?

Threshold Limits
- Can a case be established for incremental increases in the threshold limit, to gradually decrease the drainage burden?
- Should the limit be higher for CO₂ than for CH₄? Whilst the consequences of a CO₂ outburst are obviously more severe, is an outburst more likely in a CO₂-rich seam than in a CH₄-rich seam? Some research suggests not! This needs to be investigated.
CASE STUDY
OUTBURST & GAS MANAGEMENT

Phil Eade

INTRODUCTION

Illawarra Coal is part of Carbon Steel Materials that is in turn part of the BHP Billiton organization.

Illawarra Coal operates five underground coal mines, viz Appin, Tower, West Cliff, Cordeaux and Elouera. One of these mines is Cordeaux Mine which ceased production in April 2001 and is currently being operated on a care and maintenance basis until its ultimate future is decided. Approval for a new mine, Dendrobium, has been granted and construction has commenced with the aim of replacing dwindling Elouera Mine production and reserves by 2004.

Appin, Tower and West Cliff mine from the Bulli or No. 1 Seam of the Illawarra Coal Measures. This seam is considered very gassy by world standards and each mine has extensive underground gas drainage systems in place to control both gas emissions and the outburst hazard. Seam gas present is predominantly methane but some areas of the mines encounter high carbon dioxide levels, generally localised around geological structures within the seam.

BACKGROUND

A brief outline of the major features associated with gas management and outbursts is given below.

Appin Mine

Appin Mine was established in 1962 and is the oldest of the Division's operating mines. It is situated immediately south of the township of Appin, 37 kilometres from Wollongong. The Bulli Seam at depths of up to 550 metres, provides prime coking quality coal used mainly for making steel at Port Kembla, Newcastle and Whyalla.

Longwall mining was introduced at Appin in 1969. The bord and pillar extraction method used previously had not allowed the mine to achieve production levels required to keep the mine viable. Strata control and stress levels at the mining depths at Appin were the main issues faced in pillar extraction by continuous miner.

Extraction by longwall mining presented a new set of risks to be controlled. Due to the high seam gas content and the adjacent coal seams below, gas emissions around the longwall made managing gas levels, particularly methane, in the ventilation circuit and face area a difficult task. In addition, gas emissions, particularly from the virgin side of gate road development panels made ventilation of the mining developments difficult due to intake pollution and high return gas levels.

As a response to the gas emission problem a methane drainage system was introduced at Appin in 1981 including a surface suction plant and pipe reticulation of the gas to the surface.

Tower Mine

Tower Colliery is situated 40 kilometres north west of Wollongong, between the rural townships of Wilton and Douglas Park. Mining started there in the Bulli Seam in November 1978. Its boundaries enclose significant reserves of prime quality, low ash coking coal, which is used mainly to make steel.

1 BHP Billiton Illawarra Coal
In early 1979, a large fault zone was encountered around the pit bottom area. As a result, contractors were engaged to drive through the stone and this halted coal production from June to October 1979.

Late in 1982, mining operations were wound down, mainly due to the economic recession at the time. A small workforce kept the mine operating at reduced output.

Mining operations remained at this reduced level until the mid 1980's, when it became clear that longwall mining was necessary for the mine’s viability.

Tower, with its large reserves of prime coking coal, was selected to become a high producing longwall operation. Tower's operations were increased and finally, in March 1988, longwall mining began.

Tower Colliery is a gassy mine with high emissions. Effective methane drainage is essential for its continued safety. The methane drainage plant contains eight vacuum pumps linked to a network of underground pipes and drainage holes. Both longwall and development panel mining have benefited from the extensive methane drainage in place at Tower.

**West Cliff Mine**

West Cliff Mine commenced coal production in October 1976 following development by Coal Cliff Collieries Limited, owned by CRA Limited. The mine and associated infrastructure was purchased by BHP Billiton in March 1997 due to its fit with our other Bulli Seam operations. The seam is approximately 480 metres deep and workings are adjacent to the Appin holding.

As with the other two mines, soon after production commenced it became apparent that high gas levels in the mine ventilation system were detrimental to mine safety and productivity. Gas drainage studies began in 1978 and culminated in the successful commissioning of a gas drainage system, including surface exhausters, in March 1980. The mine experienced outbursts soon after production commenced with gas drainage soon becoming imperative to control the phenomenon. Longwall mining commenced around 1982 and current annual production is budgeted at over 2.3 million tonnes of run of mine coal.

**Utilisation History**

Methane drainage as practiced at Appin and Tower invariably results in the production of fuel gas with variable characteristics. Principally, this variation relates to composition and flow.

Being mindful of methane’s potential as an energy or chemical resource, BHP examined possible uses for the gas. Having due regard to its location and composition characteristics, it was concluded that the most viable economic use for the gas was for conversion to electrical energy for sale to the local electrical distribution authority.

In April 1986, BHP commissioned a nominal 14MW gas turbine alternator unit and associated gas compression, flame arrestor and gas filter ancillary plant at Appin Colliery. This unit consumed a portion of the captured gas available from the mine to produce electricity for sale to Prospect Electricity, now Integral Energy. Unfortunately this plant experienced a number of major failures and was permanently decommissioned. The gas turbine unit used distillate as a standby fuel for start up and to avoid shutdown during periods of transient loss of mine’s gas fuel. Replacement of this unit was commissioned in 1995 in the form of gas engine based power plants of 94 MW nominal output distributed between Appin and Tower mine sites.

West Cliff followed a very similar path to that indicated above in relation to gas utilisation. Early in 1986 they commissioned a 12.5 MW nominal output gas turbine complete with a 4500 cubic metre capacity floating bell gas holder.

**Outburst Experience**

Appin, Tower and West Cliff Mines all mine coal from the Bulli Seam that has over the years exhibited a proneness to outburst. This experience has been well documented with numerous outburst events being recorded at these three and other Bulli Seam mines.
CURRENT PRACTICE

General

There exists a strong link between the management and control of gas emissions in the underground workings and the management and control of outbursts at the three mines. Gas drainage is the prime tool used in reducing pollution of intake ventilation flows by seam gas emissions and also to reduce the gas content of the coal to below predetermined threshold limits. These limits have been found, to date, to be largely successful in reducing the risk of an outburst event occurring.

The link appears most obvious with respect to in seam gas drainage where any gas that is drained from the coal prior to mining will reduce the gas content of that coal. Pre drainage of the coal prior to mining is a function of the location and density of the boreholes, the lead time the holes are draining prior to mining and the permeability of the coal to gas flow. The presence of geological structures and anomalies can also effect the homogenous draining of a block of coal.

Post drainage of gas is achieved by drilling boreholes at an angle below the operating longwall blocks to intersect the coal seams below and to collect a portion of the gas emanating from these coal seams as it migrates toward the longwall area. Goaf formation and resulting strata relaxation and fracturing makes the gas contained within the coal seams below the Bulli Seam a significant source of gas around the operating longwall area.

To complement the gas drainage in the control of the outburst hazard, each mine has a comprehensive outburst management plan aimed specifically at preventing outburst occurrences. Gas content and composition analysis of coal sampled ahead of mining is an integral feature of the outburst management plans. The detail and philosophy of these management plans will be further discussed later.

Gas drainage holes are often drilled well ahead of mining as an exploration tool to determine existence or otherwise of structure and anomalies. These holes are often over 1000 metres in length and they double as an early start to gas drainage as well as a valuable mine planning tool.

Appin Mine

Appin Mine produces around three million tonnes of run of mine coal per annum, with the majority from longwall retreating faces with widths in excess of 250 metres. Two heading gate roads are driven on both the main gate and tailgate side of the longwall panel.

Inseam Drilling

Appin utilise two Kempe K200 series hydraulic drill rigs with N size drill heads. For the year ending June 2001 over 45,000 metres of in seam drilling was completed and approximately 38,000 metres are scheduled for the current year.

The majority of in seam drilling is done from stubs driven in to the future longwall block from the virgin side gate road. The holes are drilled in a fan pattern with holes ranging from 320 metres to 450 metres in length. The objective of these holes is to drain both the longwall block and the next gate road development panel. Where relatively high permeability is thought to exist the hole extremities are designed to be 30 to 45 metres apart. Conversely, in areas where structures are known to exist, closer drilling patterns with hole spacings down to 10 metres are often employed due to lower permeabilities.

All in seam holes are drilled through 6 metre long, 100 millimetre diameter copper standpipes which are fully grouted into the coal. Hole diameters are nominally 92 millimetres using a PCD bit manufactured by Boart Longyear. An Acudrill Down Hole Motor (DHM) is used and when drilling, the drill string is generally rotated along with the DHM to give increased penetration rates. Rotating the drill rods in this manner results in a slightly larger than 92 millimetre hole. Drill rods are 3.0 metre NRQHP which are similar to CHD apart from the thread. They are manufactured by Boart Longyear.

Boreholes are surveyed using a MECCA survey tool manufactured by AMT with the tool down the hole and communicating to the hole collar via the drill string and Mecca system. Information received from the survey tool is logged and includes survey tool azimuth, distance down the hole whether left or right of the standpipe, whether above or below the standpipe and the direction the drill bit is facing.
Drillers log each hole in a book every six metres. Information logged includes visual colour and nature of cuttings and return water, whether drilling conditions hard or soft, gas surges while drilling and any other unusual events.

Inseam pre-drainage typically reduces seam gas contents from 14 cubic metres per tonne to between 2.5 and 4.5 cubic metres per tonne provided holes are drilled with a lead time of between five and six months. This assumes homogenous permeability and areas clear of structure. Where this is not the case additional or earlier drilling is required.

**Cross Measure Drilling**

Two Kempe 200 series hydraulic drill rigs with B size drill head are used to drill cross measure gas drainage holes beneath the longwall blocks. For the year ending June 2001, over 32,000 metres of hole were drilled with 35,000 metres scheduled for the current year.

The cross measure holes are drilled from the roadway adjacent to the operating longwall which is 45 metres from the maingate and are rotary drilled. Where pillar lengths are 150 metres, drill sites are established at 50 metre intervals. Each site comprises five boreholes, three of which are drilled at minus 20 degrees and two at minus 25 degrees. Four of the holes are angled back toward the approaching longwall while the fifth is drilled normal to the gate road direction. These holes are designed to intersect the Balgownie and Wongawilli Seams below the Bulli Seam being mined.

The holes are drilled using 1.5 metre long BW drill rods and 65 millimetre diameter PCD drill bits, both supplied by Boart Longyear. Length of hole varies between 90 and 120 metres depending upon declination, inter seam burden and general dip and target the base of the Wongawilli Seam. All holes are drilled through 3.0 metre standpipes with nominal bore of 80 millimetres which are fully grouted into the floor.

All holes are cased with slotted casing from just above the Balgownie Seam intersection to the bottom of the holes. Although the top section of the hole is not cased, it does not appear to have affected hole drainage performance. Drillers log each hole as they extend rods, recording stone and coal intersections.

Weekly gas balances around the longwall block indicate a gas capture efficiency of approximately 45 to 55 percent. There would indicate that approximately equal volumes of gas around the longwall panel report to the ventilation current and the gas drainage system.

**Goaf Drainage**

Currently Appin drain very little gas from old goaf areas. Approximately 150 litres per second of methane is being drawn from a worked out area of the mine. This has in the past been up to 350 litres per second but difficulties in effectively sealing the areas to stop dilution with air has reduced this figure.

**Outburst Management**

Core sampling for both gas composition and content is carried out in line with the Appin Mine Outburst Management Plan. They are taken at 150 metre intervals and more often where structures are suspected. All cores are sent to our Technical Services Laboratory for analysis and reporting.

Philosophy and detail of the Outburst Management Plans will be described later.

**Tower Mine**

Tower Mine is budgeted to produce between 1.3 and 1.4 million tonnes of run of mine coal in the current financial year. The presence of geological structures and in some areas difficult gas drainage and drilling conditions have severely hampered the mine’s production capacity.

**In Seam Drilling**

The mine employs 2 Kempe 37 Kw drill rigs utilising 3 metre NQ Longyear Mecca rods, downhole motors and 96.1mm diameter Longyear and Wadam PCD bits to drill approximately 23,500 metres of in seam branched holes per year. Due to poor gas drainage conditions it is likely that this quantity of drilling will be exceeded in the current year.

Holes are drilled through a 100 mm diameter, or sometimes 150 mm, 6 metre copper standpipe that is fully grouted in to the coal. All holes are surveyed with Mecca or Acoustic survey tools and are logged by the drillers for survey and any anomalies encountered whilst drilling. These records are kept on file for future reference.
Typically a fan pattern is used across future longwall blocks with 20 metre spacings between adjacent holes and
lengths up to 400 metres. With short lead times and difficult drainage conditions, hole spacings have at times been
reduced to as low as 8 metres. Significant issues have related to the inability to drill effective drainage holes in
one particular area due to holes collapsing. Remote mining on development has been necessary at times where it
has not been possible to reduce gas contents of the coal below the outburst threshold limits. Some success has
been experienced in these structured areas with drilling into the roof over the anomaly where hole stability is
improved in order to access coal ahead of the face for drainage.

Cross Measure Drilling
A Longyear LMC 55 drill rig is used to drill cross measure boreholes to intersect the underlying Wongawilli Seam
ahead of the operating longwall face. Around 12000 metres are rotary drilled annually with this rig utilising
Longyear and Wadam 65 mm PCD bits and 1.5 metre long BQ rods. Steel standpipes of 75 mm diameter and 6
metres in length are fully grouted into the coal.

A fan pattern is used with varying dip angles and length up 150 metres. Holes are not surveyed but return water
colour is monitored and recorded by the drillers. Casing of the holes has not been practiced in the past but a trial
is being planned.

Overall with in seam and cross measure drainage, gas capture around the longwall panel approaches 50% of the
gas desorbed and emitted.

Goaf Drainage
Tower does not currently drain gas from old goaf areas although drainage from a recently sealed area of the mine
is being investigated.

Outburst Management
Core sampling for both gas composition and content is carried out in line with the Tower Outburst Management
Plan. All cores are sent to the Technical Services Laboratory for analysis and reporting. In areas where the coal is
difficult to drain, many cores show minimal desorption until they are crushed, after which significant quantities of
gas are released. Measured gas contents over 15 cubic metres per tonne have been recorded.

West Cliff Mine
The West Cliff Mine is currently mining the third longwall panel in a block of coal adjacent to old workings at
Appin. The previous longwall domain at West Cliff was characterised by areas of high gas content in excess of 20
cubic metres/tonne, ranging in composition from 90% plus methane to 90% carbon dioxide. Methane
predominates in the current area which is intersected by a number of geological structures which present a
challenge to gas drilling and drainage

In Seam Drilling
West Cliff utilise two Kempe K200 Drill Rigs recently transferred from Tower Mine and one Longyear LMC 55
Rig to drill between 65 and 70,000 metres of in seam hole annually. Downhole Motor directional drilling is used
in conjunction with acoustic DDM Upgrades and Mecca survey instruments to guide and accurately plot hole
trajectory. Holes are surveyed at 6 metre intervals.

In seam holes are drilled in a modified fan pattern across the longwall block. Each hole is generally branched once
with holes sub parallel to each other ahead of the next gate road development. Standpipes of 100mm diameter and
5 metres in length are grouted into the hole collar. They are either copper or galvanized steel depending upon the
collar location in relation to mining activity. Hole diameter is 96 mm and length varies between 320 and 1000
metres. The longer or more irregular holes double as exploration holes and are typically drilled to prove structure
or to provide drainage around difficult to drill or drain areas. PCD bits are used in conjunction with CHD76 rods if
the acoustic survey tool employed or NRQ-HP rods if Mecca tool used.

All holes are logged for anomalies by the drillers and records kept for future reference. Development of a survey
and drilling system with the ability to distinguish between roof and floor strata and to provide feedback for
guidance is seen as a very useful potential improvement to increase drilling and drainage efficiency.

Cross Measure Drilling
A Longyear LMC 55 rig is used to rotary drill between 22 and 25,000 metres of cross measure hole annually. They
are drilled using 1.5 metre long BWJ rods and 65mm PCD bits and stabilisers. Galvanized steel standpipes, three
metres in length, are used on each hole. A fan pattern of four to five holes angled below and toward the advancing
longwall face at a declination of 20 to 28 degrees is drilled with up to four sites per pillar length. Holes are not surveyed but are logged and recorded by drillers. Hole length varies between 130 and 165 metres in length.

Perforated casing of 48 mm diameter has been trialled on cross measure holes but has not yielded improvements in efficiency or gas flows under current West Cliff conditions. Gas drainage performance based upon cross measure flows only indicates capture efficiencies of between 30 – 40% around the longwall block.

**Goaf Drainage**
Goaf drainage is not practised at West Cliff due to past inability to control leakage and purity levels.

**Outburst Management**
Core sampling for both gas composition and content is carried out in line with the West Cliff Outburst Management Plan. All cores are sent to the Technical Services Laboratory for analysis and reporting.

**Gas Utilisation**
The drained mine gas from Appin, Tower and West Cliff is currently utilised in power stations located at both Appin and Tower. The gas turbine at West Cliff was decommissioned when repair was not economically justified and an alternative generation source became available at Appin

The gas drainage systems at Appin and Tower Mines are interconnected via a physical link joining the two mines underground. Drained mine gas captured in the pipeline can effectively be used to feed either utilisation plant. Currently some Appin mine gas is diverted to Tower to supplement that power station’s fuel supply. In addition both power stations are connected to natural gas from the AGL system which can also be used as a supplementary source of fuel when economics permit

Mine gas that is collected from the West Cliff methane drainage system is reticulated to the Appin power plant via an overland pipeline connecting the two sites.

The Appin power station consists of 54 gas engines with a nominal generating capacity of 1 MW each while the Tower power station comprises 40 such units. The electricity generated is sold to Integral Energy under a long term agreement. The power stations are owned and operated by Energy Developments Limited who are contracted to BHP Billiton to produce electricity from the mine gas.

**OUTBURST MANAGEMENT PLANS**
The three mines utilise a Mine Safety Management System approach to manage the outburst and gas related hazards. More particularly an Outburst Management Plan, specific to each mine, is used with the stated objective of effectively controlling the risks associated with outburst. The detailed plans are based upon common principles with some variation between sites arising from different mining conditions, history and experience.

Plans set out the methodologies and activities that are mandatory for predicting potential for outburst and, as far as practicable, preventing their occurrence. Elements of the plans address the protection of personnel from the effects of an outburst should all other barriers be found to be deficient in any way. Threshold levels of methane and carbon dioxide are central to the prediction and prevention strategies in the plans. They nominate the gas content levels above which no mining will take place or where specific full risk assessments must be used to determine the risk of mining. Threshold limits for 100% methane range from 9 to 9.5 cubic metres per tonne while for 100% carbon dioxide, the range is from 5 to 6 cubic metres per tonne.

The Outburst Management Plans contain many controls and measures to ensure the effectiveness of the plan to control the outburst risk. Some of the main measures include

- Drilling, gas drainage and sampling requirements
- Collection of relevant information
- Stipulation of decision making and communication procedures
- Clear allocation of responsibilities under the plan
- Training both in the operation of the plan as well as in the recognition of outburst warning signs
- First response equipment and planning in the event of an outburst occurring
The plans are constantly under review with formal audit requirements built in to keep them current, effective and up to date.

GAS DRAINAGE PERFORMANCE

The table below is a record of the volume of methane captured at the group mines for the year ending June 2001. The gas captured represents the volume reporting to the surface of the mines in the gas drainage pipeline. The ventilation gas is that gas which reports to the ventilating air circulating around the mine and exhausting to the atmosphere at the surface.

<table>
<thead>
<tr>
<th></th>
<th>Gas Captured</th>
<th>Ventilation Gas</th>
<th>Total Gas Make</th>
<th>Proportion Gas Capture</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>metres$^3$x 10$^6$</td>
<td>metres$^3$x 10$^6$</td>
<td>metres$^3$x 10$^6$</td>
<td></td>
</tr>
<tr>
<td>Appin</td>
<td>92.5</td>
<td>171.3</td>
<td>263.8</td>
<td>35%</td>
</tr>
<tr>
<td>Tower</td>
<td>58.6</td>
<td>136.4</td>
<td>195</td>
<td>30%</td>
</tr>
<tr>
<td>West Cliff</td>
<td>46.9</td>
<td>117.3</td>
<td>164.2</td>
<td>29%</td>
</tr>
<tr>
<td><strong>TOTAL</strong></td>
<td><strong>198</strong></td>
<td><strong>425</strong></td>
<td><strong>623</strong></td>
<td><strong>32%</strong></td>
</tr>
</tbody>
</table>

The proportion gas capture represents the gas drainage efficiency for the mine and shows significant room for increase if technology and gas drainage techniques are improved. Captured methane represents an opportunity to recover a proportion of gas drainage costs through utilisation provided a viable economic arrangement can be established. In addition, as the gas capture increases, the risk of mining related gas delays and outburst occurrences is reduced.

Technologies designed to utilise methane contained within the ventilation stream are actively being investigated and trialled. There is also an environmental link with potential green house gas reductions if the methane emissions can be effectively utilised and burnt.

OPERATIONAL ISSUES

There are many issues that confront our mines and operators in relation to outburst control and gas management. A selection of these are noted and discussed briefly in the points that follow. They are meant to act as thought provokers and a rough checklist of areas our operators perceive could provide increased safety and efficiency to underground mining if further examined. Issues are not presented in any particular order or priority.

Outburst Threshold Limit Review

There is often the temptation to increase gas content threshold limits for samples that return just above the limit analyses. A factor of safety no doubt exists in the setting of the limits but it is very difficult to scientifically quantify what it is in a specific case. A better understanding of the outburst mechanism is required to justify significant variation to the threshold limits. At least some research should continue understanding this fundamental end of the research spectrum.

Difficult to Drill Areas

Difficult to drill areas of mines are often encountered in inseam gas drainage drilling. The coal may be soft with subsequent hole instability or highly stressed as in areas surrounding geological intrusions or structures. Drilling to reduce gas contents below threshold limits is therefore hampered with subsequent risk of discontinuities in mining operations.
Some mines have successfully traversed such local zones by drilling into the roof or floor in the area of the disturbed zone to drain coal on the other side. Casing of holes as they are being drilled or shortly thereafter may aid stability through such zones. Drill bit technology also needs to be considered where varying material is encountered in the one hole and to optimise drill rates.

**Difficult to Drain Areas**

A common issue across our mines arises when mine planning and access do not allow adequate lead time to drain seam gas content to acceptable levels. The most common response to this problem is to increase hole density and reduce space between holes in the drainage pattern. Some areas of the mines have been encountered where regardless of this response the seam gas will still not drain at any appreciable rate. Other approaches may be considered, including:

- A number of mines have successfully fractured horizontal in seam holes with high pressure water and have successfully stimulated increased gas flows
- In some cases where the drainage holes are connected to a source of suction it may be possible to provide additional negative pressure to further enhance desorption rates
- In individual critical cases it may be beneficial to ream out the hole to provide both a reduction of resistance to flow and an increased surface area for the desorption process
- Early identification of difficult to drain areas may give additional lead time for drainage to occur by flagging up the need for early drilling. Some research in the microscopic investigation of samples is showing promise in this aspect.

**Planning**

Although significant planning of gas drainage and drilling takes place there is probably still more that could be done to optimise the gas drainage and drilling performance. One mine has noted that results have been improved by involving the actual driller to a greater degree in planning and taking into account as many variables as practical with past performance feedback.

Similarly an ergonomic review of the drilling site has designed out bad work practice, enhanced OHS aspects of the task and improved drilling rates.

Mine planning has a role to play in outburst and gas management in that lead time is necessary to drill and drain to reduce gas content of the coal efficiently. Unexpected adverse geology can often lead to mine development in areas with little notice. Gas drainage to acceptable levels may still be possible by increasing the density of drilling but is not likely to be as cost effective or efficient compared to an area where ample warning is available.

**Hole Trajectory**

The majority of in seam holes are currently surveyed in three dimensions. This is necessary to confirm hole and sample location. Currently the general method of establishing the hole location in relation to the seam floor or roof is by periodically touching these interfaces. A system of continuously monitoring borehole location relative to roof and floor would greatly enhance drilling performance.

**Flow Monitoring and Feedback**

There is likely to be a benefit from case studies of gas flow performance as feedback mechanism for design of future holes. This is no doubt done to some degree for both in seam and cross measure drainage but more extensive surveys and analysis could no doubt increase efficiency. The potential benefit lies in detailed case studies of high flowing holes in comparison to surrounding low flowing ones. The reason for the variation is important.

**Determination of Outburst Potential**

Authorisation to mine must be accompanied by confirmation that gas content of the coal ahead of mining is below a threshold limit. This generally involves drilling and sampling. Gas analysis of the coal samples can be complete
in around three hours at the laboratory but more realistically, taking into account the drilling component, twenty-four hours is more likely.

An on site method of determining gas content or proneness, particularly where levels prove to be either well above or well below the limits, would be beneficial in saving potential delay times waiting for results or increasing gas drainage activity. The laboratory analysis may still be necessary to determine the near limit values but may be superfluous in determining the obvious pass or obvious fail sample results.

CONCLUSIONS

Outburst and gas management is generally effectively managed at the three mines as described. However there is always the need to improve understanding of the outburst phenomenon and the mechanism of gas drainage. This need is primarily driven by safety concerns resulting from outbursts or gas emissions not being effectively controlled. The efficiency of the control measures has an obvious potential impact on productivity and economic viability of the coal mining operations. There is always room for improvement.

There exists a definite link between gas drainage and outburst control. This link can be extended to gas utilisation and even to greenhouse gas control as that debate continues. The latter two issues may become substantial drivers that influence the safety and productivity of operations.

ACKNOWLEDGEMENT

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GAS DRAINAGE EXPERIENCE AT THE
NORTH GOONYELLA MINE

Peter Allonby

ABSTRACT: The North Goonyella Coal Mine has been owned in Joint Venture by RAG Australia Coal and Thiess NG Pty Ltd since November 2000. The mine is operated under contract by Thiess. Effective gas drainage is a fundamental requirement for the safe and efficient operation of the mine.

Drainage has been effective since 1999 and with access to Australian and German knowledge, techniques will continuously be improved to enable the mine to economically mine deeper and gasier reserves without compromise in the safety of the mine and its people.

INTRODUCTION

The North Goonyella Coal Mine in the Bowen Basin was the first underground mine in the Goonyella Middle seam. It was also the first thick seam high capacity longwall operation in Australia.

The seam dips at 1 in 12 and there is a rapid increase in gas content in the mine’s northern workings to around 10m³/t at 250m in depth. The seam gas is predominantly methane.

The lease is highly structured with a high incidence of both normal and low angle thrust faults. The mine clearly must be regarded as having outburst potential. The mine layout is shown in Fig. 1.

Fig. 1 Mine Layout

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1 Thiess Pty Ltd
OUTBURST MANAGEMENT

The majority of Australia’s outburst knowledge has been derived from numerous events in the Bulli seam of the New South Wales Southern Coal Field but it must be noted that both methane and carbon dioxide outbursts occurred some years ago at mines to the north and to the south of North Goonyella.

Based on the Bulli seam experience the gas and structural conditions at North Goonyella dictated that outburst management precautions be implemented.

A technical evaluation of conditions at North Goonyella determined the following key outburst management factors;

1. Accept a gas outburst threshold of a Desorption Rate Index of 900 “equivalent to the threshold below which outbursts will not occur in the Bulli seam”. This equates closely to 7 m³/t at North Goonyella and 9.5 m²/t in the Bulli seam.
2. Provide a protection barrier of 15 m around development headings (compared with 5 m in the Bulli seam).

Significant differences from the Bulli seam are the seam height, permeability and seam dip. The application of Bulli seam knowledge to the Goonyella Middle seam must be carried out with great caution.

GAS DRAINAGE

Gas drainage in advance of development at North Goonyella has been practiced since 1999 to mitigate outburst risk and to control general body gas levels. Drilling commenced from the travelling road of LW1N Maingate through LW2N to pre-drain MG2N development. MG4N is currently being developed. Typical gas drainage patterns are shown in Fig. 2.

There is sufficient gas pressure to enable a fan of underground holes to free vent via surface-to-seam boreholes called gas risers. Drilling is via conventional down hole motor technology with continuous survey.

Drilling conditions are challenging due to the prevalence of structure but such drilling provides a valuable addition to the mine’s exploration tool box.
Permeability is generally good but two low permeability zones have been encountered. These zones were identified during gas drainage drilling which allowed actions to be implemented to minimise delays to development.

A specialist’s report in March 2000 stated “gas drainage under North Goonyella conditions is likely to break new ground in gas drainage understanding”. The mine operator is constantly striving to gain that understanding and improve the cost effectiveness of its gas drainage operations.

The gas drainage system implemented initially has proven very effective to date and the only improvements made have been:

1. Riser diameter has been increased to reduce back pressure because hole flows increase with depth.
2. Surface and underground “plumbing” has been similarly increased in size.
3. Hole lengths have been increased to ensure the 15 metre zone around development headings are effectively drained. The build-up of fines at the end of the holes limits the effectiveness of the hole ends and the extremities of the holes are being recharged by gas from depth. Holes are now drilled 70 metres beyond the development headings. The effect of hole ends on drainage is shown in Fig. 3.

Gas conformance cores are taken to the edge of the 15 metre protection zone at nominally 100 metre spacings. Cores are generally taken from the surface and the full coal seam is cored. Vertical variability within the seam is up to 2 m³/t. The lower third of the seam generally has the highest residual gas content. The core from the lower third of the seam is analysed and if its content + 2 m³/t confirms with the 900 DRI, no further analysis is carried out. The cores from the upper two thirds of the seam will be analysed if the lower core fails. If any core fails the area is deemed inadequately drained and no mining will proceed until gas contents have been further reduced and compliant cores obtained. In the two low permeability zones encountered so far numerous additional drainage
holes were drilled and cores taken before mining in the compliant adjacent heading proved effective in enhancing permeability.

In-seam cores are taken on occasions, again targeting the lower third of the seam. Cores will be taken at a number of horizons if there is any doubt regarding the representiveness of the initial cored horizon.

Core frequency is increased around identified structures or in areas where hole flow measurements indicate an area may have poor drainage.

In simple terms, cores are taken where the highest residual gas is expected.

OUTBURST MINING PROCEDURES

No procedures exist or have been contemplated for mining in non-compliant areas.

VALIDITY OF BULLI SEAM KNOWLEDGE

Standards adopted for the Bulli seam have been used as a basis for setting conditions for North Goonyella

Barrier Pillar Sizing

There was little science behind the determination of the 15 metre protection barrier for North Goonyella.

This assumption has been recently challenged with two considerations.

1. Is the 15 metre barrier excessive at current depths?

2. Is the 15 metre barrier adequate for greater depths?

A study was initiated to look at the geomechanical strata behaviour and gas fluid flow mechanics as they are expected to vary with depth.

The behaviour at 400 metre depth was then compared with the Bulli seam.

The study suggests that the North Goonyella behaviour is quite similar to the Bulli seam thus the application of Bulli seam knowledge may be quite valid.

The study is yet to be finalised.

THRESHOLD LIMIT

The Desorption Rate Index (DRI) is a measure of gas desorption rate derived from GeoGAS's method of fast desorption gas content testing. In its determination, a standard mass of coal (200 g) is crushed at a standard rate of crushing. The quantity of gas desorbed after 30 seconds crushing is used as a relative indicator of desorption rate. From gas content thresholds successfully applied in the outburst prone Bulli seam mines a DRI of 900 has been identified as a threshold below which outbursts will not occur, regardless of the structural state of the coal. For the Bulli seam, a DRI of 900 is equivalent to a gas content of 9.5 m$^3$/t pure CH$_4$ and 6.0 m$^3$/t pure CO$_2$.

For the same gas content, the Goonyella Middle seam at North Goonyella has a higher rate of gas desorption. The 900 DRI occurs at a gas content of 7.0 m$^3$/t for essentially pure CH$_4$ and has been accepted as gas content threshold applying to mining at North Goonyella.

There has been no reason to challenge the validity of this threshold for North Goonyella.
EFFECTIVENESS OF GAS DRAINAGE

The 900 DRI threshold equates closely to 7m³/t. Gas drainage is designed to reduce gas to <4m³/t and the mine ventilation system is designed for 5m³/t.

Residual gas contents at the development headings, in practice, are rarely greater than 2-3m³/t.

Intersection of gas drainage holes by the continuous miner creates little problem with general body gas levels but often causes problems with water seepage on to a floor which degrades badly. Drainage holes in roof coal in close proximity to the development roof horizon cause more difficulties because they are difficult to seal.

OUTBURST HISTORY

An outburst was recorded on 22nd October 2001.

The continuous miner had ceased cutting and was positioning to commence bolting when the operators noticed coal flowing from the face “like black water with waves in it”. The miner is set up with two gas detectors. The detector at the cutting heads detected 3.5% but the detector at the tail which is set to trip the machine in a general body of 1.25% detected no increase.

By the definition of an outburst as a “sudden release of coal and gas under pressure” the event was arguably not an outburst. Fig. 4 shows outburst material at the drill rig.

Irrespective of the interpretation of definitions the event has been regarded by site management as an outburst and a thorough investigation into the event has been carried out and gas drainage and outburst management practices reviewed.

The initial observations at the site showed;

1. Approximately 6 tonnes of pulverized, dry coal loosened from the face.
2. No significant variation in mining conditions prior to the event.
4. No sign of structure.
The site was cleaned and supported to allow a thorough investigation. This investigation identified:

1. The presence of a strike slip fault.
2. Mylonite in the fault.
3. Red discolouration of coal in another heading along the projection of the fault.

All employees on roster were taken to see this set of “classical” Bulli seam outburst indicators.

A comprehensive program of core sampling in three horizons identified no abnormal gas variances in the vicinity of the structure.

It certainly appears that conditions existed which may have resulted in a sizeable outburst had gas drainage not been effective.

CONCLUSIONS

North Goonyella experience to date with gas drainage and outburst management practices is not inconsistent with the Bulli seam.

The mine however is continuing to increase in depth (and has a potentially mineable lower seam) and will continue to increase its understanding of gas and structural behaviour. This knowledge, combined with predictive tools will endeavour to establish practices and procedures for managing the changing circumstances at the mine prior to those changes becoming detrimental to the safe and efficient operation of the mine.
**USE OF BOOSTER FAN VENTILATION AT WEST CLIFF COLLIERY**

David Benson¹

**ABSTRACT:** Large underground coal mining operations, such as West Cliff are continually having to balance and co-ordinate ventilation requirements for each working face, with the overall management of gas both CO₂ and CH₄ within the mine. As mining operations expand, along with areas of goaf the potential for gas make increases. It becomes increasingly more difficult to maintain adequate ventilation to the development panels with corresponding effects or consequences on coal production.

In the case of West Cliff mine, the performance characteristics for the gas drainage system as well as the overall ventilation performance for the entire mine were fully reviewed and assessed. Various options to improve the overall efficiency of both the gas drainage system and the mines ventilation system were identified and considered. Ventilation planning and simulation allowed the various options to be reviewed in detail, highlighting the benefits and short comings of each modification or adaptation to both systems.

The use of booster fans was considered in respect of the following potential benefits:

- Provision of additional air quality at strategic locations within the mine and separate from the longwall split.
- Increased pressure to drive airflow into working splits
- Reduction of intake to return pressure differential, outbye of the booster fan location
- Reduction in outbye leakage
- Reduction of main fan airflow requirements for a given inbye performance
- Optimisation of power utilisation

The use of booster fans for underground mining operations has an extensive history of satisfactory and effective utilisation over seas, especially in the UK, Canada and Germany. Hence the use of booster fans to meet the ventilation and gas management needs of West Cliff could be thoroughly researched. This included the following inherent hazards:

- Potential for uncontrolled recirculation
- Potential ignition sources and metallic contact
- Potential for fire
- Spontaneous combustion
- Potential to pass gas plugs through fans on re-start
- Mechanical failure

This paper outlines the overall research and assessment approach adopted by West Cliff Colliery, focusing on the following key components:

- History of Booster fan use
- Overseas practice
- Department of Mineral Resources approvals
- Risk assessment process
- Major Hazards identified
- System description and design
- Control strategies

The experience gained at West Cliff Colliery demonstrates that booster fans represent a cost effective means of ventilating strategic sections of the mine on a life of mine basis. The risk assessment approach and the controls implemented as a direct consequence of this approach have supported the successful installation and operation of booster fans.

¹ West Cliff Colliery
ASSESSMENT OF RIB INSTABILITY HAZARDS FOR STRATA MANAGEMENT SYSTEMS

John Shepherd¹

INTRODUCTION

The support and control of unstable coal mine ribs is an on-going problem for the industry but in actual fact systematic research investigations were started almost 20 years ago in Australian Coal Industries Research Laboratory (ACIRL). This came about through the interests of the Australian Coal Association and the Queensland Coal Association Thick Seam Mining Committee. Some work was also carried out on coal mine rib mechanics under the NERDDC program in the mid 1980’s. Several collieries were directly experiencing fairly acute rib control problems at this time and the research work was linked with these. A number of publications which have been largely forgotten resulted from this work: O’Beirne and Shepherd (1984), Shepherd et al (1984), O’Beirne et al (1985, 1986, 1987), but these and later studies form a good basis for the methodology of rib hazard identification. Other rib support research has also been carried out at the University of New South Wales: (Hebblewhite et al, 1998) have highlighted the need for matching the support to the dilational movements found in ribsides.

The assessment of rib conditions can be viewed as occurring in three broad phases during the life of mine workings as follows:

- as early as possible during the cutting of the development faces.
- continue through the formation of the pillars and while the pillars stand because there is often time dependent movement.
- during and after secondary extraction as stress abutments develop in both pillar and longwall extraction.

As a general rule in most seams rib instabilities are not problematical at cover depths of less than 100m unless the coal is particularly weak. However, during secondary extraction even shallow workings are subject to abutment stresses.

WHAT PRODUCES RIB HAZARDS?

Hazards are produced by the interaction of the natural coal seam variables and various mining-induced factors which can be modified according to the mine design (see Table 1). This paper is not examining the rib failure mechanisms in detail but will attempt to identify the principal types of hazards that should be accounted for in a strata management plan.

<table>
<thead>
<tr>
<th>Natural factors</th>
<th>Mining induced/design factors</th>
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<tbody>
<tr>
<td>Coal seam banding (plies) and their strength</td>
<td>Roadway width, first workings; stress abutments</td>
</tr>
<tr>
<td>Presence of dirt bands</td>
<td>Pillar size versus depth</td>
</tr>
<tr>
<td>Seam thickness</td>
<td>Roadway profile</td>
</tr>
<tr>
<td>Cover depth (stresses)</td>
<td>Mining direction</td>
</tr>
<tr>
<td>Seam dip gradient</td>
<td>Working height (in thicker seams)</td>
</tr>
<tr>
<td>Cleats</td>
<td>Straightness of ribsides</td>
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</table>

In general, rib instability hazards are produced by the coal fracture systems, some of which are cleats and some induced during the mining processes. Some early work on the interaction of cleat and mining induced fractures

¹ Shepherd Mining Geotechnics (Aust.) Pty Ltd
was carried out at Leichhardt Colliery by Hanes and Shepherd (1981). Mineralised cleats in the vitrinite (bright) bands were found to propagate through the dull bands to form induced fractures.

Later on, work by Shepherd et al (1984) and O’Beirne et al (1987) demonstrated an array of quite complex fracture interactions in ribsides that produced instabilities. The development of these induced fracture systems depends especially on cover depth, the ply banding in the coal and the distribution of dirt bands which have markedly different physical properties from the coal.

All the factors listed in Table 1 can play a role depending on the local circumstances and the mine layout and mining method.

**ASSESSMENT OF RIB INSTABILITY HAZARDS**

Assessment of rib instability should be carried out by competent geotechnical personnel and should generally consist of two types of activity:

- detailed rib mapping throughout the life of the workings, but especially starting at the development faces. This should include logging the coal brightness, strength testing and rib fracture identification. It is most important that the cleats are distinguished correctly from the induced fractures.
- installation of pillar rib instrumentation such as extensometers and instrumented rib bolts.

The aim of this work is to define the hazards for the purposes of risk assessment and it may initially determine the need for rib support. Instability of ribsides is a function of coal dilation, fracturing and the resultant size distribution of the detaching material. Dilation can occur by tensile extension in the ribs, or the combination of this and shear if there is sufficient confinement. Induced cracking can start ahead of a development face, interact with pre-existing cleats or joints and rapidly produce hazardous situations. A particularly useful hazard assessment scheme that can be incorporated in to hazard and risk assessments is given in Figure 1 (after O’Beirne et al, 1986).

![Figure 1](image)

**Fig. 1** Probable mode of rib spall according to drivage direction in first workings, cleat and MIF (Mining Induced Fractures) shown schematically in plans (a) to (f). S = slabs/plates (spot support), B = blocks/columns (extended spot support), P = particles (liner support)

If support is required, it should be installed as soon as possible. A common feature of underground workings is the accidental damage to rib support hardware and experience indicates that this is rarely fixed. Later on, this increases the hazard severity quite markedly and there are a number of cases where mine personnel have been severely injured precisely at the locations of damaged support.

For personnel protection, the main hazards are coal slabs and blocks and these commonly exist in ribsides as follows:
• undercut high blocky ribs
• large cleat-bounded slabs
• large cleat-bounded blocks and/or columns prone to toppling or sliding if unsupported.

In general, the panel layout will determine where the high risk places are, such as at pillar corners on intersections in first workings and in stress abutment zones in secondary extraction. During pillar extraction, the highest risk is in the fender ribs adjacent to the active lift and along the ribside opposite. In longwall retreat panels the highest risk places are generally within the front abutment zone at the maingate face corner and in the chain pillar ribsides for some distance outbye from the chocks. This is a particularly hazardous area because it contains significant amounts of coal face clearance equipment.

In some seams, the risk from rib hazards can be reduced by limiting the working height to about 3m. This may work reasonably well in first workings but may be less effective in secondary extraction where abutment stresses may increase insidiously, resulting in sudden rib spall events that are notoriously unpredictable and have resulted in deaths and serious injuries.

CONTROLLING RIB INSTABILITY HAZARDS

Mapping at a large number of sites has resulted in the development of a simple classification system for rib spall material. The size of the material detaching is critical in terms of the risk to personnel and in Table 2 coal particle sizes are given. The small particle sizes are generally not hazardous unless unusually high stresses occur during development or extraction when it is possible to have non-gassy bursting during which the coal is forcibly ejected. The main risk stems from the larger material which needs pinning up to the ribs by support. A useful support system concept was published by O’Beirne et al (1986), and this is summarised as follows:

• spot support installation of bolts or dowels for slab and large column control
• extended spot support – bolts or dowels linked by strapping. These are particularly useful at pillar corners, in stooks or at any site containing narrow failure zones.
• Liner support – where full protection is required in high abutment zones or where the ribs disintegrate into particles. These can cover all or part of the working section.

In terms of a strata management plan, once the rib hazards have been assessed, it is preferable to relate the support needs to the hazard classes. In view of the regulations now in force, and OH & S principles, it is necessary to follow the guidelines laid down by the DMR (1999) and Standards Australia (2000). A generalised scheme to achieve this is given in Table 2. In reality, this would need to be site specific after geotechnical studies had been carried out, and it should only be used as a broad guide.

Table 2. Classification scheme for spalled blocks and outline of probable support needs.

<table>
<thead>
<tr>
<th>Shape of hazard</th>
<th>Smallest dimension in plane of ribsides (m)</th>
<th>Rib support needs</th>
</tr>
</thead>
<tbody>
<tr>
<td>Blocks and slabs</td>
<td></td>
<td></td>
</tr>
<tr>
<td>- very large</td>
<td>&gt;1.0</td>
<td>Close pattern of bolts or dowels and straps or mesh</td>
</tr>
<tr>
<td>- large</td>
<td>&gt;0.3-1.0</td>
<td></td>
</tr>
<tr>
<td>Columns and small blocks</td>
<td>0.01-0.3</td>
<td>As above, spacing to suit</td>
</tr>
<tr>
<td>- medium</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Particles</td>
<td>&gt;0.02-0.1</td>
<td>Mesh or liner type</td>
</tr>
<tr>
<td></td>
<td>&lt;0.02</td>
<td></td>
</tr>
</tbody>
</table>
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Strata Management in Weak Roof Conditions at Crinum Mine

Dan Payne¹ & Barry Ward²

Abstract: Crinum Mine is located in the Bowen Basin and is operated by BM Alliance Pty Ltd. Longwall extraction commenced in 1997 in the 3.4m thick Lilyvale Seam and seven panels have been mined to date in the current block. The roof strata are variable and include some of the weakest longwall mining conditions in Australia.

The immediate roof is predominantly a 15 to 25 MPa thinly bedded sandstone but includes weaker siltstone layers, typically 1m to 2m thick, with an average strength of 10 MPa. In some parts of the mining area the whole of the immediate roof appears to change laterally to a laminated siltstone with a strength as low as 2.5 MPa.

As the presence of these weaker horizons can influence the effectiveness of the primary and secondary support, as well as the roof stability on the longwall face, delineation of roof conditions forms a significant part of the strata management process. Strata are interpreted into geomechanical units from the sonic velocity logs to enable zones of similar conditions to be identified. More detailed interpretations are made along section lines for maingate development and panel hazard plans.

Roof support and longwall management strategies are based on the delineated roof conditions. Roof performance is monitored by means of dual height telltales, mechanical, and three and four point electronic telltales, GEL extensometers. A substantial base of mining experience has been accrued, enabling support requirements to be matched with confidence to the predicted roof conditions.

Introduction

Crinum Mine is located in the Bowen Basin and is operated by BM Alliance Pty Ltd. Longwall extraction commenced in 1997 mining the Lilyvale Seam and seven panels have been mined to date in the current block. The Lilyvale Seam, which is the lateral equivalent of the German Creek Seam, has a consistent thickness averaging 3.4m.

The geological setting is that of a shallow plunging syncline with an axis down the middle of the block. Dip values vary between 2.5° and 3.5°. The depth of cover ranges from a minimum of around 95m to a maximum of 200m. Longwall panels are dip oriented and mined to the rise; face length is 270m and the full seam thickness is extracted. Fig. 1 shows the layout of the current workings.

Gateroad development is two heading at 35m centres with cut throughs at 130m centres. Roadways are 4.8m wide and nominally 3.3m high, cutting 100mm roof and leaving a minimum 200mm of coal in the floor as the stone floor is too weak to withstand trafficking. Drivage is conventional cut and bolt using Joy 12CM30 continuous miners with four onboard Hydramatic roof bolters which bolt within 2m of the cut face.

The roof strata are variable and include some of the weakest longwall mining conditions in Australia.

Roof Conditions

The lower roof section from the Lilyvale Seam to the Corvus Seam has been interpreted in terms of four major geomechanical units that can be correlated across the panels with a fair degree of confidence. Identification of these units is based largely on interpretation of the sonic velocity logs. Fig. 2 illustrates the sequence of roof units.

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The major component in the lower roof, Unit 2, is a moderately strong thinly bedded medium sandstone with numerous siltstone laminae and interbedded thin weaker siltstone layers. It has a thickness usually between 10m and 15m but can range up to a maximum of 20m in places. The estimated average strength of this unit tends to be consistently between 15 and 25 MPa.

Sometimes there is a weak to moderately weak siltstone, Unit 1, directly overlying the seam. It has a patchy development over most of the block, with a thickness typically less than 0.5m, but is more persistent to the north and south, where the thickness is from 0.5m to 1.4m. The estimated average strength of Unit 1 is typically between 7 and 10 MPa but drops to as low as 3MPa in some areas.

The interbedding in the main sandstone is highly variable with no distinct correlation of individual layers between boreholes. However, two thicker, more persistent siltstone horizons have been identified within Unit 2 and have been designated as Units 2A and 2B, as indicated in Figure 2. For the most part these two geomechanical units correspond to correlatable geological horizons but in some cases they appear to die out laterally and reappear at a different horizon and thus may not necessarily be geologically continuous. The estimated average strength of these weaker horizons varies from 7 to 15 MPa, with an average of around 10 MPa.

Unit 2A refers to horizons occurring directly on top of Unit 1 or directly on top of the seam where Unit 1 is absent. These are usually from 1m to 2m thick but have been interpreted as up 6.3m thick in the southern part of the block. The presence of Unit 2A material usually means a reduction in strength at the primary bolting horizon. Unit 2B refers to weaker horizons at higher levels from 2m to 10m above the seam roof. These are also typically 1m to 2m thick and their presence can influence the strength of cable anchorage horizons.

Overlying the main roof sandstone is a very much weaker thickly laminated siltstone, Unit 3, with a thickness usually between 2m and 7m, and an estimated average strength typically between 5 and 10 MPa. Normally this unit and the remainder of the strata, Unit 4, are too high in the sequence to impact on roof support design. However in some places the weak Unit 3, as a geomechanical unit, increases in thickness at the expense of the more competent main sandstone, reaching a maximum thickness of up to 9m in places. It is believed in these instances that Unit 2 grades laterally into Unit 3, not as a geological variation but as a geomechanical change, as the sandstone matrix appears to soften and become clayey causing a significant strength reduction. Estimated strengths as low as 2.4 MPa have been interpreted from the sonic logs.

All available sonic velocity logs are examined and interpreted in terms of the geomechanical roof units described above, and the results compiled into a borehole database. This geomechanical database is used to provide the basic framework for the maingate roof sections described below.

**ROOF SUPPORT**

Roof support design has to cater for what is relatively weak conditions with low strength laminated rock. Current roof support patterns have been evolved through experience, trial and monitoring and are as follows:

1. **Primary Support**
   
   Moderate roof: 6 x 2.1m bolts at 1.0m centres, with mesh mats. Bolts are anchored with two stage resin and are pre-tensioned to nominally 10t. Although this has been the normal support pattern used at Crinum, a decreasing trend in average roof strengths has reduced the usage of this pattern to 50% of development drivage in gateroads and 0% in the Mains.

   Poor roof: 8 x 2.1m bolts at 1.0m centres, with mesh mats, bolts in a 6 + 2 pattern. This pattern is used where the bolting horizon strength is less than about 10-12 MPa; which, as previously stated, is about 50% of development drivage in the gateroads and 100% in the Mains.

2. **Secondary Support**

   Groutable Megabolts are currently the long tendon support employed and are installed as follows:
   
   - in response to roof movement i.e. trigger response
• in widened sections e.g. installation roads, drivehead areas
• in critical areas e.g. face take-off headings and intersections
• in longwall belt road areas where geologic interpretation suggests weak roof and some deterioration has become visible.

The length of megabolts employed (6m, 8m or 10m) depends on the lowest horizon at which reasonable roof strength exists as determined by the sonic velocity logs using both roof strength horizon contours and roof unit sections described below. The support density is designed to work in cooperation with the primary support pattern and the geometry of the opening, and match the degree of deterioration already experienced and the service required whether longwall abutment, roadway widening, or mine life requirement.

Megabolts are point anchored with resin which sometimes results in heavy loads being experienced at the roof line. 300mm plates are required at Crinum due to the weak roof and the tendency for 200mm plates to exceed the bearing capacity and punch into the roof. When very high loading is observed on 300mm plates the Megabolts® in the area are grouted with Ordinary Portland Cement (OPC) eliminating the potential for full loss of support if the cable breaks at the bearing plate. Grouting has been a very effective form of tertiary support. In critical areas such as faceroads and takeoff intersections where rates of movement are too high to safely inject OPC, PUR has been injected into the Megabolts. This technique fully encapsulates the Megabolts as well as injecting PUR into the roof strata without the requirement for additional drilling which would entail time, expense and further deterioration of the roof due to water injection. It is estimated that only 5% to 10% of Megabolts installed at Crinum are grouted.

GEOTECHNICAL INPUT INTO THE STRATA MANAGEMENT PLAN

Geotechnical information is used at three levels in the strata management process.

In the first instance, a contour of the bolted horizon roof strength for the mine is prepared from the sonic velocity logs of surface boreholes. Figure 3 shows the interpreted bolting horizon strength over the current longwall block from which the contour plots are generated. The contour plot has been back analysed for accuracy in predicting roof control problems, telltale results and long tendon support requirements and found to be extremely representative. The updated contour is then used to budget primary support patterns involving costs and development rates and secondary support requirements utilising costs and material requirements and even assist in mine planning longwall setup and takeoff points.

In the second instance, the roof UCS contour is detailed over individual development panels. The 2m roof strength contour assists in the decision to implement different primary support patterns which are designed such that longer bolts using two pass drilling is not required at the face. Sufficient roof stability is achieved by the primary pattern so as to allow completion of the development sequence before secondary support is required which ensures there are no production delays. This is achievable 99% of the time.

As well as the 2m roof strength, 2-4m, 4-6m, 6-8m and 8-10m roof strength contours are prepared over the development panel prior to the commencement of mining. The upper horizon contours allow proper assessment of the minimum required length of long tendon support, saving both time and money. Fig. 4 shows an example of sequential roof strength horizons.

In addition, and also prior to development, a 12m roof section is prepared showing the variation in roof conditions. Borehole spacing is a minimum of 200m along the gateroads but additional infill drilling is frequently carried out to reduce the spacing to 100m, depending on the whether conditions are considered as critical. These sections are based on the roof units in the geomechanical database but are taken to the next level of detail, that is, sub units or local variations are included, together with any other features such as low strength or sheared horizons as indicated on individual borehole logs. Average roof strengths for the units are also included.

Fig. 5 shows an example of a maingate roof section. This plan is used to show the anchorage horizons, indicate where the roof unit contacts and discontinuities are, show how much immediate roof is weak and assist in the assessment of primary and secondary support patterns.

It is a requirement of the Strata Management Plan that the roof strength contours and the roof section diagrams are included in the panel folders and posted on the crib room bulletin boards for each development section. These plans, as well as many other geologic and roof support issues, are reviewed with the crews prior to the commencement of a new gateroad.
A full hazard plan is prepared for each longwall prior to extraction. In addition to the 2m roof strength contour and the 12m roof section along the panel, other geological information is also added as follows:

- a seam to surface section along the panel showing the change in depth of cover, the location of overlying seams and the position of strong or water bearing strata horizons
- a 0.5m roof strength contour to help assess the tendency for immediate roof flaking or slabbing
- all geological mapping information, contours of any any strong beds, and all telltale results.

It is a requirement of the Strata Management Plan that the hazard plan is included in the panel folders and posted on the crib room bulletin board for each longwall. This plan, as well as many other geological and roof support issues, is reviewed with the crews shortly after the transfer from the previous longwall.

**STRATA MANAGEMENT PLAN**

The three main elements of the strata management plan are:

- hazard plans / support rules
- responsibilities / monitoring / action response
- support testing / auditing

Hazard plans are developed for both development and longwall panels using the available geological and geotechnical information. This information is used to familiarise miners and deputies with upcoming conditions and design primary and secondary support rules for development. These plans as well as telltale results are referred to when a final maingate inspection/review is done to assess any additional secondary support requirements prior to longwall operations.

All employees of and visitors to the mine have responsibilities under the Strata Management Plan. Deputies (ERZ Controllers) in particular have a high level of responsibility. Dual height telltales are installed in all intersections and any other areas of anomalous roof conditions by the development crews at the face. These are monitored by the deputies daily and the readings taken to the surface for input into the database.

Four levels of alarm can be generated based on roof movement in either horizon (10mm, 20mm, 40mm and fall of ground). In addition, alarms can be generated by other forms of roof or rib deterioration, such as when encountering faults, having abnormal slabbing of the roof ahead of the miner, or guttering, cracking or flaking of the roof or rib outbye the miner during development, and on the longwall by slabbing of the roof, roof movement in the gateroads or significant spall of the face or gate ribs. All alarms are acted on immediately and then reviewed by the Strata Control Group once per week to ensure actions and monitoring has been completed.

As part of the program of monitoring, when alarms have been reached which result in long tendon support installation, a data sheet is attached to the rib which includes the date of telltale installation, dates of reaching the different alarm levels, date of long tendon support installation and the readings at the time, and the most recent readings. This allows on the spot evaluation of roof stability in an area where secondary support has already been installed.

In critical areas such as installation roads, takeoff areas where roof movement has exceeded telltale limits, or areas which have been grouted with OPC, three or four point GEL extensometers (electronic telltales) are installed to a minimum of 10m by a contractor. Readings are taken by deputies and brought to control for input and analysis, but in addition, a sheet of readings is installed on the rib and with a resolution of 0.1mm, stability can be quickly assessed underground.

Deputies are also responsible for first mapping of geological anomalies encountered at the face. Mapping information is recorded on sheets in crib rooms and taken to the surface. This prevents loss of information due to stonedusting and is used when final mapping is done and recorded on the mine plan.

The Underground Mine Manager, Longwall Coordinator, Development Coordinator, Geotechnical Engineer and Shift Coordinators also have responsibilities under the action response plans and make up the Strata Control
Group which reviews all aspects of the Strata Management Plan at the Strata Management meetings, which are held as a minimum on a monthly basis.

An equally important aspect of the strata management plan is roof support testing and auditing. Three complete audits have been developed to audit roof support at Crinum.

A primary support audit is carried out quarterly by the geotechnical engineer usually in conjunction with the roof bolt supplier. This involves a surface materials inspection checking storage, quantities, best before dates, condition and delivery of all materials used for primary support. Then an underground inspection is done at the face auditing the same things as on the surface as well as installations, insertion times, spin times, hold times, hole depths, hole diameters, encapsulation, tail lengths, bolter torques, bolt loads, patterns, and all other aspects involved in the installation of the primary support. Discussions are held with crews if any deficiencies are found or improvements can be made.

A secondary support audit is carried out quarterly by the geotechnical engineer usually in conjunction with the cable bolt supplier or installation contractor. It involves virtually the same audit modified to suit long tendon support installation.

A longwall support audit is carried out quarterly by the geotechnical engineer usually in conjunction with the longwall coordinator or the longwall production coordinator. This involves auditing the gate end conditions, use and status of Gun Set supports, the hydraulic delivery system including pumps in use and operating pressures, the maingate data including leg pressures and status of positive set, the longwall shield gauge readings and leg extensions and the tailgate support and conditions. This information is collated with the continuous monitoring of leg pressure results from the surface and presented to the longwall coordinator and longwall mechanical and electrical engineers.

Actions for improvement are generated from the audits and all audit results are reviewed with the crews.

CONCLUSIONS

The following conclusions can be drawn from the Crinum experience with regard to mining in weak and variable roof.

3. systematic evaluation of sonic logs can provide a general understanding of the variation in roof strength, can identify areas of concern and can provide the basis for delineation of roof support categories.

4. roof strength plans can enable comparisons to be made of proposed development with previous experience and hence improve predictability of mining conditions ahead of mining and improve planning and budgeting.

5. hazard plans with roof strength sections are an important control element for determining roof support design and monitoring requirements and for making changes on the run during development and extraction.

6. on site ownership of the strata management process is essential to manage roof monitoring and to augment changes based on cumulative experience.

ACKNOWLEDGEMENTS

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CRINUM LONGWALLS 7 TO 11

GEOMECHANICAL ROOF AND FLOOR UNITS

0.6–2.9 m = Unit Thickness Range (metres)

Figure 2
2002 Coal Operators’ Conference

Tribute to Dr Ripu Daman Lama

Figure 4 - Sequential Roof Horizons - Interpreted Roof Strength

Figure 5 - Maingate Roof Section

Legend

UNIT 2 - Geomechanical roof unit two

W - Weak potential parting horizon

H - Height above coal seam

O - Average estimated strength for unit (MPa)

*Stratone 5509 is located towards T08 in 2T off

Unit Descriptions:

UNIT 2A - PERITONITE - Usually 1.0 to 4.0m, maximum 7.8m, varies from 7 to 15 MPa.

UNIT 2B - PERITONITE - Moderately strong to very thick bedded \textit{PERITONITE} and associated lithologies. Varies from 7 to 15 MPa.

UNIT 2B - \textit{PERITONITE} - Usually 1.0 to 4.0m, maximum 7.8m, varies from 7 to 15 MPa.

UNIT 2B - Moderately strong to very thick bedded \textit{PERITONITE} with associated lithologies. Varies from 7 to 15 MPa.

UNIT 3 - Weak to extremely weak \textit{LIMESTONE} and associated lithologies. Varies from 0 to 5 MPa.

UNIT 3B - \textit{LIMESTONE} - Usually 1.0 to 4.0m, maximum 7.8m, varies from 7 to 15 MPa.

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UNIT 3B - \textit{LIMESTONE} - Usually 1.0 to 4.0m, maximum 7.8m, varies from 7 to 15 MPa.
INTRODUCTION

Hawcroft Miller Swan (HMS) reviews approximately 120 mines per year, around the world. Of these approximately 60% mine coal and the remainder metal and other minerals. Also approximately 60% of these mine underground. From our experience it is apparent that the mines with a well-developed understanding of their risk profile and a systematic approach to the way they do business, enjoy better business outcomes in terms of safety, productivity, cost control, profitability and reputation. Others without systems in place, consume a phenomenal amount of time and energy dealing with day-to-day issues, without ever gaining effective control of the critical issues that drive their business.

This paper seeks to share HMS experiences in relation to risk management in Australia with particular focus on the underground coal mining industry of New South Wales and Queensland and geotechnical matters. It provides a brief overview of the legislation in both states and the use of risk assessment in relation to statutory compliance and as a business management tool. Comment on the benefits and limitations of risk assessments are given from the perspective of an organisation providing risk management services to the industry, or utilising industry-specific examples.

Over the past decade the Australian underground coal industry and the mining industry in general has experienced an increasing application of risk assessment processes as a proactive initiative for improving safety. Tougher economic, environmental and legal pressures, coercion from employee, government and social groups (including local trade unions and the ILO), the recognition that a contract of employment does not include acceptance of personal injury and an awakening to the principles of due diligence, have all fuelled this new perspective.

It is true that underground coal mining is inherently more hazardous than other industries, but this doesn’t mean we should simply accept it to be more dangerous. Accident investigation has demonstrated that in most cases sufficient information was present before an accident for someone to have intervened and prevented it from occurring.

In the philosophy of risk management, the fatalistic acceptance that accidents will happen has been replaced by the realisation that impending loss is in most cases predictable and therefore preventable.

Damage control and reactivity in Australia has been overtaken by active participation to gain understanding of the hazards before engineering changes, implementing systems and building competence to gain effective control by eliminating or effectively mitigating the risks. The process most widely used is risk assessment. Done properly, this process incorporates local knowledge and experience with external expertise in a forum where the principles of involvement and consensus are used to identify and prioritise hazards and to recommend appropriate means of controlling risk.

On average HMS facilitate one risk assessment every two weeks. Risk assessment topics and applications vary from the very simple to the subtly complex. Some organisations are quite experienced with the risk assessment process and embrace it for all the benefits it provides, while others are in denial, applying the process because of statutory compliance or corporate directive. The more experienced workplaces have become quite adept with the process and efficiently achieve significant benefits while others need on-going education and coaching to get it right then battle through as management and workplace cultures are reprogrammed away from the compliance-based mindset.
WHY DO RISK ASSESSMENTS

In general there are two main reasons why mines do risk assessments – for statutory compliance or because within the organisation there is a higher level of appreciation for the benefits of a risk management and a systematic approach to operating their business.

For those that embrace the risk management philosophy there is the acceptance that good safety performance, like good productivity are simply outcomes of a well-managed system.

From a world wide perspective Australia seems to be leading the pack when it comes to risk management and the utilisation of the risk assessment process for safety and whole of business considerations. Mines reviewed throughout mainland Europe and the United States were generally not using risk assessment as part of their business management strategy. Some US mine operators are not familiar with a generic risk matrix. South African mines have a greater appreciation of the risk assessment process. Having been introduced to the philosophy in 1996, they are a couple of years behind Australia and have confined its use to purely health and safety applications.

Larger international corporations such as Rio Tinto, BHP Billiton and Shell/ Anglo Coal have embraced risk management and incorporated the philosophy as a Corporate Standard and irrespective of the country, those sites are in general, well advanced in applying the process.

In NSW risk assessment was introduced approximately ten years ago and eagerly promoted by the Chief Inspector of Coal Mines. Since that time and subsequent to the Moura Inquiry, the Queensland coal industry followed suit. Both states moved to enshrine risk assessment in legislation in 1999. NSW and Queensland are equally well advanced in their application of risk management and in particular, risk assessments.

More recently the Western Australian Department of Mines required mandatory risk assessments for justifying areas wherever rock bolting is not adopted following significant strata-related failures and fatalities.

Despite Australia having the jump on the rest of the world in risk management there is by no means reason to become complacent in the sense of a job well done. There is still a long way to go and no excuse to not continue to be the world leader.

STATUTORY COMPLIANCE

The principles of risk management in Australian Mining Legislation are either by direct reference or through the imposition of Australian Standards, “accepted practice and due diligence”.


The NSW 1999 Regulations have introduced the principles of risk management over the existing framework of the CMRA 1982. The resultant NSW legislation is a combination of prescription and self-regulation, however in the ability to self regulate, mines are required to develop and implement systems for specified core risk areas, e.g. inrush, ventilation and emergency escape. The pre-existing Act of 1982 remains for now unchanged and the requirement for Managers to make Rules and Schemes continues.

The first reference to strata control in the CMRA 1982 is Section 37 (2) (c) (ii) which defines one of the functions of the Mine Manager as “ensure that the roof and sides of working places and roadways in the mine (other than roadways located in a part of the mine which is fenced off in pursuance of the regulations) are adequately supported where necessary for safety”.

1 Core risk areas are areas of operation of coal mines with generic hazards that have the potential to cause multiple fatalities.
Later in the Act Section 102 requires the Mine Manager as to make support rules, while Section 105 requires that those rules are provided to the District Inspector for Confirmation.

In the NSW Underground Regulations, Division 9 sets out in detail the Support Rules. Clause 48 prescribes in detail the process of developing, implementing and monitoring **Support Rules** and specifically includes the following matters:

- estimate the types of geological conditions likely to be encountered in roadway development
- assessment of the stability of roadways to be developed in geological conditions likely to be encountered
- development of support measures that will provide roadway stability in geological conditions likely to be encountered
- preparation of support plans that explain in full detail the means of roadway support required to be installed and prepared in a manner such that they may be readily understood by those required to install roadway support
- provision of safe, effective and systematic work methods for the installation, and subsequent removal where required, of roadway support (including support in connection with the carrying out of roof brushing operations)
- provision of adequate equipment and resources to effectively install or remove roadway support
- monitoring of the stability of roadways after formation and support installation
- training of employees, including: support design principles, support plan interpretation, placement and removal, understanding the need for and importance of the various support systems, recognition of indicators of change that may affect roadway stability
- recording of geological features that may affect roadway stability
- recording of roof failures that have the potential to cause injury to persons
- conducting of periodic compliance audits (not exceeding 12 months)
- reviewing of the application and effectiveness of the support rules at intervals not exceeding 12 months.

In a further requirement of Division 7 – Working Dimensions and Breakaway Rules of the underground Regulations the Manager must make Breakaway Rules “for fixing the maximum width of intersections, the manner of commencement of drivage of roadways, showing the sequence of forming the intersections and the manner of support”.

In Section 37 (2) (h) “the manager must ensure he is in possession of all available information relevant to the behaviour of strata surrounding the mine and its relationship to the safe working of the mine and all available information regarding disused excavations or workings in the vicinity of the mine”

In relation to the stability of workings, Clause 38 of NSW Underground Regulations and Section 139 of the Act prescribe the minimum permitted pillar sizes and location and size of barriers respectively.

Although the NSW legislation can be considered largely prescriptive in nature, In Clause 6 of the general Regulations, owners are required to assess risks to health or safety regularly and ensure that identified risks are dealt with in the following order of priority:

(a) eliminate the risk,
(b) control the risk at its source,
(c) minimise the risk by means that include the design of safe work systems,
(d) in so far as the risk remains, provide for the use of personal protective equipment, having regard to what is reasonable, practicable and feasible, and to good practice and the exercise of due diligence.”

Clause 6 locks the owner/manager into adopting a risk-based assessment of operations in general (including strata control). Indeed many NSW operators have already adopted a risk management approach to strata control and routinely conduct risk assessments for new panel layouts as part of Section 138 Applications. Alternatively a risk assessment may be required by an inspector on submission of Support Rules in accordance with Section 105 of the Act for confirmation.

Many mines have developed support rules in the form of a strata control management system and incorporated matters in addition to the prescribed matters of the Regulations. Furthermore compliance with Section 4.2.2 of AS 4804 - Australian Standard for Risk Management Systems requires that hazard identification is adopted in the development of safety management systems. Accepted processes are described in AS4360 and MDG1010.
In a similar manner Section 30 of the Queensland Act requires a risk management approach be adopted according to the following criteria:

- identify
- analyse
- assess risk
- avoid or remove unacceptable risk
- monitor levels of risk and the adverse consequences of retained residual risk
- investigate and analyse the causes of serious accidents and high potential incidents with a view to preventing their recurrence
- review the effectiveness of risk control measures, and take appropriate corrective and preventive action
- mitigate the potential adverse effects arising from residual risk

In general there are more direct references to adopt risk assessments in the Queensland Coal Mining 1999 legislation than the corresponding NSW legislation. The Act identifies strata control as a principal hazard, requiring a hazard management plan to be developed while Clause 63 of the CMS&HA 1999 requires principal hazard management plans to "identify, analyse and assess risk associated with principal hazards". Irrespective of this Section 4.2.2 of AS4804 requires that hazard identification is undertaken for the development of safety management systems in general.

Section 29 of the Queensland CMS&HA 1999 places the responsibility of reducing the risk to persons from coal mining operations to an "acceptable level" or within "acceptable limits" and "as low as reasonably achievable". This is similar to Clause 6 of the NSW General Regulations, requiring "reasonable, practicable and feasible, and to good practice and the exercise of due diligence" from operators.

Clause 317 of the Regulations specifically requires that a risk assessment must be carried out for second workings including consideration of the following matters:

- any surface features, artificial structures and water reserves that may create a hazard if disturbed by the workings
- any other workings located in close proximity above, below or adjacent to the proposed second workings, whether in the same or an adjacent mine
- known geology affecting the intended workings
- anticipated gas make
- pillar stability
- proposed method and sequence of coal extraction
- proposed methods for the following
  - strata control and support

Clause 318 required that the system of extraction be "based on" the risk assessment identified in Clause 317 and Clause 320 requires a further risk assessment for subsequent changes to an extraction system.

Clause 323 (2) requires the underground mine manager to ensure suitable strata support methods are designed and implemented for the working place. Unlike the NSW legislation the Queensland legislation refrains from defining the process of design, implementation and monitoring of support systems.

In relation to the design and layout of panels and pillars, Section 138 CMRA 1982 requires that mines obtain Ministerial approval for mining by methods other than bord and pillar (Pillar extraction, longwall or mini wall mining, auger mining, punch mining, single entries). The NSW Department of Mineral Resources has published guidance material for gaining a Section 138 Approval (MDG 1001 and MDG 1005 for extraction systems and draft guide lines MDG 1011 for single entry drives). Reference to the assessment of risk is made in the guidance material (and draft guidance material) and a Section 138 Approval is usually contingent upon a risk assessment and a geotechnical assessment having been done as a minimum.

Clause 321 of the Queensland Regulations simply requires that mines ensure the stability of mine workings in relation to pillar strength and stability and strata support requirements, while Clause 320 makes reference to dangerous mine subsidence.
It is apparent that whilst the legislation pushes mines to adopt a risk management approach to strata control, a level of prescription has been retained as a safety net to maintain effective regulation.

OTHER REASONS

Despite the statutory requirements of the legislation, there are many other good reasons for conducting risk assessments.

Risk assessments consider broader issues to a greater level of detail commensurate with the level of risk. Time is not wasted addressing irrelevant matters and important issues are not skipped over to simply comply with legislation.

It is not possible to prescribe in legislation and associated codes, standards and guidelines all the permutations to cover all mines, their systems and mining conditions, let alone the other idiosyncrasies of each mine that are relevant at the particular time. Risk assessment is a situational analysis whereby the subject can be assessed in the context of its environment. Prescription does not incorporate the site specific knowledge and issues.

Risk assessment can be scoped broadly or focused to suite the need. Site personnel utilise the experience and expertise to develop appropriate controls.

Risk assessments provide a forum in which to:

- identify information gaps. When undertaken in the early stages of design, it has the ability to identify whether more data or research is required in order to make properly informed decisions
- determine if design assumptions are relevant and appropriate for the application
- assess the impact on associated and related systems
- allow balanced assessment and prioritisation of multiple and diverse risks. Bland compliance with prescription limits the opportunity to manage the total system
- provides opportunity to capture the full range of hazards relevant to the particular situation in terms of business interruption, material damage and reputation - not only safety
- flush out implementation issues

The process of identifying and quantifying and prioritising risk directs focus on the important issues so the mine does not waste time pursuing pet topics or matters that prescription might otherwise set priority upon without it being relevant at the particular mine. For example, outburst may not be of relevance at some mines but may be of critical importance at others.

There are many examples of the limitations of prescription through Regulation and guidelines. One such example in NSW is the failure of the Wardell guidelines to address all relevant matters in panel and pillar design. Mining layouts designed in accordance with the specified guidelines did not take into account the behaviour of pillars with claystone in the surrounding strata and resultant mine subsidence exceeded the design amount.

Where specific design guidelines are provided, there is a real danger that they can be misinterpreted or applied incorrectly to geological/ mining conditions for which they were not intended. In some instances compliance with specified guidelines has been interpreted as having followed a satisfactory design process. Prescription can perpetuate an attitude where compliance is taken as sufficient without any further investigation. A full risk assessment however, scoped to incorporate safety, business interruption, material damage and reputation will lead operators to applying a level of rigour commensurate with acceptable risk exposure and utilise appropriate design principles including proper engineering practice.

WHAT IS A RISK ASSESSMENT

Risk assessment is a structured situational analysis of a particular interaction between equipment/ people/ environment to identify, quantify and prioritise risk against a predetermined standard and identify appropriate means of control.

There is no such thing as a “one size fits all” or a superior risk assessment technique. The Safety Systems Society Handbook (1993) covers over seventy risk assessment techniques while the NSW Department of Mineral Resources (DMR) publication MDG 1010 (DMR, 1997) provides examples of the most common techniques for
the mining industry. Each has its own strengths and limitations. The question is which technique is most useful in assessing strata control.

**RISK ASSESSMENT TECHNIQUES**

This section briefly touches on some of the risk assessment techniques and their applicability to strata control in underground coal mines.

The Workplace Risk Assessment and Control (WRAC) technique is commonly used throughout the industry for identifying potential operational production and maintenance loss. It is the technique most commonly used. WRAC has the capacity to focus on a variety of aspects including, support type, installation method, inspection regimes and other activities. It can be applied in areas of the mine or at particular times of activity.

WRAC is most effective when it is scoped with appropriate detail, including clear objectives and the boundaries of the system have been defined. Long and short term stability can also be assessed if known, depending on the mining process used.

Fault Tree Analysis (FTA) is used when it is necessary to show the logical structure of major undesired events by graphical means. The outcome is highly visual for ease of understanding failure mechanisms. Fault Tree Analysis may be extended to include a quantitative analysis, whereby probabilities of separate events are analysed to give an overall probability of a particular major loss event. Data is often not available nor this level of analysis necessary for mining applications. FTA can be very time consuming and produce few benefits beyond WRAC.

Both WRAC and FTA techniques however are good for identifying multiple failures. For example strata failure due to one or all of the following would be identified:

- Incorrect type of support used for conditions
- Incorrect assessment of environmental conditions
- Failure or inadequacy of inspections
- Incorrect installation
- Support equipment failure
- Insufficient skills/competencies

Human behaviour or error and equipment component failure studies need to be closely scoped so they do not become unwieldy. Behavioural risk assessments can be difficult as they are dependent upon development of templates for perfect behaviour. This is particularly difficult if multiple tasks are involved. They also assume that the system, environment and machines are perfect, when in fact this is often not the case.

When dealing with machinery used for the installation of supports or working within a defined support regime the Machinery Hazard Identification method may be appropriate.

Failure Mode and Effect Analysis (FMEA) is appropriate for focusing on hardware failures, and quantifying risk if probability of failure can be defined. This technique is good for analysing equipment designs and modifications and pre-commissioning assessment, for example development of a mobile bolter.

In general, qualitative risk assessments are adequate for most circumstances in the mining industry and quantitative risk assessment may not even be possible because of an absence of reliable data.

An example of the commonly used WRAC (qualitative) risk assessment technique follows.
Examples of qualitative descriptors for consequence are provided below in Table 1.

<table>
<thead>
<tr>
<th>Rating</th>
<th>Injury</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 – Catastrophic</td>
<td>Multiple Fatalities</td>
</tr>
<tr>
<td>2 – Major</td>
<td>Fatality</td>
</tr>
<tr>
<td>3 – Moderate</td>
<td>Serious Bodily Injury</td>
</tr>
<tr>
<td>4 – Minor</td>
<td>Lost Time Injury</td>
</tr>
<tr>
<td>5 – Insignificant</td>
<td>First Aid</td>
</tr>
</tbody>
</table>

A holistic view of mining activity in the context of the business can be taken by including additional consequence descriptors such as business interruption, material damage, environmental impact and reputation. Many Australian mines now include these considerations in risk assessments, having recognised the benefit of taking a holistic view of the mine as a business. In practice the most severe risk, in terms of safety, business interruption, material damage, environmental impact or reputation, is adopted.

This approach recognises that safety, like productivity and profitability are simply outcomes of a well-managed system.

In determining the consequence, both instantaneous and cumulative loss should be considered. Subsidence, like health related considerations should be considered from an exposure and repeat activity viewpoint to capture long term consequences.

Examples of qualitative descriptors for likelihood which are self explanatory are provided below in Table 2. They usually result in some conjecture and it is often necessary to remind risk assessment teams to risk rank in absence of soft barriers.

<table>
<thead>
<tr>
<th>A - Certain</th>
<th>B - Probable</th>
<th>C - Possible</th>
<th>D - Remote</th>
<th>E - Improbable</th>
</tr>
</thead>
<tbody>
<tr>
<td>Will occur</td>
<td>Likely to occur</td>
<td>Could occur</td>
<td>Unlikely to occur</td>
<td>Practically impossible</td>
</tr>
</tbody>
</table>

Risk is determined by considering the consequence and likelihood in relation to a matrix such as that shown below in Table 3.

<table>
<thead>
<tr>
<th>A – Certain</th>
<th>B – Probable</th>
<th>C - Possible</th>
<th>D – Remote</th>
<th>E - Improbable</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 – Catastrophic</td>
<td>1 (E)</td>
<td>3 (E)</td>
<td>5 (H)</td>
<td>7 (H)</td>
</tr>
<tr>
<td>2 – Major</td>
<td>2 (E)</td>
<td>4 (E)</td>
<td>8 (H)</td>
<td>12 (S)</td>
</tr>
<tr>
<td>3 – Moderate</td>
<td>6 (H)</td>
<td>9 (H)</td>
<td>13 (S)</td>
<td>17 (M)</td>
</tr>
<tr>
<td>4 – Minor</td>
<td>10 (S)</td>
<td>14 (S)</td>
<td>18 (M)</td>
<td>21 (L)</td>
</tr>
<tr>
<td>5 – Insignificant</td>
<td>15 (S)</td>
<td>19 (M)</td>
<td>22 (L)</td>
<td>24 (L)</td>
</tr>
</tbody>
</table>

The matrix shown in Table 3 is a 5 x 5 safety risk matrix. The number in each of the coloured cells indicates the risk index for particular consequences and likelihood. One (1) is the most severe risk raking and twenty-five (25) the least severe. Other risk matrices are available. Some don’t distinguish between single and multiple fatalities and others are presented as 3 x 3 matrices. Most mining operations have found a risk assessment technique they feel comfortable with, and have developed their own qualitative risk matrices based on AS/NZS4360 and MDG.
1010. It is worthwhile noting that it is impossible to achieve a risk rank lower than 1 E (11) for catastrophic consequence unless the hazard is eliminated completely.

The most important task in risk assessment is to rate the likelihood and consequences, calculate the risk rank of all related events that could result in loss and arrange those in order of priority to gain an understanding of the most important areas.

Items can take the form of an activity, an object, a condition or a combination of these. People must clearly describe what it is being ranked under the chosen heading in the table e.g. hazard or loss scenario, and be consistent in the language used as this will help for later sorting.

While the risk assessment method chosen is important for capturing hazards and reflecting their seriousness, it is the outcomes of the risk assessment that is most important. People should not take the risk rank as definitive nor waste time haggling over the risk rank. Risk ranking it is not an exact quantification of risk but a means of prioritising actions and allocating resources to control hazards.

Exposure data can be added to the equation, but separate reference is usually not warranted, as it can be reflected in both likelihood and consequence ratings.

**RISK ASSESSMENT TEAMS**

Risk assessments can be done by an individual, however to maximise the breadth and depth of analysis, team work is best. Individual assessments are best reserved for less complex tasks such as local risk control process, job safety analysis (JSA) or hazard awareness. Group analysis is preferred for larger topics to take advantage of the collective knowledge, experience and technical expertise of a group to provide a more comprehensive coverage of the topic.

The benefit of utilising groups to assess risk include:

- A cross section of people who are involved with planning, implementing, operating, monitoring and reviewing the process are included, leading to more comprehensive coverage from all aspects or perspectives, highlighting issues of which management or technical experts may not be aware.
- Technical people become aware of implementation and operational issues they might not otherwise have considered in the design.
- Solutions will be practical for those required to implement them providing at least a chance they will be implemented as designed.
- Management can qualify the economic and operational feasibility of suggested solutions.
- Suggested controls are likely to consider all competency, skills and literacy.

Risk assessments are usually not the forum for assessing geotechnical engineering design unless the participants are technically qualified. Rather most risk assessments are used to check whether the correct design parameters have been used for the proposed design and to consider implementation of the design in the mine environment.

**HMS OBSERVATIONS**

Despite the lack of direct reference by legislation requiring risk assessment for each type of system, mines have adopted risk assessments as their preferred vehicle for incorporating consultation in developing and reviewing safety management systems. In NSW strata is usually dealt with during the broad-brush risk assessment. The depth to which sites drill down is dependent on the individual site.

Furthermore an increasing number of mines are taking the opportunity to consider business interruption, material damage, environmental impact and reputation in addition to health and safety. Some have taken the process to the next level to identify and evaluate up-side opportunities to determine appropriate improvement strategies.

Some mines do risk assessments for the sake of compliance and are inclined to allocate tight and often unrealistic timeframes, grab personnel on rehabilitation to make-up numbers, exclude union participation and use strong management direction to speed the process along. It is important to let the process unravel, identify the hazards and potential loss, prioritise risk and specify controls and not lose sight of the objective of the exercise.
Risk assessments are often disrupted by people with operational responsibilities attending to mine needs, particularly at shift change. Some operations hold risk assessments off site or at times when other operational personnel are available to avoid disruptions.

Risk assessment applications vary from the very simple to complex. Whilst strata issues may be dealt with at some sites during wide broad brush risk assessments, other more detailed or very specific assessments deal with particular mining systems or equipment. In some cases the assessment will consider a detailed design, including the final draft of a geotechnical design, whilst others may take advantage of the risk assessment group to review and identify hazards and controls on a range of panel design options. Unless the risk assessment team is comprised of a team of expert personnel a risk assessment is not usually the appropriate forum to critique the geotechnical design. However it is appropriate to identify how design assumptions can be compromised and to identify additional or changed factors the geotechnical design may have neglected to address, e.g. wide roadways are required to operate large mining machines.

It is also important to establish that the risk assessment team understand the process before commencement. There is often need for a quick training session before commencement, although pre-assessment training is becoming less common.

The site acceptance of the risk assessment process is variable. Some sites embrace the process as a valuable tool that affords the opportunity to have a real impact on safety and business performance. Other sites see the system as a matter of process or a waste of time and participate to fulfil some statutory criteria and take a passing interest in the overall process. Generally the more familiarity the site with the process the more benefit is derived and the more efficient it becomes.

There is a wide variation of risk tolerance across the industry. Risks that are acceptable to some mines are not to others. Risk tolerance at particular sites can influence the outcome of the risk assessment and hence prevent hazards being identified or appropriate controls being adopted.

Risk tolerance levels can develop in the culture as perceptions, built by a history of what has worked in the past, and accepted practice accumulate in the collective memory of the organisation. Current operational constraints or perception that risks are necessary in the interest of preserving jobs or cutting costs also have an effect. It could also be simply a lack of risk management philosophy by management. An added danger is that if the facilitator is not aware of a high level of risk tolerance the risk assessment teams could down rate risk or gloss-over important hazards. Furthermore if the facilitator is a part of the same risk-tolerant culture the risk team may never be challenged and high or extreme risks go unassessed. Closed-shop organisations may not be aware of the folly of the things they do before it is too late.

It is important that all people in the risk assessment team reach consensus as the assessment proceeds. Repeated interjection by overzealous people, with pre-set agendas or pre-determined outcomes can, if not controlled, result in short cutting the process or extent of analysis. It is important that the facilitator ensures that all team members get equal say before reaching consensus.

For tightly constrained risk assessments it is imperative that the scope reflects the timeframe of the assessment and that all assumptions and limitations are properly defined, understood and carefully recorded. This helps to circumvent the problem of people becoming sidetracked by incidental issues.

Most clients prefer to conclude the group session before finalising the action plan. For small risk assessments this may not be a problem because the senior person in the team can ensure the actions are followed through. Similarly for complex risk assessments it would be grossly inefficient to occupy the time of personnel allocating tasks for hundreds of actions. Diligence however is required in follow-up to ensure actions are assigned to appropriate personnel and that those actions are actually completed.

HMS conducted a review of a number of risk assessments covering the following strata-related topics:

- Pillar extraction – Wongawilli, Pillar stripping, breaker line supports
- Longwall extraction
- Cut/flip mining
- Massive strata
- Section 138 application
- Mobile bolters
- Longwall mining through disused shaft
• Drift excavation
• In close proximity to overlying workings
• Thin seam operations

TYPES OF STRATA HAZARDS

The review identified the following distribution by energy type is shown in Table 4

Table 4 Risk Data by Energy Type

<table>
<thead>
<tr>
<th>Energy Type</th>
<th>Proportion</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gravitational</td>
<td>82.4%</td>
</tr>
<tr>
<td>Mechanical</td>
<td>7.2%</td>
</tr>
<tr>
<td>Chemical</td>
<td>6.4%</td>
</tr>
<tr>
<td>Thermal</td>
<td>2.4%</td>
</tr>
<tr>
<td>Pressure</td>
<td>1.6%</td>
</tr>
</tbody>
</table>

Whilst the vast majority of hazards are concerned with falling roof material a significant number of hazards relate to the operation of equipment associated with roof control or a mining system being assessed for strata hazards.

Some examples although not exhaustive, of the types of hazards identified by the strata-related risk assessment review are:

• rib failure
• operator position under unsupported roof
• recover C/M from unsupported roof areas
• ventilation disruption by choked goaf
• windblast and gas emissions from goaf
• undetected change in strata conditions, adverse geological conditions
• roof failure in work areas adjacent to goaf
• fall of roof between supports
• BLS over-ridden by goaf fall
• Person crushed by BLS or continuous miner
• failure of supported ground, bed separation, anchor failure, poor installation or setting
• shape & size of pillars or roadways, roadway alignment
• pillar failure or collapse - punching, weak interfaces, width to height, grade
• load transfer, abutment stress, periodic weighting, face slabbing
• orientation to structures, joint sets, cleat
• support setting or installation hazards

The diversity of the above list demonstrates the ability of the risk assessment process to flush out a broad array of hazards. More importantly the list demonstrates how comprehensive a prescribed list of considerations might be required in order to identify specific hazards for all mines.

Considering the examples listed above it is apparent that no level of prescription would identify relevant hazards for each site. Two examples are set out below
Example 1.

Consider the operation of a large remote control continuous miner in cut flit development under strong roof, with good bridging properties. Two hazards at this operation could have been ranked as follows:

<table>
<thead>
<tr>
<th>No</th>
<th>Area</th>
<th>No.</th>
<th>Activities/Process</th>
<th>No.</th>
<th>Hazard</th>
<th>C</th>
<th>L</th>
<th>R</th>
<th>Existing Controls</th>
<th>Additional Controls</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Strata Control</td>
<td>1.1</td>
<td>Remote control cutting</td>
<td>1.1.2</td>
<td>Crushed by RC C/M in 5.5 m wide roadway</td>
<td>I</td>
<td>C</td>
<td>4</td>
<td>• Safe Stand Areas defined</td>
<td>• Drive wider roadways for safe clearance</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>• Operator competence</td>
<td>• Redesign support system for wider roadway.</td>
</tr>
<tr>
<td>1</td>
<td>Strata Control</td>
<td>1.1</td>
<td>Remote control cutting</td>
<td>1.1.1</td>
<td>Failure of supported Roof in 6.5 m wide roadway</td>
<td>I</td>
<td>D</td>
<td>7</td>
<td>• Support Rules</td>
<td>• Design Support System for the place</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>• Inspections</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>• Operator competence</td>
<td></td>
</tr>
</tbody>
</table>

In Example 1 the risk assessment has identified that the risk of the remote control continuous miner crushing the operator is greater than that of being injured by a fall of supported roof. In mitigating risk, the risk assessment team must deal with risks in order of severity – highest to least. The team could have decided that an appropriate control to reduce the risk of crush injury from a continuous miner is to drive roadways wider to provide more clearance. To make this recommendation the team must make the fundamental assumption that an alternative roof support system can be designed for the wider roadway without increasing the risk to personnel by a fall of supported roof.

The benefit of using risk assessment processes to distil priorities from an array of hazards is demonstrated by this example. In the first instance the absolute necessity of protection personnel from falling roof material is the most basic of mining practice, hence the provision of support. Secondly the C/M operator has highlighted the possible danger of working in the vicinity of the RC C/M and offered a possible solution. If geologists and geotechnical engineers are present they have the opportunity to comment on the concept of driving wider roadways and to give an initial opinion as to whether effective redesign of support appears feasible. In addition, bolter operators can assess the proposal with a knowledge of the operational capabilities of the mobile bolter. Management can have input from the perspective of the impact on factors such as control of widths and cost of support. Fundamentally the process provides a balanced approach and detailed assessment on the solution of seemingly conflicting controls.

Having considered the proposal it is important to ensure the wording of the proposed control is consistent with the intent of the workshop. In this case all indications at the time of the workshop are that the concept is feasible, however the final width and support system design are as yet unresolved. Despite the column heading being changed from “Additional Controls” to “Potential Controls” the entry may be better stated as “Assess the feasibility of wider roadways to provide more clearance”. The final decision will be based on a proper geotechnical assessment and if in NSW gaining an exemption from the Department of Mineral Resources from the relevant part of the Regulations.

It is important to clearly record the actual intent of the risk team in the potential controls to avoid confusion during implementation or reviews.
Example 2.

Reconsider an alternative situation where a remote control continuous miner is driving longwall gate roads in weak roof.

<table>
<thead>
<tr>
<th>No</th>
<th>Area</th>
<th>No.</th>
<th>Activities/Process</th>
<th>No.</th>
<th>Hazard</th>
<th>C</th>
<th>L</th>
<th>R</th>
<th>Existing Controls</th>
<th>Potential Controls</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Strata Control</td>
<td>1.1</td>
<td>Longwall retreat</td>
<td>1.1.1</td>
<td>Failure of 5.5 m wide supported Roof</td>
<td>C</td>
<td>4</td>
<td></td>
<td>• Reduce roadway width</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>• Install secondary support before longwall retreat</td>
<td></td>
</tr>
<tr>
<td>1</td>
<td>Strata Control</td>
<td>1.1</td>
<td>Gate road</td>
<td>1.1.2</td>
<td>Crushed by C/M in 5.5 m wide roadway</td>
<td>D</td>
<td>7</td>
<td></td>
<td>• Install single pass C/M</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>development</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

In Example 2 the team identified roof fall on the Long wall retreat as the highest risk, however the mining conditions and mining method have led to a completely different outcome, substituting a standard C/M for a single pass, wide-head machine. The risk of crush in a narrower roadway is mitigated by the wide head configuration, limiting the ability of the C/M to slew during cutting.

Although the above examples are obvious they do illustrate the benefit of conducting site-specific risk assessments to sort out where the priorities lie. Again it is apparent that prescription in legislation is not equipped to determine the relative importance or severity of hazards for individual sites.

MAXIMUM REASONABLE CONSEQUENCE OF STRATA HAZARDS

From the HMS analysis of strata hazards it is apparent that the majority of hazards (82.4%) are objects falling. In an underground environment, unless the objects are small they have the ability to inflict serious injury or result in death.

Wherever the hazard relates to unsupported roof, the injury can only be a fatality because no controls are in place to regulate the size of material. Likewise in some instances, high ribs for example, the consequence must be serious injury to fatality.

The consequence of falling material can be reduced by installing hard barriers - support. By using support the size of material can be controlled somewhat so the hazard changes from “fall of unsupported roof” to “fall of roof between supports”. If for example 100 x 100 mm mesh is installed than the object size can be controlled to less than the mesh size, consequently the maximum reasonable consequence may be reduced to a lost time injury. Considering the application of soft barriers to a similar example eg personnel operating remote control continuous miners are prohibited by the Regulations and Support Rules from going under unsupported roof without setting some type of support. If the operator considers that his knowledge and experience allow him to make an accurate assessment of the roof integrity, he could decide to disregard the rule and duck out to reset the miner. In this example the soft barrier is quickly rendered ineffective by factors that at that moment are beyond the control of the law or management. The maximum reasonable consequence in this case is fatality.

The preceding two examples consider different situations but the facilitator needs to be aware of the factors that influence decisions people make and risk when conducting the risk assessment to ensure that the group does not disregard or underestimate consequence on these types of issues. The hazards are situational and the group needs to give careful consideration eg the size of the falling object before determining maximum reasonable consequence. For roof fall hazards a useful means of grading hazards may be for the group to develop a guide such as the example in Table 5.
**Table 5. – Example - Object size/ Injury Guide**

<table>
<thead>
<tr>
<th>Mass of Falling Object</th>
<th>Maximum Reasonable Consequence</th>
</tr>
</thead>
<tbody>
<tr>
<td>Less than 5 Kg</td>
<td>lost time Injury</td>
</tr>
<tr>
<td>5 Kg to 10 Kg</td>
<td>serious Injury/permanent disability</td>
</tr>
<tr>
<td>More than 10 Kg</td>
<td>Injury resulting in loss of life</td>
</tr>
</tbody>
</table>

The Mass of Falling Objects selected in Table 5 have been arbitrarily selected for the purpose of the example and could vary, depending upon the mining height, strata type, local geology and stress.

We therefore know that it would be inappropriate to rate the consequence of a person injured by a fall of unsupported roof as anything less than a single fatality because there is no control over the size or quantity of material falling.

In a similar manner the maximum reasonable consequence of rib fall injuries is related to the size of material that falls.

**LIKELIHOOD**

For qualitative risk assessments the terminology likelihood is preferable to probability or frequency. As previously stated there is generally an absence of adequate data for statistical assessment of probability. Furthermore people who are not familiar with probabilities will revert to qualitative descriptors to evaluate likelihood in their own minds.

Likelihood relates to the likelihood of the event. For example when assessing the hazard of roof falling on a person in a particular part of the mine, the likelihood is of a person being injured by a fall of roof and not the likelihood of the consequence of the incident.

Factors that may affect the likelihood include:

- Type of strata and support used.
- Frequency of exposure to the hazard eg. exposure to bad roof on a travelling road compared to bad roof in a waste areas.
- Location of personnel in relation to the hazard eg personnel operating equipment adjacent to the goaf compared to personnel grading an effectively supported transport road.
- Prevailing conditions in the area eg. roof, rib, goaf pressure, structures.
- Variability of the environment and presence of clear indicators eg. identification of orthogonal joint sets in a pillar split.

Considering the risk matrix, it is apparent that one cannot reduce the risk of a catastrophic consequence hazard to any less than 1 E (11) by making the event as “Practically Impossible”. The only way to reduce the risk of a catastrophic consequence event further is to completely eliminate the hazard itself.

**PRIORITISING ACCORDING TO RISK**

The reason for prioritising according to risk is to enable allocation of resources for the development of appropriate controls to maximise early benefit.

In accordance with AS/NZS 4360 (1999), risks should be “assessed in the context of existing controls”. This causes considerable confusion for some operations. It has been found as a general rule, risks should be assessed with hard barriers in place and assuming soft barriers are not in place or are ineffective. If we ignored hard barriers that are functioning effectively, we would practically be assessing the risks of mining in the last century,
and vastly over-estimating the risks. On the other hand, if we have not properly assessed the adequacy of soft barriers prior to the assessment, we may have the false belief that we are protected when we are not in reality, therefore under-estimating the risks. While competency is something we can measure and prove (or otherwise), other “soft barriers” such as adherence to procedures and supervision can be variable, and are certainly not foolproof controls. Soft barriers require constant monitoring, and a great deal of hard work to remain effective controls. Human factors are discussed in more detail later in the paper.

Therefore while some hazards may appear to be ranked a little high considering the efforts an operation is expending in controlling it, and a good track record may currently exist, this method serves to reinforce the importance of constant vigilance in this area.

**RISK CONTROLS - ELIMINATE OR MITIGATE RISK**

The hierarchy of controls reflects a grading from that is most effective to the least effective. *Table 6* illustrates the types of strata controls commonly identified in relation to the recognised hierarchy of controls for minimising risk.

<table>
<thead>
<tr>
<th>Hierarchy of Controls</th>
<th>Strata Control Examples</th>
</tr>
</thead>
<tbody>
<tr>
<td>1. Eliminate the hazard</td>
<td>Redesign of panel width to prevent full goaf caving and eliminate windblast hazard</td>
</tr>
<tr>
<td>2. Substitute alternative process/ equipment</td>
<td>Fixed cable handling device attached to continuous miner to replace cable hand</td>
</tr>
<tr>
<td>3. Engineering/ Isolation, including guards and barriers</td>
<td>Breaker Line Supports replace breaking off timber in Wongawilli extraction</td>
</tr>
<tr>
<td>4. Procedures/ administration/ training</td>
<td>Pillar extraction operating procedures</td>
</tr>
<tr>
<td>5. Personal Protective Equipment</td>
<td>Hard hat</td>
</tr>
</tbody>
</table>

Items 1 to 3 in Table 6 can in general be considered hard barriers in that they are designed to physically separate personnel from the hazard. Item 3, isolation and guards, can also be considered in part soft barriers because they are reliant on people to ensure they are properly in place before they can provide the desired protection. Controls of this nature are usually accompanied by procedural type controls. Items 4 and 5 are soft controls because they rely on diligence to be effective.

Personal protective equipment is often the last resort for providing protection from hazards but is often used in conjunction with other higher-level controls.

Mines should take care when deciding upon the level of control that choosing procedural over the seemingly more expensive high-level controls could well result in higher overall costs. The following example demonstrates this principle:

**Example 3. – Rib Control**

Workings oriented sub-parallel to the major cleat in a bord and pillar operation were experiencing increasing rib spall with increasing depth of cover. Slabs up to 20 m long and 1 m thick were detaching in panels being developed for extraction. There was a cultural resistance to supporting ribs and the mine was financially constrained.

Installation of rib dowels would have placed additional financial strain on the business and the allocation of limited resources to rib bolting all headings would impinge on productivity.

The solution derived was to change the orientation of subsequent panels (a level 1 control) and adopt a rib support program based on assessment of ribs (a level 4 control). Rib dowels installed are a level 3 control. The rib support program was sustainable, resulting in mainly only pillar corners requiring support, without affecting productivity.
Mines are now realising the resources required for maintenance of work procedures. Most operations have at least one person on site, dedicated full time to managing systems and procedure documentation. In addition as well as the training costs there is the on-going commitment to involving personnel and outside experts in reviewing and updating. It makes good business sense to eliminate hazards at their source once and for all through the application of higher-level controls and to minimise the use of procedural controls.

LIMITATIONS OF THE RISK ASSESSMENT

Despite the increased popularity of risk assessments, there are areas of concern that mines and facilitators need to be aware of in their use.

The purpose of risk assessments is not to provide excuses for not being diligent in the manner in which risks are managed. Invariably a risk assessment will lead to further work, research, data collection, training or analysis or design work, but focused to reduce risk exposure in the subject area. They should never be taken as a definitive solution in isolation but an action plan based on valid assessment of the risks.

Additional limitations of risk assessments include:

- Risk Team disbanded or partially disbanded before the risk assessment controls are finalised. This applies particularly operational staff at shift change.
- Inappropriate selection of team members such that the required experience and expertise is not covered eg. topping up numbers from people on rehabilitation.
- Participants who attempt to direct the process towards preconceived outcomes.
- Failure of the group to reach consensus in determining risk due to pre-set agendas – usually management or rushing to complete.
- Poor or inadequate definition of the hazard or loss scenario.
- Tendency to underestimate consequence and cultural risk tolerance. In-house facilitators could be part of a risk tolerant culture and fail to challenge the team to consider risk from an unbiased viewpoint
- Assessment with soft barriers effectively in place can lead to false assessment of the actual risk
- Unqualified transposing of results from one risk assessment site to another.
- Ambiguous or incorrect wording of potential controls.
- Failure to assign people to risk assessment outcomes or actions and failure of management to follow-up to ensure they are done.
- Failure to implement the agreed controls.
- Failure to review the risk assessment at some future time to confirm that the controls are effectively implemented.
- Disinterested participants don’t exercise diligence during the process. Everything ends up a 3 C (13).
- Over-complication or over-simplifying the risk assessment scope. The process must be time and cost effective yet comprehensive enough to ensure important issues are properly addressed.
- Assumptions and limitations in relation to the risk assessment topic are inadequately defined.

HUMAN ELEMENT

It is important to understand that behavioural factors play a significant role in the effectiveness of the risk controls that are nominated. For example, despite legislation, rules and procedures people still get injured doing exactly the things rules were designed to prevent. It is for this reason that soft barriers should be disregarded during risk assessments and the hierarchy of controls should be closely adopted in order to achieve effective risk reduction.

Pitzer (2000) investigates risky behaviour in relation to personnel working around remote control continuous miners. In his paper he found that mine workers identified the following face-area risk in order of severity:

1. Rib spall
2. Roof fall
3. Hit by shuttle car
4. Struck by miner
5. Hit by loose rock
There was a perceived lower risk from roof fall than rib spall because of an ingrained assumption that the roof was controlled by support whereas ribs are often not, and of equipment being competently controlled by an operator whereas there was no one in control of the ribs. (Pitzer, 2000).

Another interesting finding by Pitzer was that personnel routinely enter dangerous work areas such as unsupported roof areas despite legislation and rules to the contrary. Mineworkers stated that they would enter unsupported roof areas without using temporary support to rectify equipment breakdowns or quickly reset equipment. This was often done with the full knowledge of the supervisor. Risk assessment facilitators and risk management system development teams need to be aware of and understand the motivation of these foolish acts of risk taking behaviour that may be present in individuals and in some workplace cultures.

Whilst ever there is potential for risk taking behaviour to exist or be introduced mines need to place greater emphasis and insistence on hard barriers over soft controls.

Human error encompasses more than blatant disregard for rules. Mistakes, slips and lapses are also possible.

**MISTAKES**

Mistakes are self explanatory and result for a variety of reasons. The may occur because a person thought he was using the correct procedure, or had the correct information, and didn’t. These occur because people assume they have the correct information and neglect to verify the information is correct. It may result from being supplied incorrect procedures (wrong version) or data (out-of-date or wrong purpose).

**SLIPS & LAPSES**

Mining people can all relate to having at some time found themselves in a position that, had they had their wits about them we would not have happened eg. like being chased up the rib by a continuous miner boom as the operator backs it out of a lift. Even the more experienced personnel, in a place fully signposted and with which they are intimately familiar, can mistakenly find themselves in a less than desirable position.

**Example 4 – OOPs I didn’t realise I was under unsupported roof**

This example refers to the cut flit system of mining. A senior experienced mining official proceeded to walk past the “Last Bolt” sign in a recently cut place, looked up and realised he was well inbye the last row of bolts, and warning sign. This time he escaped injury!

Following this example the “Last Bolt” signs were upgraded to a portable barricade, consisting a steel tube that screws onto the last bolt. Two hinged arms are folded down to the horizontal position to form a barricade across the roadway at chest height. On the barricade was attached “No Road” signs and a blackboard for mining officials to record inspection details.

**DEVIANT VIOLATIONS**

Occasionally personnel believe they know a quicker or better way of doing things and disregard the rules. Thankfully instances of this type of violation appear to be less frequent than other human errors, however a person may choose to disregard a rule, believing he is working in the interest of the mine and that this level of experience allows him to make that judgement.

**CULTURAL VIOLATION**

Use of external resources such as OEMs, geotech consultants and the like should be encouraged, both as team members, and specialist facilitators and scribes. This will avoid bias and vested interests compromising the results of the risk assessment, and ensure the process and results are adequately documented.

**Example 5 – Working Under Unsupported Roof**

At a mine operating the cut and flit system of mining, personnel were told to “hog-out” and move the remote miner to perform the next cut and not to worry about cleaning the slack coal off the floor. This was some how
interpreted as management condoning the relaxation of safety standards. The slack coal left on the floor caused problems for the bolter.

Due to traction difficulties of the mobile bolter in slack coal, the bad habit developed of sending an Eimco into the unsupported place to clean the floor before bolting. The paradox of this practice is that prior to remote control miners the rule ingrained in the mine culture was that the miner would not proceed beyond the last line of supports. The Eimco operator relied entirely on one control, the roll bar-type canopy fitted to the Eimco that did not provide complete cover for the operator and had a very much lower rating than a manual miner canopy.

Although this behaviour had been identified by the risk assessment process as high risk and the principle of not going under unsupported roof had not changed, acceptance of this practice by front-line and middle management, allowed it to become ingrained in the culture of the work place.

It took an outsider to recognise the ridiculous acceptance of risk behaviour before it could change. The practice took some time of targeted education and clarification of business priorities to eliminate.

Example 6 – Re-stetting Gas Trips on Remote Control Continuous Miners

At a mine using remote control continuous miners in moderately gassy conditions, it was common for the miner power to be tripped by gas in excess of 2%. The rule in the mine had always been that “No person shall go under unsupported roof”.

The practice developed whereby people, including supervisors were going under unsupported roof to press the reset button, located on the miner, to enable the power to be reset. Apparently they made a judgement that by testing the roof and making a quick dash out and back that they were reducing their risk exposure - Dumb!!!

The reality of using a remote control continuous miner in extended cuts in a moderately gassy environment, equipped with an automatic methane monitoring system set to trip power at 2% was that at some time the machine would have to be recovered from unsupported roof areas. The inherent problem with the system was that the means for resetting ie. support the roof into the miner was not part of the normal system of work and not the natural choice for personnel. Add to this the willingness of supervisors to break the rules, led to this becoming accepted practice.

Effective elimination of this practice required a fundamental redesign of the equipment provided to operators with greater control over the mining process. Firstly the operator was provided with a numerical display of gas readings that was able to be read from afar. Secondly the automatic methane monitoring system was upgraded to provide continuous monitoring via battery back-up after the mains power was disconnected and the ability to be reset from a remote location – the DCB. In addition intensive retraining and compliance auditing was conducted to ensure the practice did not continue. The area was to be supported to repair other breakdowns.

Management are not always aware of the practices which apply and which are through by workers to be acceptable to management. Many examples can be recounted in risk assessments where management’s understanding of safe activities in the workplace have been shattered by comments to the contrary by face personnel. It is for this reason that it is essential to include operators in the risk assessment process and also reason for introducing outside perspectives on what is really an acceptable level of risk.

RISK MANAGEMENT SYSTEMS

NSW legislation is very focused on providing Rules eg. support, breakaway and brushing for face operations and specifying the manner in which the rules themselves are to be developed, implemented, monitored and reviewed. NSW does not require a hazard management system to administer these rules. Queensland legislation however requires the development of a principal hazard management plan for strata, based on risk assessment and is less prescriptive.

Despite the legal requirements, strata control management systems or plans are becoming increasingly common in NSW. With reference to appropriate standards - in particular section 4.2.2 of AS4804 the Australian Standard for Safety Management Systems, safety management systems should be based on the assessment of hazards.

Management systems are the means by which the interrelationships of the various system elements; such as rules, procedures, standards, responsibilities, authority levels, and action response procedures, combine to control risk.
At some mines the management systems are in themselves component or modules of an integrated business or safety management system established in electronic form on a server with full document management and security.

Risk management consultants often come across all encompassing and very comprehensive management system documents that despite their excellent content are cumbersome and difficult to follow. There are few people apart from management and those involved with the system development that require this level of detail in a one-stop-shop. We recommend to clients that they break the system up into digestible elements so the people required to use them can do so with ease. Administrative-type sections of the system should be removed for those that need to conduct those tasks. For example the continuous miner operator skills and knowledge requirements are completely different from those of a Shift Undermanager.

A natural hierarchy of system elements in an integrated mine safety system would place matters like document control, auditing and responsibilities database as higher level systems, applicable across all hazard management systems. Miner operators would only be concerned with support rules and support installation procedures for example. Undermanagers would be involved in training, development, monitoring and compliance auditing.

It is important to clearly define roles within the strata control management system so people take the initiative to ensure implementation to standard. To each person’s role should be attached the standard to be achieved and means for that person to measure compliance to the particular standards. For example having miner operators report each shift on the sections of roadway they have developed, including width and alignment and reasons for any non-compliance. Similarly personnel installing roof supports should report on the areas supported, conditions encountered, the level and type of support installed and reasons for any non-compliance with the support standards. The mining official in charge of the panel should report on the findings during the shift and actions taken in response to matters of non-compliance. It is important to get the activities for each role and the corresponding responsibility for achieving the prescribed standard aligned. Without clear definition and logical cause and effect type structures the management system will be less than effective.

Clause 48 of NSW Regulations seeks to involve mines in a process of evidence-based design of support systems, presentation in understandable format and the training and monitoring of support systems. The principle of this regulation is consistent with sound engineering and management practice.

Strata control management systems should further develop this philosophy in accordance with the particular requirements of the mine. Geotechnical design methods and the intensity of monitoring will be very different between mines compare a cut and flit in strong strata to longwall in soft strata.

The support designs will then be developed to accommodate expected conditions for which there may be several levels of support.

A three level support regime is provided for mining below abandoned workings. Detailing the support standards for predetermined mining domains as colour coded on mine operating plans as shown in Table 7.

<table>
<thead>
<tr>
<th>DOMAIN</th>
<th>APPLICABLE TO</th>
<th>SUPPORT STANDARD</th>
</tr>
</thead>
</table>
| **Green** | Mining below xxx Seam solid coal and first workings as shown on Strata Control Domain Plan | Mains Development Green Support Standard  
Sub-panel Green Support Standard  
Sub-panel Green Breakaway Standard |
| **Amber** | Mining below xxx Seam goaf areas without pillar remnants as shown on Strata Control Domain Plan | Mains Development Amber Support Standard  
Sub-panel Amber Support Standard  
Sub-panel Amber Breakaway Standard |
| **Red** | Mining in the zone of influence of the xxx Seam goaf edges as shown on Strata Control Domain Plan and pillar remnants encountered | Mains Development Red Support Standard  
Sub-panel Red Support Standard  
Sub-panel Red Breakaway Standard |
Having arrived at an appropriate support design based on all the evidence, including those matters identified by the risk assessment process, namely the full range of design parameters - either defined or the flagged for definition by the appropriate process, the System should then roll-out those design assumptions as triggers to identify the appropriate type of support to be used and when conditions are outside the support design criteria, i.e. “Stop Mining” and holla for the geotechnical engineer to reassess the support regime.

Personnel must be adept at verifying that the design criteria used to develop the support system, standard or rule is present in the workplace in order to install the appropriate support for that ground. And quickly recognise when the mining conditions are outside the scope of the support design.

Ensure that excuses and temptations to venture under unsupported roof areas are eliminated or tightly controlled by the system.

Obtain input from the geotechnical design engineer on the strata monitoring program

**CONCLUSIONS**

In conclusion the following points should be remembered:

- Legislation has enshrined risk management in how we operate.
- Risk assessment is a management tool and can be applied holistically to the business to identify the critical areas of need, including safety and with an integrated systems approach achieve greater success.
- Risk assessment does not replace sound evidence-based engineering design, but will alert engineer and managers to all the issues in risk priority order at pre-design and pre-implementation stages.
- Risk assessment has its limitations. Become proficient in its application and maximise benefits.
- Action plans are essential. Follow-up outcomes to ensure they are implemented as intended.
- Management systems must be presented in digestible chunks for those that are to use them – don’t overburden people unnecessarily, teach them the bits in which they need to be proficient.
- Assign clear logical responsibility to personnel according to their ability to control process.
- Follow-up with support system monitoring, including behaviour compliance.
- Independently audit actual implementation of effective control of hazards and not just the paper trail. Be worried it you score 100%, you are probably not getting the whole story.

The philosophy of risk management is here to stay and is applicable to all areas of mining activity. Mines can maximise benefit if they embrace the concept and apply it to the whole of the business to identify the important issues and manage for greatest effect.

**REFERENCES**

- NSW Coal Mines Regulation Act 1982, No. 67
- NSW Coal Mines (General) Regulations 1999
- NSW Coal Mines (Underground) Regulations 1999
- Queensland Coal Mining Safety & Health Act 1999
- Queensland the Coal Mining Safety & Health Regulations 1999
- Safety Systems Society Handbook 1993
INVESTIGATIONS AIMED TO IMPROVE TAILGATE SERVICEABILITY AT DARTBROOK MINE

Greg Tarrant¹, Rod Doyle² and Ken Mills³

ABSTRACT: Dartbrook Mine has experienced rib control difficulties because of deterioration in the tailgate corner of the longwall face as overburden depth has increased. This paper summarises an investigation to optimise support and develop strategies to improve the serviceability of the tailgate roadways.

Field measurements undertaken in the tailgate of Longwall 6 identified roadway softening mechanisms, deformation characteristics and factors controlling deformation. This provides the basis for optimising the reinforcement system as part of an ongoing Strata Management Plan at the mine.

INTRODUCTION

Tailgate conditions experienced during longwall extraction have variously included:

- Severe rib softening to the extent that rib to rib reduction in width of up to 2m has been noted.
- Guttering and cavity formation ahead of the Longwall chocks at the tailgate end.
- Roof falls adjacent to the faceline in the tailgate.

A program of geotechnical work was undertaken to gain a better understanding of the deformation characteristics, the deformation mechanics and the major driving forces. This information provides the basis for design guidelines to improve roof and rib control and management of tailgate conditions.

SCOPE OF INVESTIGATIONS

The scope of the investigations included field based monitoring, analytical evaluation of pillar loading and limited monitoring of longwall chock loading. The focus of this paper is on the field investigation which was conducted at two sites: inbye 10 Cut-through and inbye 5 Cut-through. Figure 1 shows a summary of the monitoring at 10 Cut-through. The measurements undertaken included:

- Depth of rib yield on the chain pillar and block sides.
- Magnitude of roof and rib displacement.
- Timing of deformation with respect to longwall face position.
- Shear displacement in the roof.
- Load developed in the standing support.
- Load profile developed in the roof bolts.

The instruments were installed in advance of longwall mining and monitored during longwall extraction.

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² Dartbrook Coal Pty Ltd
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RESULTS

The deformation characteristics of the monitoring sites are included in Fig. 1. Results are summarised as follows:

- Softening of the ribs both sides of the roadway (5.4m wide by 4.0m high) occurred to a depth of approximately 5m.
- The softening within the ribs extended into the floor on the block side and the roof on the chain pillar side.
- The intensity of the softening increased steadily from approximately 75m outbye the faceline and accelerated within 20m of the faceline.
- The roof and floor outside the area described above exhibited minimal deformation until within 20m of the faceline. Then the movement within this area was characterised by shear displacement along horizontal planes, essentially creating a shortening of the roof and floor heave.

Results from the 5 Cut-through monitoring site are summarised in Figure 2. The location of and sense of shear displacement is apparent. A contour of displacement measured by the extensometers is also shown.
DISCUSSION OF RESULTS

Deformation is characterised by intense shearing of the rib in the top right and lower left corners of the ribs looking inbye as shown in Fig. 3. The zone of intense shearing extends sub-horizontally into the rib for an estimated distance of up to 2m. Further rib spall associated with cleat produces a softened zone extending into the rib some 50% to 65% of the rib height. The lower right and upper left ribs looking inbye typically exhibit no visible signs of deterioration.

Typically the roof conditions are excellent with no visible signs of deterioration. The angle of the roadway with respect to the seam dip is considered a major factor in producing the side to side rib softening bias. The horizontal roof of the heading makes an angle of approximately 6° with the seam. The stresses within the coal seam appear to align with the seam dip and cause biased rib deterioration.

The vertical abutment loads for future tailgates are predicted to increase as depth of cover increases which implies that the induced floor heave and roof shortening is likely to increase in subsequent tailgates. An optimised reinforcement strategy is aimed to compensate for this anticipated increased deterioration.
REINFORCEMENT STRATEGY

The monitoring program has defined the roadway deformation characteristics and allowed the formulation of a ‘working theory’ regarding the deformation mechanics.
The current reinforcement design is composed of initial primary roof and rib bolting, intensive secondary rib support and tertiary support in the form of cans and timber cribs in the tailgate. With this system, excessive rib deformation is still evident.

The monitoring results indicate that compression of the coal ribs on either side of the roadway from the approaching vertical abutment loads results in severe rib softening and, within 20m of the faceline, roof and floor shortening.

While it is not possible to avoid some increase in rib softening associated with abutment loading, it is aimed to reduce the current level of rib deformation through modifications to the existing control methods. This is a balance between geotechnical considerations, and issues such as safety, cost, machine availability and contractor access.

The underlying philosophy of the approach described is to increase the post failure strength of the softened coal and maintain roadway serviceability.

The following aspects are considered to maximise the possibility of generating confinement within the ribs through reinforcement:

- Installation of reinforcement as early as possible.
- Use of the high capacity reinforcement systems.
- Full encapsulation.
- Targeted reinforcement to the known areas of softening identified through mapping and observation.

CONCLUSIONS

Based on the monitoring data, the following guidelines are proposed to improve the level of rib deformation and reduce roof and floor shortening:

- Use of fully encapsulated long tendons.
- Biased long tendon pattern targeted to increase reinforcement density in areas of expected softening.
- Length of tendons increased to 6m where possible.
- Use of ‘caged’ type tendons over plain strand tendons.
- Installation of long tendons prior to side abutment loading where possible.
- Continued use of rib mesh and supplementary bolts to maintain the integrity of the immediate rib skin.
- Standing support varied to suit the prevailing conditions.
THE DEVELOPMENT OF HAZARD PLANS AT KESTREL MINE

Nick Gordon

ABSTRACT  Kestrel Mine is currently longwalling in the 2.4-3.3 metre thick German Creek seam, at depths between 220 and 295 metres. Run of mine production for the year 2002 is budgeted at 5.3mt, ramping up to 6mt by 2004. Achieving these levels of production will require a thorough understanding of the geological and geotechnical environment. The identification of potential hazards prior to intersection, is vitally important to ensure a consistent coal flow, free of major disruptions and downtime. Hence the preparation of development and longwall panel hazard plans, is an integral part of the production process. Emphasis is on the compilation of detailed, but easy to read plans for the principal users, the workforce at the face.

INTRODUCTION

Kestrel Mine is located 48km northeast of Emerald and 354km by rail from the Port of Gladstone (Fig. 1). The mine is managed by Kestrel Coal Pty Ltd, a wholly owned subsidiary of Pacific Coal Pty Ltd, itself a subsidiary of Rio Tinto. Pacific Coal Pty Ltd gained control of the mine when it purchased ARCO’s 80% interest in the project joint venture in late 1998. Mitsui Kestrel Coal Investment Pty Ltd holds a 20% interest.

The Kestrel Coal mining lease (ML 1978) adjoins BHP Gregory and Crinum to the north and the Ensham deposit to the south (Fig. 1). Following the purchase of Kestrel (formerly Gordonstone) from ARCO, development by Kestrel Coal started on the 23rd February 1999. Extraction of Longwall 107 began on the 28th May 1999. By the end of 2001, more than 10 million tonnes ROM had been mined. Production is planned to ramp up to 6 million tonnes ROM by the year 2004.

Longwall extraction during 2002 will be in 205 (3.4km long) and 206 (2.9km long) Panels (Fig. 2). The majority of the development is scheduled in 207, 208, 301 and 312 Panels. By the end of 2003 longwalling will be...

1 Kestrel Coal
Kestrel Mine is located in a relatively undeformed and stable area of the Bowen Basin. However, as with all underground operations, there are a number of geological and geotechnical issues, which affect both development and longwall extraction:

- **Variability in strata.** This can occur within short distances laterally. This is the major issue at Kestrel and is well illustrated in Fig. 3 which shows two roof cores sampled only 200 metres apart in a gateroad development panel. Critical to the recognition of weak roof and floor zones has been the compilation of roof and floor strata condition maps, an example of which is shown in Fig. 4. The zones are based not only on strength, but also on the number of laminations and presence of weaker layers. It is these weaker layers and laminations that play an important role in roof behaviour at Kestrel. The strata condition maps form the basis for determining the primary and secondary support patterns. For example, in the current mining area, roof with a rating of two is bolted with 4x2.1m bolts/m with six bolts through intersections. Areas, at similar depths of cover, with rating five roof require a minimum of 6x2.1m bolts/m with eight bolts through intersections and a denser secondary support pattern.
Primary Roof Rating 2 (15-30MPa)

Primary Roof Rating 5 (<7.5MPa)

Fig. 3 Comparison of Primary Roof (0-2m above seam) Ratings

- **Weak roof.** Typically this occurs on the 100 side of the mine and shallower side of the 300 series panels. Roadways in an unfavourable orientation to the horizontal stress direction may be adversely affected, both on development and retreat.

- **Weak floor.** This affects the majority of the Kestrel mining area. The most significant operational issue is reduced clearance, due to floor heave, both along the longwall face and in the gates. In addition build up of floor material around the pontoons is exacerbated in soft floor conditions. Elsewhere around the mine, floor heave can damage drivehead installations. In development weak floor, especially when wet, causes difficult wheeling conditions.

- **Faulting.** Mining areas are bounded by major fault zones (>10m). Within these blocks minor faults (<3m) occur. The location of faulting in the current mining area is shown on Fig. 2.

- **Rib instability due to the cleat direction.** Ribs and CT corners oriented poorly with respect to the cleat direction are prone to rib spall.

- **High gas rib emissions.** This is caused by high coal seam permeability and necessitates gas capture drilling.

- **Sedimentary dykes.** These mostly consist of clayey sandstone (<1m wide). Occasionally associated with flanking faults (sedimentary not tectonic), which to date have not impacted significantly on the mining operation.
HAZARD PLANS

The function of the hazard plans is to communicate to the workforce the main geological and geotechnical features, which will impact on production. As such there are two types of hazard plan compiled at Kestrel, namely:

- Development and
- Longwall

These hazard plans are therefore a key component of the strata management system. As defined by Macgregor (1998) this system also includes the Strata Control Hazard Management Plan (Kestrel, 2001a), Trigger Action Response Plans (TARPs), together with the relevant Standard Operating and Working Procedures and a training and a strata management team.

A schematic to illustrate the data sources used in the compilation of these hazard plans, is shown in Fig. 5. The main source is the geological and geotechnical database, which is discussed in more detail below. This is supplemented by the Hazard Management Plans, TARPs, Managers Support Rules, risk assessments and previous mining experience.
An extensive geological and geotechnical database has been collected at Kestrel and is constantly being updated as mining proceeds, both through exploration and in seam development. This includes:

- Compilation of roof and floor zone maps.
- Underground roof and floor coring along the gateroads.
- In-situ stress measurements - both from surface boreholes and in seam measurements
- Geological and geotechnical mapping of all development panels and longwall faces.
- Sedimentological model for the lease.
- Roof and rib monitoring using telltales and extensometers.
- Monitoring of instrumented roof bolts to provide data on bolt performance.
- Roof bolt pull out tests.
- Rock property testing, including determination of bedding plane and discontinuity properties.
- Gas quantity and content testing
- Longwall face monitoring.
- Thin section petrography and X-ray diffraction (XRD) analysis to better understand the rock composition, particularly for frictional ignition potential.
- Numerical modelling of pillar and roadway behaviour.
- Numerical modelling of longwall caving and support loading characteristics.

As evidenced by the amount of data required, the plans are only as accurate as the information available, prior to mining. The individual hazard plans are discussed below in the order of extraction i.e. development followed by longwall retreat.

**DEVELOPMENT PANEL HAZARD PLAN**

It is important that prior to the start of development in a new panel at Kestrel, an assessment of the strata conditions is made to enable design of the support patterns. This is based on both the geological and geotechnical data, primarily the roof condition maps, and experience from adjacent previously mined panels. The different strata zones are colour coded on the development plan for easy reference. Along side each zone is the relevant support rule number, both primary and secondary and the Level 1 Trigger, for both roadways and intersections.
The trigger levels are based on previous monitoring at Kestrel in different roof conditions and are defined in the Development Roadway TARP of the Strata Control Hazard Management Plan.

To supplement this information, other relevant geological and operational data is recorded as follows:

- Floor strength.
- Seam thickness and grade.
- Depth of cover.
- Gas makes and content.
- Water makes.
- Stress direction.
- Location of geological features e.g. split lines, faults, dykes, and cleat direction.
- Location of boreholes both in seam and surface and cementing status.
- Location of phones, DACs, fire depots, trickle dusters, transformers and SSR90 (self contained rescuers) caches.

These plans are intended to provide a summary of the geological/geotechnical information, that is most relevant to the mining crews for each development plans. Laminated A0 colour copies are posted in the underground crib rooms. (Fig. 6). Additional copies are kept at various locations on the surface, but realistically it is the face area where this information, on predicted mining conditions, is required. The support rules referred to on the development panel plan are also displayed in the crib room. A standard used at Kestrel is the production of A4 laminated copies of the support rules with the roadway development TARP on the reverse side, for reference, attached to both sides of the continuous miner. This is where the information is most required, not back in the crib room some 200 metres away.
MONITORING AND UPDATING DURING DEVELOPMENT

During development, localised lithological changes may be encountered and these should be recorded to update the roof and floor strata condition maps. Although these maps form the basis for designing support rules, the importance of underground monitoring, to verify and continually fine tune the support design in future areas of the mine, should not be underestimated. This monitoring includes:

- Geological/geotechnical mapping by the geotechnical engineer.
- Observations by deputies and crews.
- Geotechnical monitoring (e.g. CLOCKITs) and
- Trials of new or alternative roof and rib support systems (Ward, 2000).

Mapping in a development panel includes recording all geological features and conditions. In addition, the primary and secondary support density is also recorded, as shown in the example in Fig. 7. This mapping is supplemented by geotechnical data collection such as roof bolt pull out tests and roof coring.
Fig. 8 Kestrel Mine Statutory Report

One of the most useful sources of information is that recorded by the deputies on their statutory report. The information recorded in the geotechnical inspection section (Fig. 8) which was obviously significant during the shift needs to be followed up. It has been found at Kestrel that by letting the deputies know that their statutory reports are being read and used in the compilation of hazard plans, improves the quality of information recorded (Table 1). Anyone who is recording information without any follow up and feedback will gradually lose interest and eventually stop. This information is invaluable and when talking about a strata management team the best resource is at the face.

Table 1. Deputies Record Of 205 Development

<table>
<thead>
<tr>
<th>Location</th>
<th>Date</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>A19 niche</td>
<td>17/1/01</td>
<td>Supported niche with mesh as the roof is very flaky.</td>
</tr>
<tr>
<td>20CT</td>
<td>23/1/01</td>
<td>Roof slabbing a bit in CT, more on the LHS.</td>
</tr>
<tr>
<td>20CT</td>
<td>24/1/01</td>
<td>Ribs poor on intersection breakoff.</td>
</tr>
<tr>
<td>B20</td>
<td>8/2/01</td>
<td>2 large cracks in B20 intersection. No signs of weight on bolts. Some rib fretting above phone.</td>
</tr>
<tr>
<td>A20-21</td>
<td>8/2/01</td>
<td>ODS rib flaky, 200mm rock D/S cut because roll in the seam, 28m mark inbye.</td>
</tr>
<tr>
<td>B21-22, 88m</td>
<td>16/2/01</td>
<td>Slabs out of floor.</td>
</tr>
</tbody>
</table>
LONGWALL PANEL HAZARD PLAN

With the completion of development, new data is reviewed and updated, to assist in the compilation of the longwall panel hazard plan as referred to in Fig. 5. A final inspection around the block is considered an essential part of this compilation process. This is carried out by the geotechnical engineer and a longwall operations person, usually the coordinator. Hazards identified during development from:

- geological and geotechnical mapping and
- deputies statutory reports.

are used as a checklist to ensure that none have been overlooked. Operational hazards such as offline belt road drivage, ballast, concrete, soft floor due to water, gas drainage discharge points, low clearance are documented during this inspection, for inclusion on the hazard plan.

In addition to hazards identified during development, deputies records of previous longwall extraction are very useful in predicting future extraction conditions, when used in conjunction with the geological and geotechnical database. A typical deputies record of extraction is shown below in Table 2. These reports are invaluable and also used for input into the hazard plans.

**Table 2. Deputies Record Of 204 Extraction**

<table>
<thead>
<tr>
<th>Date</th>
<th>Chainage</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>23/5/01 D/S</td>
<td>2081</td>
<td>B22 heavy left hand side, cogs needed next 2 CTs</td>
</tr>
<tr>
<td>23/5/01 N/S</td>
<td>2067</td>
<td>Some floor heave after being down at SOS</td>
</tr>
<tr>
<td>24/5/01 D/S</td>
<td>2059</td>
<td>Seam split. Not causing any problems floor a bit soft. 90-140 stone band 1m above floor at 90 but disappears at 110 (100mm thick).</td>
</tr>
<tr>
<td>24/5/01 N/S</td>
<td>2059</td>
<td>Shovelling soft floor Maingate.</td>
</tr>
<tr>
<td>27/5/01 N/S</td>
<td>2030</td>
<td>SOS floor heave 30-45</td>
</tr>
<tr>
<td>28/5/01 N/S</td>
<td>2023</td>
<td>55-70 regular slabbing to 200mm.</td>
</tr>
</tbody>
</table>

At this stage a draft plan is prepared with the available data. Prior to finalising, a risk assessment process is carried out, as detailed in the Kestrel Coal SOP – Developing and Carrying out Second Workings (Kestrel 2001b). This assessment must cover the matters mentioned in the Queensland Coal Mining Safety and Health Regulation 2001, Section 317, namely:

1. Any surface features, artificial structures and water reserves that may create a hazard if disturbed by the workings.
2. Any other workings located in close proximity above, below or adjacent to the proposed second workings, whether in the same or an adjacent mine.
3. The known geology affecting the intended workings.
4. The anticipated gas make.
5. The pillar stability.
6. The proposed method and sequence of coal extraction.
7. Support methods necessary to control the edges of the each goaf area in active workings.
8. The suitability of the plant, and its controls, used for the workings.
9. The proposed methods for the following:
   a. Strata control and support
   b. Ventilation
   c. Controlling spontaneous combustion
During the risk assessment the draft plan is presented and discussed, to identify any changes or additions. Following the risk assessment, the hazard plan can be finalised and then presented to the crews, prior to mining (Fig. 9).

Fig. 9 Longwall Extraction Hazard Plan.

As with the development plans, the longwall plans are intended to highlight the main hazards in an easy to read format (Fig. 9). It is important that any variations from those shown on the development and longwall hazard plans are recorded to allow the continual improvement in the accuracy of future hazard plans.

ACKNOWLEDGEMENTS

The author would like to thank Kestrel Coal for permission to publish and present this paper. The views expressed are those of the author, and not necessarily Kestrel Coal.

REFERENCES

USE OF CONE BOLTS IN GROUND PRONE TO ROCKBURST

Ron McKenzie¹

ABSTRACT: Big Bell Mine started using the Cone bolt as a yieldable support to combat rockburst in September 1999. Australia has little experience of mining in rockburst conditions which makes the ground support requirements a new experience to all involved. Much work went into the design of the support patterns and the sequence of operations. The workforce required training in installation procedures, the theory behind yielding support and in the importance of good quality installation practices. Testing has been carried out on various grout mixes to determine the best mix for optimum bolt performance. This paper gives an overview of the introduction of Cone bolts at Big Bell.

INTRODUCTION

Big Bell Mine is located 25 km north west of Cue in Western Australia and about 540 km NNE of Perth. The mining method used is longitudinal sub-level caving with current operations working to a depth of approximately 535 metres. A combination of natural stresses and mining induced stresses has resulted in a number of rockburst ground failures at Big Bell. Production in the lower levels of the mine was halted in September 2000 after a number of major rockbursts, one resulting in a fatality. Production prior to closing the lower levels was 1.8 million tonne per year yielding 160,000 ounces of gold.

A major upgrade of the ground support to combat rockbursts in the lower levels was begun in August 2001 after a decision was made to continue mining the lower levels of the orebody. The yielding Cone bolt tendon was chosen as the primary support. The upgrade consisted of rehabilitation of existing drives using 4 metre long Cone bolts. It also involved replacing the tubular groutable rockbolt and the grouted rebar with 3 metre Cone bolts as the main form of reinforcement in the new development drives. 24,000 Cone bolts are expected to be installed before the mine returns to normal production.

Non-seismic support of debonded Gewi-bars is used in drives more than 75 metres from the orebody.

ROCKBURST

Rockbursts are explosive failures of rock when very high stresses occur around mine openings. They result in the ejection of rock ranging from a fraction of a cubic metre to thousands of cubic metres. Fig. 1 shows the damage caused by an outburst. The seismic energy associated with the rock ejection process can reach the equivalent an earthquake of magnitude five on the Richter scale. An event of this magnitude is usually associated with fault slip activity and has only occurred in South Africa. The largest reading at Big Bell has been 2.4, with the biggest rock fall being approximately 1,000 tonnes. Geophones are positioned throughout the mine to determine intensity and location of seismic events.

Ejection velocity can be up to seven metres per second with commonly around one metre thick rock being ejected.

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REINFORCEMENT REQUIREMENTS

The large displacements and high strain rates resulting from a rockburst cannot be countered by conventional stiff reinforcement systems. For the support to be effective it must be able to absorb the kinetic energy, imparted to the rock mass by the seismic event, without failing. Support tendons spaced at one per square metre must have the capacity to absorb at least 60 kJ of seismic energy. This is equivalent to the kinetic energy in a one metre thick slab in the backs, accelerated to a velocity of 7 metres per second. The energy absorbed by conventional rigid tendons prior to failure is of the order of 4 to 6 kJ.

In addition, the support has to allow for at least 500mm of rock displacement without failing. Conventional end-anchored or grouted support tendons can fail when subjected to deformation as low as 60 mm.

CONE BOLTS

The Cone bolt was developed as a result of work done by the Chamber of Mines Research Organisation (COMRO) in South Africa and is manufactured by Steeledale Strata Control Systems.

The toe end of the bolt has a forged conical enlargement. The collar end is threaded like a conventional roof/rock bolt or may have a ‘Shepherd’s Crook’ eye to facilitate cable lacing. The entire length of the bolt is coated with a debonding agent to reduce the resistance caused by the bond between the grout and the bolt. Both types of bolt are used with a bearing plate and in the case of the threaded bolt, a nut and hemispherical washer is also used (Fig. 2).

The bolt chosen for Big Bell had a diameter of 22mm, a cone diameter of 32mm and an M24 thread. The bolt has a yield strength of 22 tonne.
Fig. 2 Threaded Cone Bolt

INSTALLATION

Two different length cone bolts are installed, three metre in new development drives and four metre in the rehabilitation of existing drives. Hole diameters vary for the different lengths. The three metre bolts in the new development are installed in 45mm diameter holes drilled by the face jumbo. The four metre bolts are installed in 51mm diameter holes simply because of the need to do extension drilling for the longer holes.

The hole is pumped full with a thick self-supporting grout and the bolt pushed into the hole through the grout. Minimal thread is left protruding to allow the mesh to be pulled to the backs. The bolt is then tied to the mesh to prevent it sliding out of the hole.

After the grout has cured sufficiently, which takes six hours in the case of new development drives, the bearing plate, ball washer and nut are fitted and the whole system tensioned using an impact wrench.

Bolting patterns vary depending on drive type whether rehabilitation or new drives-drive size, orientation, and location in relation to seismic prone areas. Generally speaking eight to ten Cone bolts are installed in a ring pattern on 1.2 metre spacings. Fig. 3 shows the bolting pattern for 4.5 x 4.5 metre ore drives.

The rate of installation is another issue, especially in the rehabilitation areas. It was originally expected that the contractor would be able to install up to 60 of the 4 metre Cone bolts in pre-drilled holes in a 12 hour shift. This has been met on occasions, but the average is well below this level.

The main reason for the lower than expected installation rate is the high number of redrills required as a result of collapsed holes in the broken ground of the rehab areas. Reliability and performance of the grouting equipment also caused problems.
The forces generated in a rockburst are transferred to the bolt via the bearing plate. Depending on the grout strength and quality of installation, the cone is drawn through the surrounding grout. The resistance to displacement is a combination of the friction between the grout and the cone and the force required to compress or crush the grout. In so doing, work is done and energy absorbed from the surrounding rock.

Once all of the energy has been absorbed, a state of equilibrium is achieved and the bolt stops yielding and contains the surrounding rock. Any subsequent build up of force in the bolt will result in the yielding process being re-initiated.

The broken rock between the bolts is supported by mesh and/or lacing.
PERFORMANCE

The major factors affecting the performance of the Cone bolt at Big Bell are the properties of the grout and the quality of installation. There have not been any rockbursts since the introduction of the full ground support Cone bolt pattern shown in Figure 3. Performance of the bolt will not be known until normal production resumes. However previous seismic events, in which cone bolts were used in localised areas, have shown that the damage from a rockburst can be contained by Cone bolts and mesh installed in a suitable pattern.

The initial grout mix used consisted of general purpose (GP) cement with a 0.33 water cement ratio, using mine water, and 2.5% w/w Sikament HE200NN. This additive is a combination of superplastiser and accelerant. The superplastiser improves pumpability and reduces the water demand of the cement. The accelerant is added to provide high early strength to enable the bolt to be tensioned six hours after installation in the new development drives.

A retardant is added to the grout in the rehab areas to give the crews more working time because of delays encountered with blocked holes in the broken ground.

Fig. 5 shows the results of pull tests on a three metre Cone bolt for static testing and quasi-dynamic testing. The static test was the standard pull test where the rate of loading of the bolt is low (less than one tonne per minute) and displacement readings are taken only after stabilisation. The quasi-dynamic test involved loading the bolt at the maximum rate of the testing equipment, 3.5 mm/sec, and recording the maximum load reached during each extension of the ram.

The results from the latter test showed that the bolt was reaching loads approaching the yield strength of the bolt. Numerous UCS tests on the grout mixes gave varying results but tended to be on the high side, that is, greater than 50MPa. However from a number of seismic events that have loaded Cone bolts at Big Bell there is only one documented case where a Cone bolt had failed. The grout mix was changed to reduce the strength. The superplastisiser was cut from the mix to increase the water demand, and hence reduce the strength of the grout, while at the same time the low slump properties were retained.
Grout strength results varied widely making it difficult to standardise the mix. However increasing the water cement ratio had the effect of reducing the strength to around 35 to 40 MPa. Pull tests have yet to be performed for these grout mixes. If the results are still too high then air entraining admixtures will be looked at for reducing the grout strength. It is not possible to keep adding water to reduce the strength. The grout becomes too thin and simply runs out the hole.

The other major issue affecting bolt performance is quality of installation. It is important that the grout column in the hole is continuous and that no air pockets are present.

When the bearing plate, ball washer and nut are fitted during the tensioning procedure it is important to ensure that the plate pulls the mesh into cavities so that the mesh contours the rock surface.

Tensioning was done using a pneumatic impact wrench set to stall at approximately six tonne. This was sufficient to bed the plate and mesh to the backs and to tighten any “pregnant” mesh in the rehab areas.

**TRAINING**

Considerable work was done by the geotechnical engineer and consultants at Big Bell to produce a ground support management plan to combat rockburst. Integral to the success of the plan is the performance of the ground support crews. It is important they maintain the correct grout specifications, installation quality and installation rate. To achieve this, training programs were designed that involved classroom sessions as well as time underground with each crew during each stage of the installation. The crews learned not only how to do the job properly but gained an understanding as to why they were doing it.

Big Bell, like a lot of mines in Australia, is a fly-in, fly-out operation, which often leads to a high turnover of the workforce. This places added importance on the training system to ensure that every worker is competent in all aspects of Cone bolt installation. Retaining experienced workers is also important.

**CONCLUSIONS**

The future of mining operations at Big Bell depends, to a large extent, on the success of the Cone bolt installation in both the rehabilitation drives and the new development drives. The performance of the bolt is heavily dependent on the specifications of the grout, the quality of installation and correct pattern drilling. A lot of grout testing has been done to assess strength and curing times. The workforce has a clear understanding of the importance of installing the bolt correctly.
An audit of installation procedures was performed three months after the start of the rehabilitation work and a recommendation made to modify grouting equipment and procedures, primarily to improve installation rates.

Constant training, assessment, auditing and testing are required to ensure success of the project.

BIBLOGRAPHY


A NEW TECHNIQUE TO DETERMINE THE LOAD TRANSFER CAPACITY OF RESIN ANCHORED BOLTS

Najdat Aziz 1

ABSTRACT: This paper describes a new technique to evaluate the load transfer capacity of different resin anchored rock bolts. With an increasing number of rock bolts currently being introduced into the Australian market for use in a variety of ground conditions, a new technique to determine bolt load transfer capacity is necessary.

The assessment of the performance bolt with regard to load transfer mechanisms is conducted in the laboratory under Constant Normal Stiffness (CNS) conditions. This method of testing is considered as being a realistic way of evaluating bolt surface roughness as the tests are carried out under different confining pressures thus accommodating the changes in ground conditions such as high horizontal stress while allowing for surface dilations due to rubbing of rough surfaces against each other.

INTRODUCTION

In the market today there is a variety of different rock bolt designs deployed for strata reinforcement. These rock bolts vary in appearance, based on the way they are manufactured, and how they are to operate in strata support applications. This paper is only concerned with resin or grout anchored rock bolts and is not concerned with point anchored or friction anchored rock bolt system. Nevertheless, the basic resin or grout anchored rock bolt consists of a solid steel bar with some form of rib or thread profiles hot rolled onto the outside of the bar, as well as a nut and a thread at one end of the bar to enable the nut to be tightened up against the bearing plate and rock face. Irrespective of the bolt type, it is this surface profile that plays a major influence on the effective functioning of the bolt as it influences the load transfer mechanisms between rock, resin and rock bolt.

Currently there are two common methods of assessing the load transfer capability of bolts, the well-known pull out test, and short length push test. Both tests are conducted under constant normal load condition, which is applicable to shearing across planar and regular surfaces whereby the process of shearing does not produce any noticeable vertical displacement across the shearing surfaces. Thus, both systems of testing ignore the additional forces generated due to vertical displacement of the resin during the shearing process caused by bolt ribs. The results of these tests can also be influenced by such factors as the annulus thickness of the resin encapsulation and improper mixing of the resin in the hole, commonly known as gloving.

Strain gauged instrumented rock bolts installed underground is the commonly accepted method of determining the load performance of a bolt and thus the shear stress developed at bolt-resin interface (Fuller & Cox, 1975; Gale, 1986; Fabjanczyk and Tarrant, 1992 and Signer, Cox & Johnston, 1997). The shear stress developed at any point along the bolt length could then be calculated by the following formula:

$$\Delta \tau = \frac{F_1 - F_2}{\pi d l}$$

Where,
- $\Delta \tau$ = Shear stress at bolt-resin interface,
- $F_1$ = Axial force acting on the bolt at strain gauge position 1, calculated from strain gauge reading,
- $F_2$ = Axial force acting on the bolt at strain gauge position 2, calculated from strain gauge reading,
- $d$ = Bolt diameter, and
- $l$ = Distance between strain gauge position 1 and strain gauge position 2.

1 University of Wollongong
One of the major shortcomings of the above method is that, it does not consider the effect of horizontal stress or the confining pressure on the shear stress at the bolt/resin interface.

Accordingly, testing for load transfer mechanisms of a bolt can realistically be achieved if conducted under CNS conditions as it represents a better simulation of the changing stresses in the field. This paper describes the CNS test of bolts in the laboratory and highlights the latest modification to the future methods of testing, currently been carried out at the University of Wollongong. The findings from the laboratory study is supported with field investigation to evaluate the behaviour the two different profiled bolts under changing ground stress conditions.

**BOLT-SURFACE PREPARATION**

A 100 mm length of a bolt was selected for the surface preparation for CNS shear testing. The specified length of bolt was cut and then drilled through. The hollow bolt segment was then cut along the bolt axis from one side and preheated to open up into a flat surface as shown in Fig. 1. The surface features of the bolt (ribs) were carefully protected while opening up the bolt surface. The flattened surface of the bolt was then welded on the bottom plate of the top shear box of the CNS testing machine. Table 1 shows the specification of two types of bolt used in the study, known as Type I and Type II bolts respectively.

![Fig. 1. Flattened bolt surface](image1)

![Fig. 2. A typical cast sample](image2)

**Table 1. Specification of bolts**

<table>
<thead>
<tr>
<th>Bolt</th>
<th>Core Diameter (mm)</th>
<th>Finished Diameter (mm)</th>
<th>Rib Spacing (mm)</th>
<th>Rib Height (mm)</th>
<th>Profile Width (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Base</td>
</tr>
<tr>
<td>Type I</td>
<td>21.7</td>
<td>24.4</td>
<td>28.5</td>
<td>1.35</td>
<td>4.75</td>
</tr>
<tr>
<td>Type II</td>
<td>21.7</td>
<td>23.2</td>
<td>12.5</td>
<td>0.75</td>
<td>3.50</td>
</tr>
</tbody>
</table>

**SAMPLE CASTING**

The welded bolt surface on the bottom plate of the top shear box was used to print the image of bolt surface on cast resin samples as shown in Fig. 2. For obvious economic reasons the samples were cast in two parts. The top, one-fourth layer of the sample was cast in resin and the remainder cast in high strength casting plaster. The properties of the hardened resin after two weeks were, uniaxial compressive strength ($\sigma_c$) = 76.5 MPa, tensile strength ($\sigma_t$) = 13.5 MPa, and Young’s modulus (E) = 11.7 GPa. The cured plaster showed a consistent $\sigma_c$ of about 20 MPa, $\sigma_t$ of about 6 MPa, and E of 7.3 GPa.
CNS SHEAR TESTING APPARATUS

Fig. 3 is a general view of the CNS testing apparatus used for the study. The equipment consisted of a set of two large shear boxes to hold the samples in position during testing. The size of the bottom shear box is 250x75x100 mm while the top shear box is 250x75x150 mm. A set of four springs are used to simulate the normal stiffness ($k_n$) of the surrounding rock mass. The top box can only move in the vertical direction along which the spring stiffness is constant (8.5 kN/mm). The bottom box is fixed to a rigid base through bearings, and it can move only in the shear (horizontal) direction. A hydraulic Jack is used to apply the desired initial normal stress ($\sigma_{no}$), which was measured by a calibrated load cell. The shear load is applied via a transverse hydraulic jack, which is connected to a strain-controlled unit. The applied shear load can thus be recorded via strain meter fitted to a load cell. The rate of horizontal displacement can be varied between 0.35 and 1.70 mm/min using an attached gear mechanism. The dilation and the shear displacement of the joint are recorded by two LVDT’s, one mounted on top of the top shear box and the other is attached to the side of the bottom shear box. A total of 12 samples were tested for two different types of bolt surface at initial normal stress ($\sigma_{no}$) levels ranging from 0.1 to 7.5 MPa. Each sample is normally subjected to five cycles of loading in order to observe the effect of repeated loading on the bolt/resin interface. The stress profile, as described above, is defined as the variation of shear (or normal) stress with shear displacement for various cycles of loading. A constant normal stiffness of 8.5 kN/mm was applied via an assembly of four springs mounted on top of the top shear box. An appropriate strain rate of 0.5 mm/min was maintained for all shear tests. A sufficient gap (less than 10 mm) was allowed between the upper and lower boxes to enable unconstrained shearing of the bolt/resin interface.

Fig. 3. CNS apparatus
EFFECT OF NORMAL STRESS ON STRESS PATHS

Fig. 4. Shear Stress profiles of the Type 1 bolt from selected tests

Fig. 4 shows the shear stress profiles of the bolt/resin interface for selected normal stress conditions for the Type I bolts. The difference between stress profiles for various loading cycles was negligible at low values of $\sigma_{no}$ (Fig. 4a). This was gradually increased with increasing value of $\sigma_{no}$ reaching a maximum between 3 and 4.5 MPa (Fig. 4b). Beyond a 4.5 MPa confining pressure, the difference between stress profiles for the loading cycles I and II decreased again (Fig. 4c). A similar trend was also observed for the Type II bolt surface (not shown in the figure). At low $\sigma_{no}$ values, the relative movement between the bolt/resin surfaces caused an insignificant shearing and slickensiding of the resin surface, thus keeping the surface roughness almost intact. For each additional cycle of loading, the shear stresses marginally decreased, especially in the peak shear stress region. However, as the value of $\sigma_{no}$ was increased, the shearing of the resin surface was also increased, and the difference in stress profiles for various cycles of loading became

DILATION BEHAVIOUR

For the first cycle of loading, Figs. 5a and 5b show the variation of dilation with shear displacement at various normal stresses for Type I and Type II bolts, respectively. For various values of $\sigma_{no}$, the maximum dilation occurred at a shear displacement of 17 - 18 mm and 7 - 8 mm, for Type I and Type II bolts, respectively (Figs. 5a and 5b). The distance between the ribs for both bolt types is shown in Table 1. Therefore, it may be concluded that the maximum dilation occurred at a shear displacement of about 60% of the bolt rib spacing.
EFFECT OF NORMAL STRESS ON PEAK SHEAR

Figs. 5c and 5d show the variation of shear stress with shear displacement for the first cycle of loading at various normal stresses, for both Type I and Type II bolts, respectively. The shear displacement for peak shear stresses increased with increasing value of $\sigma_{\text{nn}}$ for both bolt types. This was due to the increased amount of resin surface shearing with the increasing value of $\sigma_{\text{nn}}$. However, there was a gradual reduction in the difference between the peak shear stress profiles with increasing value of $\sigma_{\text{nn}}$. The shear displacement required to reach the peak shear strength is a function of the applied normal stress and the surface properties of the resin, assuming that the geometry of the bolt surface remains constant for a particular type of bolt as evident from Figs. 6 and 7.

Fig. 5. First Loading Cycle Dilation and Shear Stress profiles for both Types I and II Bolts

OVERALL SHEAR BEHAVIOUR OF TYPE I AND TYPE II BOLTS

Fig 6 shows the shear stress profiles of both Type I and Type II bolts for the first cycle of loading. The following observations were noted:

- The ultimate shear strength profiles for both types of bolts is very similar throughout the normal confining stress range, suggesting that it is the ultimate shear strength of the resin which is the controlling and dominant factor at play in this situation.
- Shear displacements at peak shear are higher for the bolt Type I indicating the safe allowance of more roof convergence before instability stage is reached.
- Post peak shear stress values are higher for the bolt Type I indicating better performance in the post-peak region, as closer spaced ribs would tend to break up the resin between ribs more rapidly, and therefore there will be a greater drop off in residual shear strength.
- For both bolts, the maximum vertical displacement or dilation, due to relative displacement of bolt against the resin occurred at a shear displacement of about 60% of the bolt rib spacing.
EFFECT OF NORMAL STIFFNESS

The laboratory experiments were carried out with spring assembly with an effective stiffness of 8.5 kN/mm. In practice, the stiffness of resin/rock system will be usually higher than the laboratory simulated stiffness. As the stiffness increases, the effective normal stress on the bolt/resin interface at any point in time will also increase, as per the following equation:

\[ \sigma_n = \sigma_{no} + \frac{k_n \delta_v}{A} \]

where,
\( \sigma_n = \) effective normal stress,
\( \sigma_{no} = \) initial normal stress,
\( k_n = \) system stiffness,
\( \delta_v = \) vertical displacement (dilation), and
\( A = \) area of the bolt surface.

It is thus reasonable to suggest that, because of the deeper and wider spaced rib profile, the effective normal stresses values will be higher for the Type I bolt as compared to the Type II bolt as long as the confining pressure remains low and higher vertical displacement (\( \delta_v \)).

Fig. 7 shows the variation in peak vertical displacement (dilation) of the resin/bolt contact surfaces with the applied initial normal stress. It can be seen that at low initial normal stress condition, the stress concentration on the resin surface around the bolt ribs is not sufficient to cause resin failure. As a result, the bolt with deeper and wider spaced rib profile will offer higher shear resistance due to higher dilation. However, at high initial normal stress level where the concentrated stress around the ribs is high enough to crush the resin surface, the bolt that has
lower rib spacing is least influenced by increased stress. This is indicated by a relatively flatter diminishing peak dilation curve for type II bolts as compared to type I bolts. In other words, the bolts with lower rib spacing would offer a greater resistance at high normal stress conditions. It is also worth noting that the magnitude of the vertical displacement in Type I bolt is gradually tapering off as the initial normal stress increases.

![Fig. 7 Variation of peak dilation with initial normal stress.](image)

**FIELD INVESTIGATION**

As a part of the research project, field investigations were carried out in a local mine. Six, 2.1 m long, strain gauged instrumented bolts (three bolts from each type) were installed in the roof at a longwall panel cutthrough. The spacing between the strain gauges mounted on each bolt was 200 mm. As can be seen from Fig. 8 the pattern of roadway bolting consisted of six bolts in a row and the spacing between the rows was 1 m. The primary horizontal stress around the region was estimated at around 16 MPa. Excessive guttering at the left side of the cut through manifested the impact of high horizontal stress. Thus, the bolts on the left side of the cut through are likely to be subjected to excessive shear loading as compared to the right side bolts of the cutthrough.

![Fig. 8 Plan of instrumented site](image)
Fig. 9 shows the magnitude of load generated on Type II bolts across the cutthrough, and shows that the bolt on the left side of the cut through experienced relatively higher load transfer in comparison to the bolt at the right side. However, the level of load generated on Type I was different from that of bolt Type II. The variations in the calculated shear stresses for different bolts are shown in Fig. 10. No load build up comparison was possible for the bolts installed in the middle of the cut through as the mid section Type I bolt malfunctioned after a short period of installation. Fig. 11 shows the maximum recorded load and shear stresses generated for both types of bolts during six months of field monitoring of the site.

Fig. 9. Comparison between Type I and Type II bolts

Fig. 10. Shear stress patterns generated on both Type I and Type II bolts at the right side of the cut through

The following points can be drawn from the field study:

1) Relatively higher shear stress was generated on Type II bolts on the highly stressed guttered side of the cut through in comparison to Type I bolt.
2) Relatively higher load was generated on the Type I bolts on the low stressed and gutter free right side of the cutthrough in comparison to Type II bolts.
3) Both the above findings are in agreement with the laboratory findings and above stated empirical relationship related to effective normal stress ($\sigma_{en}$).
4) The instrumented bolts provide a suitable technique in conducting comparative tests in the field to evaluate the suitability of any particular bolt for the prevailing ground conditions.

<table>
<thead>
<tr>
<th>Bolt Type</th>
<th>L</th>
<th>R</th>
</tr>
</thead>
<tbody>
<tr>
<td>Axial Load (kN)</td>
<td>87</td>
<td>66</td>
</tr>
<tr>
<td>Shear Stress (MPa)</td>
<td>6.3</td>
<td>5.8</td>
</tr>
<tr>
<td>Bolt Type II</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Axial Load (kN)</td>
<td>220</td>
<td>19.5</td>
</tr>
<tr>
<td>Shear Stress (MPa)</td>
<td>16</td>
<td>1.2</td>
</tr>
</tbody>
</table>

Fig. 11. Maximum recorded load and shear stresses

CONCLUSIONS

The paper described a new approach to evaluating the load transfer mechanisms in bolts with respect to variations to changes to surface profile of the bolt. The CNS method demonstrated that the technique is a viable alternative to conventional tests for evaluating effectively the load transfer capacity of different rock bolts. The laboratory findings were supported by the variations of the level of load build up on the bolts with respect to the type of the bolt. The benefit of this suggested technique can only be fully appreciated by conducting comparative studies in the field.

Research is currently being undertaken on two more new techniques, which will provide a much faster methodology of evaluating the load transfer mechanism of bolts in the laboratory. These include testing of intact bolts under biaxial conditions and double shear test. The development and refinement of these testing procedures will enable a better understanding of rock bolt behaviour and thus enable engineers to design more effective strata support products.

REFERENCES


INTRODUCTION

Thin Seam Mining Pty Ltd (TSM) is contracted by Allied Coal Pty Ltd (Allied) to mine the Balgownie Seam at Gibson’s Colliery, Russell Vale. TSM will utilise place change systems of work as practiced in West Virginia, USA. For that purpose TSM has imported purpose-built, low-profile mining equipment from the USA. The equipment has been retrofitted in Wollongong with electrical systems that comply with Australian Standards. Presently the TSM workforce consists of approximately 14 operators from West Virginia, and 14 from Australia.

There is more coal mined in the state of West Virginia than in Australia. Most of that coal is mined in coal seams less than 1.5m in thickness. At Russell Vale the Balgownie Seam comprises a high quality coking coal and is approximately 1.2m thick. It was previously mined in the 1960’s and 1970’s via a longwall operation, however that operation proved unviable. The inability of previous operators to extract only the coal seam was one of the reasons previous operations were not sustainable.

Gibson’s Colliery has accessed the Balgownie Seam directly from subcrop on the Illawarra Escarpment. This required the use of concrete portal structures, steel sets, roof bolts, timber, and void fill material. It will be necessary to cut across a number of existing Balgownie Seam roadways in the near future. Those workings are not accessible at present.

In the area of interest, the Balgownie Seam lies approximately 10m below the Bulli Seam where previous operators extracted up to 50% of Bulli Seam coal.

Some of the distinctive features of Gibson’s Colliery include;

1. Up to 70% recovery by first workings alone.
2. A high quality coking coal that is will be blended with other coals, or sold alone.
3. Both the Balgownie seam and the overlying Bulli Seam are believed to be devoid of appreciable levels of water and gas in the area of interest.
4. Roof support that comprises 1.2m mild steel bolts in the mains, and 0.9m bolts in production panels. Secondary support where required comprises 6m long cable bolts.
5. Bolts installed with a Fletcher roof bolter utilising vacuum drilling. Cable tendons are currently installed with a conventional air leg roof bolter.
6. Place changing systems with plunges currently set at a maximum of 14 metres.
7. 100 metres of advance per production shift with a single continuous miner production unit
8. A positive (forcing) ventilation system that offers notable advantages over an exhausting system
9. Utilisation of existing surface infrastructure that has lowered capital investment
10. Two production shifts per day, and one maintenance shift, 5 days per week.
11. A workforce that believes it is possible to mine thin seams safely and at high production rates

GEOLOGY AND STRUCTURE

The roof of the Balgownie Seam consists of an overall fining-upwards sequence from coarse grained sandstone/pebble conglomerate through laminated to fine grained sandstones and mudstones. Within this overall trend, there are local variations in the nature of the immediate roof to the Balgownie Seam, ranging from carbonaceous mudstones through planar and cross-bedded sandstones to pebbly conglomerates. However, the
The vast majority of the roof is a sandstone - flaggy in parts, but mostly massive cross-bedded sandstone. The strength of the sandstone in the roof of the Balgownie seam ranges from 50 to 100 MPa.

Joints in the sandstone roof of the Balgownie Seam trend at approximately 355°, 100°, and 050° with respect to Grid North. At the shallow depths in which mining has been conducted, some of the joints present as decomposed zones. It has been found that there are a number of small throw normal faults that are aligned parallel to the 355° joint set.

The floor of the Balgownie Seam consists of carbonaceous mudstone, coal shale and laminates.

**MINING HISTORY**

**Bulli Seam**

Knowledge of the remnant pillars in the Bulli Seam is based on the original cloth plans held by the colliery. These show areas of hatching marked “pillars extracted” or similar. The nature of the remnant pillars inside such areas is unknown – it is possible that there are large pillars or stooks left in these areas. Elsewhere the plans show Welsh bords and main headings.

Fig. 1 shows a non-validated digitised version of the plan over the immediate area of interest: the Bulli Seam pillars are shown. Taking this interpretation at face value, Gibson’s Colliery will be located under large pillars and goaf.

**Balgownie Seam**

The Balgownie Seam was worked by longwall methods in the mid 1970s immediately to the south. This area was also under Bulli pillar goaf. Discussions with the previous Balgownie Seam miners provided the following information:

1. Roof support was mostly by props and bars
2. Difficult roof control when at shallow depths – headings driven southwest from the escarpment were very poor (note that the Gibson’s layout has different orientations).
3. Conditions improved markedly as they went deeper, and this coincided with a change in direction of the driveages towards the west.
4. There were occasions when the mining conditions were typified by heavy rib crush, a step in the roof, open cracking (dipping between 40° and 60°), and a flush of water. These were interpreted at the time to be associated with remnant pillars in the Bulli Seam goaf.
5. There was a general belief that the poor roof conditions and the larger roof falls developed underneath the Bulli Seam goafs, possibly under remnant pillars in the goaf.
6. Minor roof falls developed in the presence of bedding partings close to the roof.
7. Faults and associated joint zones were often associated with roof falls.

**RISKS**

Two specific risk types were identified as having the possibility of affecting the project.

**Generic Mining Risks**

In July 2001, a core risk assessment identified the following geotechnical risks:

1. Catastrophic collapse of pillars
2. Roof or rib fall

Both the core risk assessment and the subsequent risk assessment on gas and ventilation, suggested that one option was to form wider roadways – 6.2m. These were seen as offering larger cross-sectional areas for ventilation and wider spaces between mining equipment. Approval for wider roadways has not yet been sought. Risks associated with the movement of mobile equipment in confined spaces was addressed in a separate risk assessment on crush injuries.

The strata control risk assessment identified the following hazards:

1. Fall of previously supported roof
2. Fall of scat between supports
3. Struck by rock while scaling
4. Struck by rib

It is significant that thin seam and ultra close mining were considered not to introduce additional geotechnical risks compared to ‘normal’ seams.

**Business Risks – Ultra-Close Mining**

In the late 1960’s or early 1970’s a number of physical models were developed by ACIRL. The models showed the onset of floor heave when Balgownie roadways were located under and immediately adjacent to Bulli Seam pillars. For the case of the roadways under the Bulli pillars, a tensile failure developed in the roof. For roadways adjacent to the pillar, shear and tension failures developed in the roof. The nature of the shear failures in these physical models is consistent with the observations made by the earlier miners in the Balgownie Seam. Finite element modelling has been conducted to confirm the physical models. These models predict roof failures and floor heave of similar nature to those seen in the physical models.

Based on the modelling, a review of US work, and the experiences of TSM, a number of failure mechanism are possible, depending on the interaction of the stresses with the rock types. Beyond approximately 70 metres of cover, remnant pillars in the Bulli seam and goaf edges may impact on the strata conditions in the Balgownie seam. Roadways offset from any overlying pillar remnants may experience floor heave and roof falls. As remnant pillars are approached, there will be shear failure in the roof, rib spall and floor heave. Directly under any pillars, mining conditions may be better, with rib crush and possibly the opening of joints in the roof. Roadways aligned parallel to any joint sets and under remnant pillars may suffer from roof tensile failures.

The mining strategy that has been adopted is that once an area associated with a remnant pillar is recognised, the number of roadways in the adverse area will be minimised. No roadways will be formed in these areas if roof joints are present and sub-parallel to the roadways. Because the pillar stresses needed to create the poor ground conditions have to be high, the area that will be affected will be relatively small. It should be noted that to get high stresses, small isolated Bulli Seam pillars must be present.
MANAGEMENT

Some of the management issues are discussed below.

Layout

The mine layout was developed to maximise inherent roof performance and to prevent catastrophic pillar failure. The layout is based on aligning roadway so that they are not parallel to the regional joint sets in the interburden. In addition, the layout is flexible so that it can change when remnant pillars are encountered.

Pillar sizes have been set on the following guidelines:

1. Production panels with a factor of safety of 1.6,
2. Interpanel pillars with a factor of safety of 2.0 assuming full extraction of the panels
3. Main pillars with a factor of safety of 2.0

Pillar stresses are based on overburden load assuming the Bulli Seam has not been extracted. This means that the loads will be underestimated when under remnant pillars in the Bulli Seam. The risk that this introduces in terms of pillar performance is judged to be acceptable because:

1. There is no intention to form pillars in such ground
2. The location of the remnant pillars in the goaf is not known so no practical alternative strategy is available.

Width to height ratios of the pillars are all greater than 10, so it is likely, that should any pillar failure develop, that the result will not be the shedding of load to other pillars, but they will fail in a plastic manner. This means that any failed pillars will remain load bearing after failure.

Hazards And TARPS

The geotechnical hazards for Gibson’s are the same as for all underground coal mining – unpredicted roof geology, poor support installation, faults - plus the business risk of ultra-close mining.

Trigger Action Response Plans (TARPS) have been developed for the range of strata control variables anticipated at Gibson’s Colliery. The TARPS require operators to respond to conditions that are not normal, and to report variations. That has required Gibson’s Colliery to train its operators in the range of variables anticipated.

Training has included operators walking roadways with a geotechnical engineer, and witnessing first hand some of the conditions experienced by operators of the past. This has been possible because Gibson’s Colliery has utilised several hundred metres of roadways that were developed between 30 and 80 years ago.

Managers Support Rules are considered support standards. Philosophically, the need for operators to “install additional support if considered necessary” is seen as a departure from the support standards and must be reported. This action is encouraged and warrants further investigation as the support standard may be deficient, and require amendment.

Each shift, operators are required to drill a test hole to confirm the nature of the roof strata, and to confirm that the required pre-tension is achieved by the support system. Regular testing of the distance to the Bulli Seam is to be included in the mining sequence.

Roof Support Standards

Flat Roof - Typical Condition

The key to roof support design in cut and flit pillar panels is to recognise the following three points:

1. The roof will fall to a unit that is in excess of say 20cm – 25cm thick
2. The stress changes related to the cutting of the place will fully develop within 5m of the face
3. There will be no subsequent stress changes.
With a demonstrably stable roof, either as-cut or after any fall, and no further stress changes, the question of roof bolt design is what is the role of roof bolts? 

The observational evidence from the 1970’s is that roof falls developed if the distance to the first major parting in the roof was less than 0.6m. We have used this as the basis of a roof support strategy based on the suspension of a rock layer of this thickness. This leads to a light pattern of short bolts. 

**Roof Support Associated With Remnant Pillars**

Based on the reported mining conditions elsewhere in the Balgownie Seam, roof falls can be expected below remnant pillars. It is noted that some of these falls may be delayed that is, not occur at the face itself. In areas of open cracks, indicative of tensile conditions, there is a risk of the roof unravelling to a significant height and possibly interconnecting to the Bulli Seam. In this situation, forcing ventilation minimises the risk of gas inflow to Gibson’s Colliery.

It is assessed that a long tendon support will be required in these areas – based on cables or coupled bolts. The design should be based on dead weight suspension. The critical design parameter for this support is the likely height of the fall – this controls the length of the tendon as well as the load capacity of the system. The likely fall height is not known but it has been assumed at this stage that it is 3m. if triangular shape and 2m if a flat fall block. This gives a dead weight 22.5 tonnes - 30 tonnes per metre of roadway respectively. Note that the key unknown is the height of the fall. Until some experience is gained, it is recommended that any suspect ground be secured, the mining face relocated, and a geotechnical assessment made.

**Rib Support**

Given the thin seam of 1.2m and especially when wide roadways are adopted, pattern rib bolting should not be required. Some spot bolting around join and fault zones may be required. 

### PROGRESS TO JANUARY 2002

The portal was developed immediately behind the main office building at Russel Vale. The strata consisted of approximately 5m of colluvium, overlying an additional 10m-15m of weathered rock. The intensity of weathering decreased with depth: immediately beneath the colluvium the weathered coal measure strata was of lower strength. The initial roadway intercepted the brick portal structures of the old Gibson’s Tunnel. These brick arches were filled with cementitious products supplied by Foscroc via Ventmine, with the grout preventing the arch from collapsing as steel sets were positioned. The installation of this grout exceeded our expectations and not only filled the voids but also penetrated into the fill above the brick arch.

The shallow workings are characterised by low horizontal stresses. Low horizontal stresses introduce the risk of joint-controlled failure with roof falls developing by slipping on vertical joints. The mining layout response was to seek a minimisation of the number of roadways parallel to the joint sets in the roof. When joint zones were intersected in areas of low confinement, the response was angled cables installed through the joints and over the coal solids. A persistent decomposed zone was identified and in that zone plunge lengths were reduced and straps and cable bolts were installed, as planned.

Up to 3 cutthrough, the roadways were formed at 2.0m high. In the first week of thin seam operations, the 2nd week of January 2002, Gibson’s Colliery achieved a 70 metre and an 80 metre shift. In the second week, several shifts in excess of 100m were achieved. Those shifts were 9 hours in length, and several hours were lost due to delays experienced while commissioning the equipment.

### ACKNOWLEDGEMENTS

To Dr. Ross Seedsman of Seedsman Geotechnics Pty Ltd. His conviction and foresight have provided Gibson’s Colliery with a cost effective and credible strata management system for the application.

To Tim Gaudry and Peter Craig of Jenmar Australia for their participation in the design and testing of strata support systems to date.

Graham (Spot) White of the CFMEU, for his participation during enterprise negotiations, and for genuinely welcoming our American colleagues into the Wollongong community.
<table>
<thead>
<tr>
<th>DEFINITION</th>
<th>NORMAL</th>
<th>LEVEL 1A TRIGGER</th>
<th>LEVEL 1B TRIGGER</th>
<th>LEVEL 1C TRIGGER</th>
<th>LEVEL 2 TRIGGER</th>
</tr>
</thead>
<tbody>
<tr>
<td>Extended plunge (more than 6m) is judged to be possible. Flat roof or falls less than 0.5m high No or very little rib spall</td>
<td>Cut and bolt installation Plunge more than 0.5m over width Less than full length not due to ground conditions Bolt installation is not to design requirements Bolts not taking torque</td>
<td>Remnant pillars Open vertical cracks in the roof Open angled cracks in roof Flush of water in the plunge Excessive rib spall Visual signs of weighting including: Noise</td>
<td>Strutcre Faults, dykes, joint zones encountered. Visual roof deformation such cracking developing along rock defects More than 0.3m throw fault Roof falls more than 0.5m high</td>
<td>Strata aleret Cannot set ATRS to roof of fall Inability to adequately support with available support standards Miner or Fletcher buried or cannot be withdrawn under its own tractive effort</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>REFERENCE</th>
<th>PINCH OUTS</th>
<th>HIGH STRESS</th>
<th>JOINTED ROOF</th>
<th>REFERENCE</th>
</tr>
</thead>
<tbody>
<tr>
<td>SLABBY ROOF</td>
<td>RIB CONDITION 0 - 1</td>
<td>RIB CONDITION 3-5</td>
<td>FAULT</td>
<td>PINCH OUTS</td>
</tr>
</tbody>
</table>

| POSSIBLE INTERPRETATION | Immediate sandstone roof beam No remnant pillars No faults or joint zones Incorrect or not maintained equipment Incorrect materials Broken ground Support standards not followed Operators not competent | Under influence of remnant pillars Fault zone Joint zone and low stresses | Geology more complicated than anticipated by geotechnical engineers Complex roof and remnant pillar interaction |

| ALL PEOPLE | Continue production and conduct work to sequence and support standards Awareness of Strata Domain Plan Review installation requirements and check supplies Install additional support to standards or stop. Inform coordinator and Mine Statutory Official During excavation minimise size and time of roof exposure | Inform coordinator and Mine Statutory Official Continue production During excavation minimise size and time of roof exposure | Stop production. Withdraw equipment. Inform coordinator and Mine Statutory Official Install supplemental support as directed. Withdraw miner No—road area Advise coordinator and Mine Statutory Official |

| MINER DRIVER | Cut to width and length as set in support standards Record plunge conditions on sheet Return to design width and length of plunge Record width and length on sheet Continue production Stop plunge Record width and length Relocate miner Await instructions from officials | Reduce plunge to allow satisfactory roof for bolting Record width and length await instructions from officials | Install supplemental support as directed. Withdraw miner No—road area Advise coordinator and Mine Statutory Official |

The AusIMM Illawarra Branch 6-8 February 2002
<table>
<thead>
<tr>
<th><strong>BOLTER</strong></th>
<th><strong>LEVEL 1A TRIGGER</strong></th>
<th><strong>LEVEL 1B TRIGGER</strong></th>
<th><strong>LEVEL 1C TRIGGER</strong></th>
<th><strong>LEVEL 2 TRIGGER</strong></th>
</tr>
</thead>
<tbody>
<tr>
<td>NORMAL</td>
<td>Cut and bolt installation</td>
<td>Remnant pillars</td>
<td>Strucutre</td>
<td>Strata alert</td>
</tr>
<tr>
<td></td>
<td>Inform coordinator if equipment fault</td>
<td>Support using latest recommendation from strata review team</td>
<td>Support using latest recommendation from strata review team</td>
<td>Withdraw bolter</td>
</tr>
<tr>
<td></td>
<td>Repair equipment immediately</td>
<td>Record on sheet</td>
<td>Record on sheet</td>
<td>No-road area</td>
</tr>
<tr>
<td></td>
<td>Record on strata sheet</td>
<td>Records any time a plunge is left unbolted for more than 4 hours</td>
<td>Install additional support as directed</td>
<td>Notify coordinator and Mine Statutory Official immediately</td>
</tr>
<tr>
<td></td>
<td>Drill test hole at beginning of shift</td>
<td>Conduct torque test</td>
<td>Place and advance no-road signs</td>
<td></td>
</tr>
</tbody>
</table>

**MINE STATUTORY OFFICIAL**

- Continue Statutory and other inspections
- Monitor instrumentation as required by strata review team
- Determine if mining should stop pending repairs/new supplies.
- Record on statutory report
- Install temporary supplemental support if required to ensure immediate stability
- Note location, geological and geotechnical details and remedial action taken on shift or statutory report
- Communicate with oncoming Mine Statutory Official as per mine communication system.
- Record on statutory sheet
- Continue statutory and other inspections
- Install temporary supplemental support if required to ensure immediate stability
- Note location, geological and geotechnical details and remedial action taken on shift or statutory report
- Communicate with oncoming Mine Statutory Official as per mine communication system.
- Record on statutory sheet
- Inspect place with coordinator and determine if mining should continue
- Invoke strata alert if mining does not continue
- Install supplemental support.
- Note location, geological and geotechnical details and remedial action taken on shift report
- Communicate with oncoming Mine Statutory Official as per mine communication system.
- Record on statutory sheet
- Inform SCP
- Monitor extent and development of fall.
- Safety contact with All People
- “No Road” entries to fall area.
- Await advice from mine manager.
- Supervise recovery of fall.
<table>
<thead>
<tr>
<th>Role</th>
<th>Level 1A TRIGGER</th>
<th>Level 1B TRIGGER</th>
<th>Level 1C TRIGGER</th>
<th>Level 2 TRIGGER</th>
</tr>
</thead>
<tbody>
<tr>
<td>COORDINATOR</td>
<td>Cut and bolt installation</td>
<td>Remnant pillars</td>
<td>Strucutre</td>
<td>Strata alert</td>
</tr>
<tr>
<td></td>
<td>Advise Mine Statutory Official</td>
<td>• Relocate production pending advice</td>
<td>• Advise miner driver if mining is to continue</td>
<td>• Arrange for initial support materials and equipment.</td>
</tr>
<tr>
<td></td>
<td>Schedules repairs/new supplies</td>
<td>• Inform SCP</td>
<td>• Communicate with oncoming Mine Statutory Official as per mine communication system.</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>MINE MANAGER</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Weekly mine-wide audit</td>
<td>Inspect strata conditions as soon as practical</td>
<td>Weekly mine-wide audit</td>
<td></td>
</tr>
<tr>
<td>(If off-site, may be</td>
<td>Authorises cut sequences</td>
<td>Organise and manage installation of supplemental support to ensure immediate</td>
<td>Authorises cut sequences</td>
<td>Inspect fall.</td>
</tr>
<tr>
<td>delegated to person he</td>
<td>Reviews statutory reports on a weekly basis</td>
<td>stability</td>
<td>Reviews statutory reports on a weekly basis</td>
<td>Coordinate and organise recovery of fall.</td>
</tr>
<tr>
<td>deems competent)</td>
<td></td>
<td>Review situation and action required with Geotechnical Engineer</td>
<td></td>
<td>Implement fall recovery operation.</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Discuss production plan with GMM – consider abandoning roadway if not beltway.</td>
<td></td>
<td>Monitor and adjust plan in cooperation with GMM and Geotechnical Engineer.</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Review geotechnical report of fall.</td>
</tr>
<tr>
<td>SURFACE COMPETENT PERSON</td>
<td>Transfer health and safety issues to shift notes</td>
<td>Contact mine manager</td>
<td>Transfer health and safety issues to shift notes</td>
<td>Approve and implement recommendations</td>
</tr>
<tr>
<td>GEOTECHNICAL ENGINEER</td>
<td>Transfer health and safety issues to shift notes</td>
<td>Transfer health and safety issues to shift notes</td>
<td></td>
<td>Notify Inspectorate if applicable</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>GENERAL MINE MANAGER</td>
<td>Update panel geotechnical profile and provide advice to Mine Manager.</td>
<td>Update panel geotechnical profile and provide advice to Mine Manager.</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Review production plan belt road</td>
<td></td>
<td></td>
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</table>
STRATA MANAGEMENT AT THE GOONYELLA EXPLORATION ADIT PROJECT

Ross Seedsman¹, Peter Brisbane² and Guy Mitchell³

ABSTRACT: The Goonyella Exploration Adit is a three heading development into the highwall at Ramp 4 at Goonyella Riverside Mine that was driven between September 1999 and November 2000. The principal was BHP Billiton Mitsubishi Alliance (BMA) and the contractor was Allied Mining Australia. Strata management began at the pre-tender phase and evolved as greater knowledge of the ground became available.

Roadways were driven 3.6m high, 5m wide in the 7m – 8m thick Middle Goonyella Seam. The cut and flit mining system was used with extended cuts of 12m to 15m and a minimum support pattern of 4 x 1.5m tensioned bolts at 1.5m centres. Extended cuts were possible, even at the maximum depth (290m), so long as an adequate thickness of massive top coal was left. Horizon control was the basic strata management tool in the project.

Extensive exploration using both drilling and 3-D seismic methods was available and proved to be reliable in detecting faults greater than 3m throw. Systems to identify the proximity to thrust faults during driveage were successful in identifying smaller scale thrust faults.

Small-scale normal faults striking parallel to the roadways were encountered, typically in only one of the three headings, and these impacted substantially on the mining rates. They were not detectable from any exploration program. The term trench roof was adopted to describe the associated roof falls – less than 2m wide, up to 1.5m high and bounded by fault or joint planes. The falls were interpreted as being vertical drop-outs from a coal roof under very low horizontal confinement.

TENDER PHASE

In 1998, as the project went to tender, the understanding of the geotechnical regime was based on extensive drilling and 3-D seismsics. The following is a summary of the geotechnical understanding as it stood in 1998.

- The mine plan involves the development of three roadways in the lower part of the seam leaving about 2m of top coal.
- The adit may be in a structurally complex area, as evidenced by a rotation of the strike of the seam by 45° and the identification of two large thrust faults.
- There is the possibility of numerous small-scale faults aligned across the direction of the adit.
- Driveage is to be subparallel to a mapped normal fault and one of the joint directions exposed in the pitwall.
- The top coal ply is thickly banded to massive with laboratory unconfined compressive strength (UCS) values greater than 20 MPa.
- A moderately strong stone roof is present above the coal.
- There is the possibility of bedding-parallel shear zones within the coal seam, in addition to those mapped/inferred in the roof and floor.
- The driveage direction is at right angles to the presumed direction of the major principal horizontal stress.
- The major principal horizontal stress in the coal seam is assumed to be lower in magnitude than the vertical stress (0.7 times).

¹ Seedsman Geotechnics Pty Ltd
² BHP Billiton/ BMA
³ BHP Billiton/ BMA
As one of three geotechnical consultants, Seedsman Geotechnics Pty Ltd (SGPL) prepared a report that identified three ground types (Figure 1) based on an approach used in civil engineering. These are as follows:

- **Typical** – the usual conditions associated with mining in an unfaulted coal seam with a coal roof - a ‘minimum’ support rule can be developed prior to mining,
- **Adverse** – mining through ‘known’ geological features such as minor faults – support rules can be formulated based on presumed conditions,
- **Special** – conditions when mining thrust faults or at depth – highly variable conditions that will need to be assessed underground and a support regime developed.

Special conditions were identified beyond chainage 2500m as a result of a combination of thrust zones, higher ground stresses related to the greater depth of cover, proximity of rider seams, higher water inflows and other factors. It was recognised that there could be undetected faults at any depth that would be classified as special ground.

Allied Mining Australia (Allied) was the successful tenderer. SGPL was appointed as the geotechnical engineer to both the principal and the contractor - this strategy would assist in removing the uncertainties of the ground from any contractual issues.

**CHANGE TO CUT AND FLIT**

Further analyses of the ground conditions, and an inspection of Moranbah North Mine, resulted in Allied and SGPL advocating cut and flit systems for roadway development. At the same time, BMA indicated that the roof and rib support designs did not need to consider the stress abutment associated with longwall retreat. These two decisions allowed major changes in the roof support design, and once these were accepted by BMA, the strata hazard management plan was developed and finalised.

The inspection of Moranbah North confirmed the interpretations of the analysis of the behaviour of coal beams. At Moranbah North, the importance of keeping a thick coal roof of about 1.5m had been recognised, as it was at North Goonyella. The Goonyella Middle Seam is comprised of a number of plies. Ply 1, the upper ply, is described as dull high ash coal with common claystone partings and occasional bedding plane shears. Ply 1 is approximately 1m thick. Ply 2 is approximately 2m thick and is described as predominantly dull coal, with few claystone partings.

The top ply is highly banded and experience shows that it is not self-supporting. Ply 2 is a massive coal band and calculations suggested that so long as 0.4m to 0.5m was left uncut it could span across a 5.5m roadway (Figure 2). If such a unit could be formed then an extended cut could be taken and hence allow cut and flit mining. The 1.5m roof coal criterion at Moranbah North was interpreted to be the composite of the 1m Ply 1 acting as a surcharge on a 0.5m structural beam of massive Ply 2 coal.
The strata hazard management plan and its associated response plans were formulated around these four components.

Invalid design

The requirements for roof and rib support, as set by BMA and Allied were:

- Safety on installation and outbye
- Design life in the range of 5 years
- Optimum development rates

The critical design issue was the long-term stability of the coal beam, and particularly if it was available at depths in excess of those at the adjacent mines. An integral part to the plan was to validate as soon as possible the two critical input parameters – coal strength and horizontal stress.

Testing of the coal in Ply 2 in both the horizontal and vertical directions revealed the importance of incipient cleat in the coal. The design value for unconfined compressive strength of Ply 2 was set at 10 MPa.

Both the vertical and horizontal stress, as measured by overcore methods in the coal roof, were consistently found to be lower than expected. The vertical stress was approximately half that expected on the basis of the depth of cover and the major horizontal stress was approximately 0.45 times the vertical stress. The current model to explain this stress model advocates coal shrinkage as it is dewatered by the mine openings.

It was concluded that the combination of this stress field and the shape of the opening resulted in very low horizontal stresses acting in the immediate coal roof. This provided more confidence in the stability of the plunges and allowed the correct prediction of continued stable plunges in unfaulted ground at the end of the adit at 290m depth.
Variation in geology at the face compared to the design assumptions

Geological uncertainties are inherent in all geotechnical endeavors. This is particularly the case in underground coal mining as the detailed nature of the immediate roof can change significantly and not be identified by the workforce. The installation of roof and rib support requires constant checking that the assumptions on which the support design was based remain valid. This requires that the operators know what to expect and have the knowledge to identify significant deviations from that. A pocket-sized booklet was prepared that described the anticipated conditions and the support standards to be used.

The known possible geological variations that were incorporated in the support designs were closer-spaced bedding in the immediate coal roof and the presence of disrupted ground around thrust faults. The exploration programs had supplied the project with very good information on the coal plies and the presence of the larger thrust faults.

Workings drove into an unidentified thrust fault and into a zone of low throw normal faults both of which resulted in reductions in the planned development rates.

Poor installation

Each bolt plays an important part in the roof and rib support. There was a degree of conservativeness in the support design but this was for the unexpected geological conditions, and not to cover for poor workmanship.

Primary bolt installation was routinely checked and no problems were identified. In common with other operations in Queensland at the same time, it was found that the cuttable rib bolts were being broken on installation due to the high torques that were being used for the roof bolting. Once the problem was identified, a direction was issued that cuttable bolts were not to be used until a torque reduction circuit was installed in the hydraulics of the Fletcher bolter.

Time dependent failure

Roof failure or unacceptable movement outbye is a safety hazard and indicates that the primary support design is invalid. Tell tales were installed throughout in all headings.

![Fig.3 Examples of tell-tale measurements - movements in mm/week](image)

The coal beam analysis gave an allowable movement of 10mm and in unstructured areas this was never exceeded. In the more structured areas, additional movement was recorded and in most cases this reduced to very low rates within ten days (Fig. 3).

In faulted ground, pretensioned cables were used with a design based on dead weight suspension. To achieve design load, it was calculated that the cables would stretch an additional 20mm after installation, so a threshold was set of 10mm for management review, and 20mm for remedial action.

It is noted that there has been one roof fall and numerous rib falls since October 2000. These are still being assessed, but at this stage the interpretation is that they may relate to an increase in vertical stress related to
depressurisation of the coal over such a wide area that the overburden is beginning to reload the coal. The mechanisms used to explain slabbing in metalliferous mines are also being reviewed.

**Strata reviews**

A strata review team was formed that met every three months or more frequently as required. The team consisted of BMA’s project superintendent, planning manager, and geologist, Allied’s mine manager and the geotechnical engineer. The team inspected the underground roadways, reviewed the geological information, and developed support regimes for special ground. The team then issued an Authority to Mine to Allied, which was valid up to a specified chainage.

Typical ground was defined as that part of the driveage having a massive immediate coal roof. To achieve this, horizon control was essential and it was instilled into the work force. They all carried the simple diagram shown in Fig. 4 which was an idea copied from Moranbah North. Since Ply 2 is about 2m thick and only 0.5m had to be left, normal faults with throws of up to 1.0m to 1.5m could be traversed (so long as the fault plane itself was supported).

**STRATA MANAGEMENT IN TYPICAL GROUND**

![Fig. 4 Horizon control for typical ground](image)

Roof bolting is needed as insurance against unknown structure in the immediate roof and future stress changes. With extended cut mining at Goonyella, the bolting design has a number of different features compared to in-place bolting for longwalls:

- The stability of the cut demonstrates that there is no adverse structure such as bedding or faults.
- All the stress changes that occur in the roof develop within about 5 m of the face and hence there are negligible stress changes after the bolts are installed.
- BMA had directed that longwall abutments did not have to be addressed.

To specify the bolting, a pragmatic design based on preventing time-dependent tensile delamination of the coal was adopted. In addition, the demonstrated stability of the plunge was utilised to move from straps to spot bolts. The support regime was:

- Four 1.5m long X type bolts per ring, evenly spaced across the roadway
- pretensioned to a minimum of ten tonnes
- chemical point anchored
- installed vertically or angled away from the centreline
- 1.5m ring spacing
- 400mm butterfly plates or equivalent
- either closed up to 1m spacing or increased to six bolts/strap in intersections
- maximum unsupported time for roadways – two days
EXPERIENCES IN SPECIAL GROUND

The project team knew of the experiences with faults at North Goonyella, and were fully focused on the possible impact of the known and unknown faults at Goonyella. The 3-D seismics had identified a number of large thrust faults well inbye of the portal (Figure 1) and the strategy was to mine up to them and develop support strategies after some experience had been gained with the seam. The large thrust faults were traversed without problems, because of the lessons learnt from a fall on a minor thrust fault outbye.

The geology that gave greater problems was a series of small scale normal faults.

Thrust faults

The first thrust fault was encountered at 28 cutthrough in C heading. This had a throw of about 0.3m and had not been detected in the seismic exploration or drilling. With hindsight, there was very strong evidence of its presence in a cored hole and, significantly, not in the geophysical logs. The reliance on geophysical logs (and particularly the picking of spikes in sonic logs) to identify slickensided zones was reduced.

Back analyses of the roof conditions as the thrust faults were approached from the footwall (downthrown) side allowed the formulation of a predictive model for the proximity to such faults (Fig. 5). The shallow, flat-topped roof falls began about 20m - 30m prior to the thrust plane being intersected. This model successfully identified all subsequent thrust faults and complemented the surface exploration: 3D seismic survey were invaluable as a planning tool and was found to be accurate for faults greater than 3m, but less than 50% for smaller throws for which the observational model was essential.

The roof support pattern was altered once any of these precursors were observed or when thrust faults predicted from the seismic exploration were approached. The major concern was that the development roadway did not intersect the fault plane before long tendon support had been installed. In such circumstances, experience had shown that a large roof fall could develop outbye.

A design approach based on the dead-weight suspension of a fall mass was used. The fall height was estimated from the geology (thrust planes followed coal riders), and the tendon length based on the ability to anchor in low strength strata above the riders.

The sequence that was used to mine through these faults involved:

1. progressively taking 1m cuts with the remote controlled miner and installation of 2.1m roof bolts (using hand-held bolters) until a 5m cutout was formed,
2. flitting of miner,
3. installation of two angled tensioned cables per metre using Fletcher bolter,
4. flitting of bolter.

Fig. 5 Roof conditions on the footwall of thrust faults

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4. flitting of bolter.
The sequence resulted in a relatively slow development rate. It was recognised that there were opportunities to increase cut-out distances, and/or to delay changing to this sequence but these were not adopted.

At the request of BMA, the initial cable design was conservative. The design assumptions and functional restraints led to a specification of 150 tonne/m of support for an 8m fall block. The fall height was based on the assumption that the rider seam and its immediate roof could be involved in any fall. Anchorage was available in the 8-10m horizon vertically above the roadway, or with 6m long angled cables anchored in 1m of sandstone or 4m of coal. This specification was later reduced to 63 tonnes/metre, which was implemented with 2 x 50 tonne cables every 1.6m - say 2 every 1.5m. In some cases, a strong sandstone unit was available within the anchorage horizon and this allowed the cable lengths to be reduced.

**Horst/Grabens**

At around 21 c/t in A heading, and particularly in C heading inbye of 29 c/t, the driveage encountered what became referred to as ‘trench roof’. This coincided with the presence of small-scale normal faults in a horst/graben configuration (Fig 6). The headings were sub-parallel to the strike of the horst/grabens.

![Fig. 6 Face mapping of horst/grabens](image)
Roof falls developed soon after the start of each plunge (Fig. 7). The falls involved the collapse of joint/fault-bounded blocks of coal from the roof. These blocks were often in the order of 300 mm to 500 mm wide and resulted in the formation of a trench in the roof. There was no rock noise associated with the falls. The longer the plunge, the higher were the roof falls—the maximum fall height recorded was about 1.5m and this happened when the plunge was extended to 5m. Trench roof was only recorded in the headings, not in the cutthroughs. The falls occur as joint-bound trenches over about half of the roadway. The fall zone migrated from the north rib to the south rib of each roadway and in this way migrated from A heading to C heading.

The falls were interpreted as being the result of low horizontal stresses acting on wedges of coal defined by the horst grabens. The low horizontal stresses resulted from the interplay of the roadway shape with a low magnitude stress field where the major principal stress is vertical. Simple wedge analysis indicated that falls were likely for discontinuities dipping at between 70° and 65°, and inevitable for surfaces dipping at less than 65° (Figure 8). The height of the falls was controlled by the onset of a more compressive regime higher in the roof.

The mining strategy was to limit the plunges to 5m, and to support progressively with 2.1m bolts, mesh straps, and butterflies. Once supported, there was no further movement. To restrict the plunge to 5m and to bolt at 1m intervals with hand held bolters was an operational decision based on the significant increase in bolting time if the falls were allowed to extend higher as a result of the plunges being taken deeper. At the same time as C heading was experiencing trench roof, the conditions in both A and B heading allowed 15m plunges. This introduced a substantial imbalance in the panel advance.

The roof support pattern adopted for the trench roof areas was empirically derived. It is focussed more on the control of immediate roof slabs and cantilevers than on the overall stability as the impression gained was that the falls rose to a height of about 1.0m and then stabilized.

More of the faults dipped to the north than to the south. As a result there were more problems with planar slides off the southern ribs—the northern ribs experienced minor toppling failures at the rib/roof corner.

**SECONDARY SUPPORT**

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**Fig. 9 Required anchorage length for cables**
Monitoring identified some areas where the post-installation movement of cables exceeded the 10mm threshold. A review of the ground conditions identified that these areas were associated with low strength strata in the anchorage horizon. The distribution of overall stone roof strengths was known from sonic logs. A nomogram was developed relating anchorage length to rock strength using standard ground engineering concepts (Fig. 9). From this nomogram came the requirement for 10m long vertical cables to achieve adequate anchorage in some areas. In other areas, higher strength sandstones within the fall horizon could be exploited if angled cables were installed over the pillars.

**RIBS**

The high ribs required support in areas where the spacing of coal joints was less than about 300mm, and also in the horst/graben areas. The support design was based on the assumption of joint defined planar slides. The dip of the joint is an important control on the weight and dimensions of the failure block (Fig. 10). Ten tonne capacity rib bolts gave a factor of safety of two against the design blocks. Depending on the height of installation, bolt lengths of 1.2 to 1.5m were required to ensure anchorage beyond the joint plane.

![Fig. 10 Rib bolt requirements](image)

The specification for spot rib bolts was:

- ten tonne capacity (10 tonnes/metre of rib)
- spacing of 1m or less
- minimum 1.5m length
- installed at no more than 2.5m above the floor
- point-anchored, with plate, nominal tension.

**LEARNINGS**

The close interaction between the mine geologist, the geotechnical engineer, the contractor and the principal allowed the rapid identification and response to changing face conditions. The drilling program and 3D seismic studies provided essential information for planning and scheduling. With hindsight, more emphasis on coring and acoustic logging would have assisted to give a better appreciation of the nature of the ground around thrust faults and to provide data on the orientation of coal structures.

Not every geotechnical feature can be found by surface exploration nor can they be anticipated by the roof support designer. Weekly geological mapping and observations from the workforce allowed the development of models so that the roof and rib could be ‘read’. Simple geotechnical models and explanations (albeit controversial) allowed everyone involved in the project to appreciate the geotechnical constraints on the development rates.
Thick coal seams have a different set of challenges related to low horizontal stresses and the role of structure. The learnings from the adit have been translated into the feasibility study for the longwall and have changed some of the fundamentals of the proposed mine layouts.

ACKNOWLEDGEMENTS

The authors would like to acknowledge the essential roles played by others within BMA and Allied Mining - Matt Cooper (BMA Project Superintendent), Frank Fulham (Allied Project Manager and Registered Mine Manager), and Bob Coutts (BMA Project Geologist).
APPENDIX

PRE-CONFERENCE WORKSHOP

CONTROL OF GAS EMISSIONS AND OUTBURST IN COAL MINES
FOREWORD

Editor’s Comments
I have attempted to accurately summarise the proceedings and in some cases to quote what speakers said. I apologise if I have misquoted anyone.

The day proved to be very interesting and informative. I thank all participants for their frank and open discussion and contributions and especially Bruce Robertson for his untiring acceptance of my request to sum up and conduct the concluding session. The sharing of experience as seen today and at similar gatherings is what will keep the coal industry advancing and solving the technical problems which accompany deeper mining in hard times.

John Hanes
February, 2002
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# Pre-Conference Workshop

Control of Gas Emissions and Outburst in Coal Mines

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RESERVOIR ASSESSMENT
GAS CONTENT MEASUREMENT

Richard Danell

See following pages for Power Point Presentation

Discussion and Questions

Bob Newman, Tahmoor – Can we assume that one lab consistently reports a higher gas content than the other labs?

Richard – There is random variability. On a statistical analysis, some labs report higher gas contents than others. For risk assessments for outbursts, we will make the results known to industry and will involve the wider industry.

Shane Coleman, New Wallsend – Is there a real difference in gas contents determined by the direct slow desorption and the quick crush methods, and if so, is it consistent?

Richard – There is no systematic difference. Quick crush was introduced to improve turn around times. There is a lot of discussion about whether there is more variability seen with the slow desorption technique and there is some question as to when the measurement is finished. We have not conducted a study to compare the two methods.

Ray Williams, GeoGas – We have done many comparisons of the two methods since 1997. The differences are not real but there is some scatter.

Alan Cook, Keiraville Konsultants – Re coal oxidation, how are the samples taken for the oxidation studies? Are they from a purged coal face? I suspect not which causes worry. There is much literature which says you should purge the coal face and I suspect CSIRO is not following the recommended methods.

Richard – This is part of a separate CSIRO study. The samples were collected after core was processed in the normal manner for quick crush and also through an evacuated mill. There were no differences in the results, so we did not take it further.
PERMEABILITY

Rob Jeffrey¹

See following pages for Power Point Presentation

DISCUSSION AND QUESTIONS

Alan Cook – You did not mention the difference between fracture permeability and bulk permeability.

Rob – For field conditions, the volume of the test sees a huge number of fractures. The presence of fractures is important in core samples as a fractured core will yield totally different results from an unfractured core. In reality, it is difficult to take a core sample from fractured coal. Core testing tends to under-represent the permeability. Fracture permeability is very important in reservoir analysis. The effective permeability in a fracture has a tremendous effect on the overall permeability as a very small opening of a fracture increases the permeability significantly. The permeability of a fracture increases as \( w^{2/12} \) where \( w \) = the width of the fracture. The fracture is the dominant flow path.

Anon. – If the reservoir is stimulated by hydraulic fracturing, then the permeability is measured, higher permeability is recorded because of the induced fracture. Sometimes, injection falloff in coal can be deceiving.

Rob – Seeing past the skin or damaged zone around a well can be a problem for short term tests. Short term tests are not as reliable as longer term tests as they do not see as far into the coal seam. David Casey’s group are doing longer tests with 4 to 6 hours of injection and 12 hours of falloff. The longer tests yield better information.

Bruce Robertson, Anglo Coal – Re application of permeability tests to underground drainage, directional permeability is a fracture-driven issue which cannot be resolved with single hole tests, but can be resolved with a well defined interference test. The direction of maximum permeability is critical to designing effective underground drainage.

Rob – There have been a number of interference tests done around Australia now, but I wish there were more as people tend to use them as a benchmark to get an idea of what the directional permeability might be, the ratio between maximum and minimum. Until we get these tests done, we suffer from a lack of data to use in the models.

¹ CSIRO
COAL MICROSCOPY IN GAS PERMEABILITY STUDIES

Lila Gurba¹

See following pages for Power Point Presentation

Discussion and Questions
Patrick Humphreys, Newlands – What is the turnaround time for tests?
Colin – Lila will have to answer this.

Ray Williams – Are there any macro markers?
Colin – I am not sure. Lila might provide more information.

David Titheridge, Tahmoor – I have recently rechecked some areas at Tahmoor in light of Lila’s work. I am beginning to see some differences in the coal. In some areas I can see some 1 mm wide mylonite zones. I have also checked the distribution of carbonate veins. I can see differences but it difficult to to quantify them and is very subjective. I now need to re3view the microscopic studies. At Tahmoor, about 30 cm below the roof, there is a stockwork of shallow dipping carbonate veins. They are best seen in a vitrinite rich layer. This is underlain by a sideritic mudstone. Where there is siderite below the mudstone, the carbonate infill is above the mudstone. The stockwork is generally not seen in the bottom part of the seam.

¹ Colin Ward, University of NSW presenting for Lila Gurba
THE ROLE OF NITROGEN IN GAS CONTENT ASSESSMENT

Andrew Filipowski

See following pages for Power Point Presentation

When gas contents are measured in the BHP Billiton gas laboratory, gas composition is also analysed. These two parameters determine where a sample lies with respect to the threshold. We have found that if N2 is greater than 20%, the gas content will be below the outburst threshold. We hypothesise that Nitrogen can be used as a faster assessment of outburst potential than gas content.

The gas composition of a coal sample changes with time. The proportion of each component relative to other components is transient. This is due to the different rates of desorption of the different component gases. Some gases desorb very quickly, eg the higher hydrocarbons and CO2. CH4 is slower and N2 is the slowest of the coal seam gases. The higher hydrocarbons are generally present in small amounts, but, as they desorb very quickly, often before the coal sample is sealed into its canister, they are often missed on gas analysis. Nitrogen remains in the coal for a long time. CO2 has a great affinity to coal, but N2 is very faithful to coal and desorbs slowly.

In virgin coal, there is typically 1.5% N2. Bulli seam coal of 15 m3/tonne gas content has around 0.2 m3/tonne N2. During the desorption process, most quickly desorbing gases are depleted quickly and the proportions start to change. N2 desorbs very slowly and its volume in the coal remains fairly constant. To visualize the process, imagine a rock island (N2) in the ocean. At high tide (virgin conditions) the rock is almost covered and there is little to be seen. As the tide recedes (desorption proceeds) the rock (N2) starts to be more obvious.

One problem associated with N2 is the N2 produced by oxidation. When a coal sample is collected under ground, the coal is exposed to air and oxygen is available for oxidation. Oxidation produces proportionally excess N2 in the gas mixture, ie from air and desorbed from the coal.

During the first few days of a desorption test, the majority of N2 in the sample container is excess N2 from air. Later, as consecutive samples are taken of the gas, the N2 from air is gradually depleted and there is more N2 desorbed from the coal. The two types of N2 can only be differentiated by isotopic analysis.

A more reliable source for a gas analysis would be through direct sampling of a drainage hole. There is no oxidation of the coal sample. When gas composition is quoted, the stage of gas desorption of the coal sample should also be quoted.

In the Illawarra, the coal seams close to the escarpment have been considerably drained of gas. Further to the west, the gas composition of the coal is close to its original composition, ie 1.5% N2, 97% CH4 and minor CO2. Tahmoor at 90+% CO2 is an exception as it is Metropolitan with its variable CH4/CO2 composition. During drainage, as the CH4 and CO2 desorb, the N2 volume remains fairly stable and its proportion in the total gas mixture increases. High gas content, low N2 proportion: low gas content, high N2 proportion. Where the gas content is very low, N2 can constitute up to 50% of the gas. If N2 is high, the content will be low and outbursts will not be possible.

With gas desorption studies of exploration cores, slow desorption tests can take up to a couple of months. With these, the CH4 desorbed decreases over time while the N2 increases. The graph shows how the gas content of a coal sample can be estimated from the N2 content of the desorbed gas. Where there is a large spread of N2 data for a gas content on the graph, this is due to sample oxidation.

When a gas sample is collected from a desorbing coal sample, the N2 comes from the air in the container plus from the coal. A bag sample of the gas from an underground drainage hole would give a more reliable test and

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1 Coal Geotechnical Services
less scatter of data. If you have a gas composition, you can assess the degree of coal degasification. This is a much more economical method of gas content assessment than desorption testing. It will not replace all content testing, but offers a quicker and less expensive method for infill testing in the mine. It also provides a good reason for not injecting N2 into drainage holes.

**Discussion and Questions**

**Ray Williams, GeoGas** – GeoGas has a 180 degree difference in the interpretation. With lower content coal samples, there is more N2 and CO in the gas mix. We therefore feel most of the N2 is artificial. Other coals, especially some Queensland coals do not show the same relationships with N2. We need to sample some gas drainage boreholes.

**Andrew** – Samples should be taken from Q2 and Q3. With the Q3 sample, we typically get deficient O2 and CO2 and raised CO. We tested some drainage holes and an exploration bore hole. On the graph, we show the results. On day 1 there was 5% N2. Over the next 15 days the N2 dropped when we expected it to rise. This most probably happens because the oxidation process is most intensive in the initial stages, when there is plenty of oxygen in the enclosed bomb air. Thus when the bomb is opened in the lab some time later and the first gas sample is taken for analysis, the nitrogen content in the sample is relatively high (say 5%). This is mainly the air nitrogen. In consecutive samples, with the depletion of air in the bomb, while the gas desorption from coal is still high, the nitrogen content in gas samples drops (to around 1%). Later samples indicate continuous increase in nitrogen content (up to around 50 - 60%), and this is believed to be mainly a coal nitrogen. There are still questions to be answered, but we have a good working hypothesis which holds promise for faster and less expensive assessment of outburst potential.
OUTBURST ASSESSMENT
UNDERSTANDING THE MECHANISMS OF OUTBURSTS USING A MECHANISTIC APPROACH

Xavier Choi

First of all, I would like to acknowledge my colleague Mr Mike Wold. Mike and I have been working together in the last few years on a number of projects related to gas in coal, and on a couple of projects on outbursts which were funded by ACARP with support from BHP and Shell Coal.

The aim of my talk is to try to get a better understanding of the mechanisms of outbursts using a simple mechanistic approach.

I think most of you know what an outburst is and some of you might know very well about the outburst that occurred in Tahmoor. However, in an outburst that occurred in the Ukraine, about 14,500 tonnes of coal were ejected and about 600,000 m³ of gas was released.

One of our main interest is to understand why does an outburst occur.

Let us do a simple experiment. We have two balloons, one filled up with water and the other filled with gas. Since the balloons are stretched to a similar extent, we would expect that the pressure in the gas and the water in the balloons to be very similar. If we increase the pressure, it will cause the balloons to expand and the tension in the membranes of the balloons will also increase. If I squeeze the balloons, both the pressure and the tension in the membranes will change. Perhaps it is a simple way to understand the meaning of poroelastic effect. As coal is a fractured porous medium either fully or partially saturated with water, we would expect a change in stress or pore pressure as a result of mining or gas drainage would induce similar responses in coal. We notice that the balloons do not burst under the current condition (or state) because the membranes are strong enough. In order to cause the balloons to burst, we need something that would provide the triggering mechanism to enable the system to overcome the energy barrier. In this case, the barrier is provided by the strength of the membrane. However, the balloons will burst if we squeeze them hard enough or if we increase the pressure high enough. When the balloons burst, rupture of the balloon will most likely be initiated at the weakest spot in the membrane. Of course I can create a weak spot by poking the balloon with a needle. We may notice that the bursting of the balloon filled with air is much more violent compared to the balloon filled with water because of the fact air is more compressible than water, and can store more energy under the same pressure. Failure at the weak spot removes the energy barrier and allows the gas to expand, causing further failure of the membrane until the system reaches a new equilibrium (or stable) state. However, during an outburst, it may go through a few metastable states before reaching the final equilibrium state. We can also see, from this simple experiment, the interaction between stress, strength and gas. In coal, a similar triggering mechanism for outbursts can be provided by geological structures, sheared coal, fractures or other features that provide the “weak spots”. As coal is a soft rock, the amount of strain energy as a result of elastic deformation can be quite limited. In order for coal to fail in a violent manner such as outbursts (or similar to a rock burst in hard rocks where a large amount of energy can be provided by strain energy during failure because of the stiffness of the rock), other form of energy need to be available, and that is where the compressed gas comes into the picture.

We know that coal will yield (or fail) if the stress in the coal gets high enough. If we keep applying stress to the coal after initial failure, due to destruction of the fabric of the coal, the coal will become weaker and softer. Therefore, during mining or roadway development, as the size of the mine opening increases, because of increase in stress concentration, we would expect the volume of yielded coal to increase with lose of some of its initial strength and stiffness.

During an outburst, not only that the coal need to have failed such that the energy barrier is removed or overcome, but there must also be compressed gas to provide the energy to drive the outburst. But, where does the gas come from?
We know that most of the gas in coal exists in the adsorbed state on the surface of the coal in the micropores, and we normally express the amount of gas adsorbed in terms of gas content. Based on laboratory observations, it is generally believed that weak (van der Waals) force exists between the gas molecules in the free and adsorbed states, adsorption or desorption can occur almost instantaneously to reach new thermodynamic equilibrium as a result of changes in partial pressure.

We have developed and used a coupled geomechanical-reservoir model to model outbursts. Under certain conditions, the model predicted that outburst will occur when we have reached a certain stage of mining. In some recent studies, we have tried to find out what are the effects of gas composition (percentage of CO2 and methane) on outbursts. We have conducted some runs to study the effects of different desorption rates using the conventional assumption that the rate of desorption is governed by Fickian diffusion, that is, rate of diffusion for a given coal is mainly controlled by particle size and gas concentration gradient. It was observed that, even by changing the desorption time constant from a few days to the order of minutes, less than 10% difference in pressure and pressure gradient around the face was observed. As this may not be able to account for the difference in CO2 and methane outbursts observed in the field, a different mechanism is proposed. To understand this, we may have to look at what happens during mining. Through borehole or face drainage, the region near the face will become partially saturated with water after gas has started to desorb. But because of surface tension (or capillary) effect, if we look at the relative permeability curves for gas and water in coal, we can see that only a small portion of water can be drained. For the case shown here, it is about 10% because the effective permeability (or relative permeability factor) to water is zero at about 90% water saturation. Considering that the porosity of coal is about 2%, the volume of free gas in the cleats and larger voids (secondary porosity system) would at most be about 0.2%, and a higher desorption rate will not have any effect in increasing the maximum amount of free gas. However, as observed in the model studies and mentioned earlier, desorption rate does have some effect on the pressure and pressure gradient around the face.

To understand the difference between CO2 and methane outbursts, it may be important to find out what happens after an outburst has been initiated and the coal has started to break up into smaller fragments. We can see from the model that the coal deforms at a high strain rate after outburst initiation. As the coal continues to expand and disintegrate into smaller fragments, new surfaces will be formed. Pressure around the new surfaces and in the voids, which are close and connected to the new surface, will drop very quickly. As mentioned earlier, gas is physically adsorbed on the coal surface, and the adsorbed gas can be released quickly when partial pressure is reduced. Based on this, it may perhaps be important to look at the shape of the sorption isotherms for CO2 and methane and the rate of change in gas content with respect to change in partial pressure. We can see that, for the range of pressures generally encountered, the slope for CO2 is at least a few times greater than that for methane. This may perhaps explain why CO2 outbursts are generally more violent compared to methane outbursts.

To summarise, we know that the initiation of outbursts can be controlled by a number of factors. We may be able to understand each individual process or factor to some depth, but to understand how they interact, especially under reasonably complex conditions encountered in some mines, can be very difficult, and that is one of the reasons why we may need to use numerical modelling. Also, to understand outbursts, we may need to treat rock/coal/gas as a system, and to understand, as a result of mining, how changes in boundary and field conditions can affect the behaviour of such a system. It should be noted that the boundary of the system is continually changing as mining progresses.

We are currently trying to understand the mechanisms of outbursts better by using a new model that we have developed recently. We would like to model each process as closely as we can as our understanding of the main processes and factors improves.

By using of a single parameter such as gas content and ignoring all the other factors for outburst prediction will likely produce a fair bit of scatter as we can observe from the data collected by Ripu. However, we know that if we can remove enough gas so that there is not enough energy to drive an outburst even in the weakest coal, we should be quite safe (however, it does not mean quasi-static type failure due to yielding of coal will not occur). However, we may have problem in maintaining production when mining through coal with high gas content and low permeability. Perhaps, after we have gained enough understanding of the mechanisms, we may be able to devise other ways of mining through structures and low permeability coal in addition to gas drainage. We may perhaps be able to mine in such a way that we can allow the coal to fail in a predictable and controlled manner, and perhaps even making use of the energy stored in the gas in breaking up the coal in a beneficial way.
THRESHOLDS AND COMPLIANCE ISSUES

Chris Harvey

The ideas presented today will be the ideas of the author and not necessarily those of the Department of Mineral Resources. The prime role of the rules and regulations is to protect people. An outburst is the violent ejection of coal and gas. Also required are a steep pressure gradient, high quantity of gas, high rock pressure and low coal. Lower permeability also plays a role but it is partly determined by the other factors. See Figure 1.

Lama in 1991 and also in the International Symposium on Outbursts, 1995, referred to desorbable gas content, sheared coal and non-sheared coal and suggested threshold values for safe mining. Based on Lama’s work, threshold values were imposed under a Section 63 notice of the Coal Mines Regulation Act. They were imposed on each mine by the District Inspector for that mine, not by the Chief Inspector. This purposefully left an avenue for review by the Chief Inspector under section 65 and provided a potential to alter gas thresholds. This eventually lead to some mines having different threshold values from other mines.

Today, we use total gas content whereas Ripu’s suggestions were based on desorbable gas. The focus is on mining or no mining based on gas content. The reason Ripu focused on gas was that it could be measured reasonably easily with a standard test. The quick crush method gives a fairly quick turnaround of information compared with the older desorption technique.

It is important to identify the potential for a risk such as outbursts to be identified before anything can be done to manage the risk. Gas threshold values are used in conjunction with outburst management plans. The OBMP’s provide a structured approach with key elements to manage the outburst risk. The Department, in order to gain some standardization or consistency in OBMP’s developed OBMP guidelines. Figure 2 summarises the procedures. Outburst prediction identifies the key factors which give rise to an outburst. The main component of outburst prevention that has been identified for and successfully applied in the Bulli seam is gas drainage.

The two key components of the OBMP system (Figure 2) are “audit” and “corrective action and review”. Without these, there is no guarantee that the permit to mine is adequate and will lead to safe mining. The audit approach is the most important fundamental because it determines that you do what you plan to do. Figure 3 shows the result of applying this approach to the Bulli seam. A large number of outbursts occurred each year up to 1994 then the District Inspectors imposed the section 63 notices in May 1994. Since then there have been 7 relatively minor outbursts. Although this is a good outcome, it also represents a problem: ie the outburst problem appears to have been solved.

Outburst risk in the Bulli seam is deemed to be successfully managed through adherence to the threshold values. But the gas thresholds only relate to one aspect of outburst risk, gas content. Gas content thresholds, like any other standard, need to be analysed and reviewed on a regular basis. An understanding of the warning signs at the face is the fundamental final barrier.

Discussion and Questions

Jason Moultrie, North Goonyella – How do you distinguish between minor and major outbursts?

Chris – Mainly on size. Generally the outbursts were less than 100 tonnes. The biggest Bulli seam outburst was 400 tonnes. The minor outbursts have usually been only a shuttle car load and people were not injured or endangered. The two outbursts which occurred on the West Cliff longwall face from undrained coal. There was no structure identified and the burst was driven by CO2. In hindsight, if anyone had been on the tailgate side of the bursts, there would have been a fatality caused by asphyxiation. There was in excess of 12% CO2 in the air. It was regarded as minor as there was little coal dislodged.

Chris Hudson, Moranbah North – With regards to mylonite identified under the microscope, how sensitive is it in a high stress environment for causing outbursts and can it be identified underground?

1 NSW DMR
Wayne Green, Helensburgh Coal – Outburst management success in the Bulli seam is the healthy safety factor with gas content. It is difficult to go much further on a lot of the outburst parameters and put them into a threshold. The reason for continuing research towards a fundamental understanding of outbursts. Until we have this understanding, it is how we can clearly account for stress or the other 14 or so parameters suggested by European workers. We need content threshold values. In the factor of safety there is a cost component to productivity and safety. I cannot see experience might not be a good guide. Quantification of structural features is very difficult.

Bruce Robertson, Anglo Coal – After the South Bulli fatality, the industry embraced the suggested outburst management procedures and made people aware of what had to be done, but as the years have passed, a degree of complacency has set in. A lot of people believe the drainage programmes are the most effective means of achieving the end but the other controls have disappeared. They tend to ignore some of the warning signs. There is a need to make miners aware that drainage is not a panacea and that other factors such as warning signs at the face should be re-emphasised in training.

Phillip Eade, BHP Billiton – The integrity of the coal plays a role in outburst initiation and propagation. The mylonite or sheared coal of the Bulli seam varies from a centimetre thickness to over a metre. Its role in an outburst is uncertain other than it has no strength and has a huge surface area for sorption of gas. It probably also has a fast desorption rate. At Leichhardt Colliery in Central Queensland, the majority of outbursts had no visible mylonite associated with them. However, the coal was heavily cleated which reduced the strength of the coal mass. The outbursts ejected the coal as a buckle failure with the cone axis perpendicular to the cleats. The major fatal outburst of 500 tonnes was from partly sheared coal, however a large amount of coal with thick sheared coal had been mined safely without outburst. At Darbrook and Tahmoor where the coal is strong, I feel that mining could be safely conducted at higher gas contents than the current thresholds. However, if a sheared coal zone were intersected at a high gas content, an outburst could occur.

Alan Cook – The Bulli seam is unusual. When mining moves into other seams settings, the Bulli seam experience might not be a good guide. Quantification of structural features is very difficult.
Ray Williams, GeoGas – Rick Davis and Ripu in the early 90’s put out a document seeking consensus that if the gas content was reduced significantly, outbursts would be prevented. This underpinned the KCC outburst management plans and those of other companies that followed Ripu’s lead. The research being conducted now in outburst modeling is good, but as far as the Bulli seam mines are concerned, it is hard to see anything replacing gas content thresholds in outburst management. However, Queensland is different. The Queensland mines mine down dip, so mining conditions change rapidly. We take these changes into account and consider the effects of desorption rate as one condition. With regards to stress, I feel that the gas content threshold could fail when working with high stress. Mt Davy in New Zealand and Ellalong Colliery in NSW both had outbursts when their gas content levels were just above the Bulli seam thresholds. But both mines had high stress conditions. Another issue which needs to be considered with mathematical modeling is the width of safety barriers between the face and any structure. In the Bulli seam, the accepted barrier width based on Ripu’s work is 5 m. But how well does this barrier width apply to Hunter Valley or Queensland conditions where the seam is thicker and other parameters differ? I think this is an area of fundamental research that could produce useful results. With respect to initiating outbursts, until someone invents a gadget that does not rely on gas content and which can be connected to the face to predict an outburst, I think we have to rely on gas content. Developing an alternative has not proven very practical. Our development rates are very high compared with European mines which use other techniques for assessing outburst proneness.

Chris – The gas pressure gradient has also been shown to be important. Whenever we take a core for desorption, should we also be taking a gas pressure measurement? When Ripu proposed his thresholds, he did postulate a gas pressure for each threshold value. So do we also try to measure pressure each time we sample content? Do we also try to measure the strength of the coal? Then by research, do we plug these factors into a mechanism to give a combined outburst risk indicator instead of just relying on gas content?

Rob Jeffrey – The gas content should relate to the gas pressure unless you are way off the isotherm. If you are mining at a rate that the isotherm is tracking the pressure, then measuring the content tells you the pressure.

Bruce Robertson – The pressure gradient near the face is critical, but it is not easy to measure quickly. Whatever is measured, it needs to be reliable. The holy grail of research would be to define precursors and warnings. We have tried microseismic, we have sniffed for strange gases, we currently look at observable features such as changes in micro and macro structures such as cleat and coal behaviour. We need to keep the door open to this type of research as if we could come up with a reliable precursor system, this would give us another barrier. All the other things we have talked about just do not seem practical.
GAS EMISSION MODELLING
COAL MINE DEVELOPMENT GAS EMISSION MODELLING

Miles Slater and Eugene Yurakov ¹

Abstract
The main aims of this paper are to describe:

- The main gas reservoir parameters used in gas reservoir modelling,
- Application of a reservoir simulator, SIMED II, to understand how such numerical simulations facilitate gas emission evaluation and control during gate road development, and
- The incorporation of mine planning and scheduling inputs, along with gas reservoir models into interactive models of gateroad development emissions.

The development emission response (over time) to initial gas reservoir parameters is determined using the SIMED II gas reservoir simulator. An interactive EXCEL spreadsheet development model uses these rib emission decline curves, mining and ventilation parameters and a range of operational variables to provide a mine planning tool for assessment of likely gateroad gas emissions.

Introduction
The sensitivity of underground mining to gas emission is rising with:-

- Planned production rates up to 6 million tonnes per annum,
- Increasing gateroad lengths (more than 4 km),
- Wider longwall faces (up to 350 m),
- Thick seam mining, and
- Deeper coal seams with higher gas content coals.

The requirement to define likely mine gas emission during gateroad development is increasingly important.

The load on mine ventilation correspondingly increases with finite limits on:-

- Practical main fan duty,
- Gas dilution capacity,
- Sustainable intake/return pressure differentials, and
- For long gateroads, the supply of sufficient air to the face circuit.

The excess of likely gas emissions over available ventilation dilution capacity must be controlled through gas drainage or gas capture techniques.

Application to Coal Mine Gas Emission Assessment
Development emission modelling combines gas reservoir properties with mining parameters to facilitate mine planning (Figure 1). The model is based on roadway gas emission decay curves, derived from the gas reservoir simulator SIMED II.

¹ GeoGAS Systems Pty. Ltd.
Fig. 1 Modelling Algorithm with Basic Input Geology and Gas Reservoir Parameters
SIMED II MODELLING

SIMED II is an implicit finite-difference code, developed to describe the flow of gas and water in coal seams. SIMED II is a two phase (gas and water), three-dimensional, multi-component gas, single or dual-porosity reservoir simulator designed to model mixtures of adsorbed gases flow. In this application it is used to model the mine roadway emission. Additionally, the production behaviour of surface wells and in-seam can be history matched or predicted.

The main gas reservoir modelling inputs in approximate order of importance are:-

- Gas content and composition
- Permeability
- Seam thickness
- Gas sorption capacity
- Desorption rate
- Relative permeability
- Porosity
- Coal compressibility
- Pore pressure

**HIGHEST IMPORTANCE**

**LOWEST IMPORTANCE**

Relevant features of the SIMED II gas reservoir simulator are:

- Simulation of the behaviour of coal seam gas reservoirs as dual-porosity1,
- Multi-component gas mixture simulation as combination of gases and water (multi-phase) within the coal seam,
- Provision for multiple coal types within the one simulation, defined by permeability, porosity and desorption isotherm parameters. This is particularly important for dipping seams, where grid elements surrounding, and along, a roadway need to be assigned varying gas content, permeability and pore pressure, and
- Modelling the effect of rib emissions in underground coalmines using a grid block face drainage option. This simulates the gas drainage to atmospheric pressure from a grid block element. If necessary the rib emission can be modified in real-time during the simulation, to mimic continuing mine development.

**EXCEL SPREADSHEET MODEL**

The main output from SIMED II modelling is a rib emission time-decay profile of a defined, static rib. An interactive EXCEL spreadsheet is then used to enable calculation of development gas emission in response to the following mining parameters:

- Panel development rate,
- Panel length,
- Air quantity at the last cut through (intake entry air is calculated),
- Air leakage, a function of roadway and stopping resistance factors and pillar dimensions,
- Number and arrangement of headings (intakes / returns),
- Pillar dimensions,
- Gas emission in terms of CH₄, CO₂ or “seam gas”, and
- Options in permeability, gas content or other basic parameters.

A common scenario might be gate road development down dip, with varying gas content, seam thickness, permeability, pore pressure and gas desorption rate. To facilitate SIMED II modelling it is usual to graph the main mining parameters and assign representative “regions”. For the case in Figure 2, three regions have been defined.

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1 Dual porosity accounts for the micro-permeability of the coal matrix, along with the macro-permeability of the cleat systems. The desorption time constant restricts the rate at which gas can move from the coal matrix to cleat and fracture system (where pressure gradient controls the flow).
The resulting rib emission decay curves for each region’s set of gas reservoir parameters are incorporated into the EXCEL model. When development passes from one region to another (eg from Region A to Region B in Figure 2), a different set of rib emission decay curves is incorporated. The model can then chart the gas emission as the panel is developed.

Fig. 2 Example Gas Reservoir Parameter Profile Along Gateroad

Gas emission in the face area combines rib emission (according to the rib emission decline curves) with emission from cut coal. The maximum emission occurs immediately prior to holing of the cut through. At this time the auxiliary fans are at their highest duty. The emission from cut coal is dictated by the desorption rate characteristics of the coal, adjusted for lump size. Related mining input parameters are (Figure 3):

- Face cutting rate,
- Residence time of cut coal in the face area,
- Mean cut coal lump size,
- Pillar dimensions,
- Roadway driveage rate,
- Shifts per day mining, and
- Days per week mining.
For the panel being modelled, the rib side is divided into pillar size intervals. The rib emission from each interval is defined according to the age of the rib and the associated flow decline curve. At any particular time, intake/return rib emission is defined as the sum of \( \text{CH}_4 / \text{CO}_2 \) emissions from all the pillar size increments up to the last open cut through, plus the gas emission entering the panel.

An example of a basic input and output model table is given in Figure 4. The panel development rate, pillar size and air quantity can be varied and the resulting gas emission and spot gas concentrations calculated.

**Fig. 3 Face Emission**

Output parameters include gas emission (l/s) and/or concentration of CH4, CO2 or “seam” gas:
- Intake at prescribed distances outbye the last cut through,
- Return next cut through from outbye face,
- Return at 1 cut through (total of gas at panel exit),
- Gateroad emission graph, and
- Face emission.
The development emission models typically target the milestones used in ventilation and longwall emission modelling. Where gas emissions are in excess of achievable ventilation dilution capacity, gas management by drainage or capture methods is required.

CONCLUSIONS

Development gas emission assessment is highly complex, with an extensive range of gas reservoir and mining parameters which need to be considered. GeoGAS Systems’ approach is to model gas reservoir properties using SIMED II and incorporate these, and mining variables, into an interactive EXCEL spreadsheet model of the likely development gas emissions.

The strengths of this approach are the ability to:
- Take into account all important gas reservoir and mining parameters, and
- Quickly get an outcome to facilitate action toward improving ventilation and/or implementation of the gas drainage.
LONGWALL GOAF GAS FLOW MECHANICS

Rao Balusu \(^1\)

See following pages for Power Point Presentation

Discussion and Questions
Phillip Eade, BHP Billiton – How do you simulate the effects of a dyke?
Rao – Winton Gale did the caving model around a dyke. This was used as input. Otherwise we used field calibration.

\(^1\) CSIRO
DRILLING TECHNOLOGY
- WHERE NEXT?
SURFACE TO INSEAM

Frans Bos ¹

See following pages for Power Point Presentation

We need to get rid of the gas before we start mining. Drilling from underground is expensive, cumbersome and somewhat dangerous. To seek an alternative, we have turned to the coal bed methane industry who seem to make money out of gas drainage instead of putting more money in. Why can we not do the same in coal mining? The CBM techniques do not guarantee even drainage of whole mining blocks, so something better is needed. Because of this, there has been considerable research and experimentation conducted to improve the application of the technology. We have experimented with tight radius drilling at German Creek and medium radius drilling at Moranbah North. The presentation will describe the methods tried and will give some ideas as to what might be done in the future.

Medium radius drilling is successful, but still expensive. Drill rig manufacturers should be encouraged to build rigs for the purpose of MRD. MRD will provide good exploration data with 9 holes sufficient to cover and hopefully predrain a 4 km block. If drilling were conducted up to 3 longwall blocks in advance, there should be sufficient time for drainage, good exploration data can be collected and near pure gas can be collected for sale. TRD could be as good as MRD if directional control while drilling can be obtained.

See Appendix A3 for PowerPoint presentation summary.

Discussion and Questions

Jason Moultrie, North Goonyella – Have you taken any test cores to check if the gas has been drained?

Frans – At Moranbah North, the 7 holes were drilled close to existing workings which changed the groundwater conditions. The permeability is high. Only one hole made much gas. The other holes were probably in pre-drained coal. The main purpose of the experiment was to prove we could successfully drill and link with a vertical hole and get some gas production.

Rob Jeffrey, CSIRO – How does the cost compare with conventional core drilling?

Frans – The cost is many times more. The metreage rate is not much more expensive, but the extras such as survey (by a third party) add on costs. The total cost would be at least 5 times the cost of normal core drilling. I believe it should be much less. Drilling in coal is very fast. A big drill rig with a tall mast is required. Around 1500 m hole length is achievable.

Rob – With any technique such as this, there will have to be a set spacing at which the holes will have to be drilled to permit adequate drainage. Modelling and experience will be required to compare the economics of this process with the economics of other methods.

Frans – Underground, drainage is limited by access and there is usually a maximum of one year lead time. If MRD allows 3 years lead time, hole spacing could be greater than underground. But check holes would still be required to determine the success of drainage.

Bruce Robertson, Anglo Coal – The indications are that both TRD and MRD are currently competitive with in-seam drilling on a $/m2 basis, based on MRD spacing of 60-80 m and TRD drilling 200 m radius. TRD is more intensive. MRD gives good exploration data. When somebody makes these techniques commercially available. Cost will probably reduce.

Ray Williams, GeoGas – If longer times are made available for drainage from surface holes, the spacing could be increased thus decreasing the costs.

¹ Anglo Coal
IN-SEAM DRILLING

John Hanes

Drill rigs have undergone some changes in the last 10 years for the better. They have been increased in power and maneuverability for drilling longer holes. Hole surveying has advanced considerably over the last 10 years. Currently the Advanced Mining Technologies’ survey tool, the DDM is the norm. A profiler or indicator of proximity to roof/floor is required by industry. Sigra’s torque-thrust tool should also be a useful add-on when its output can be interpreted.

Guided drilling is mainly conducted as “flip-flop” drilling: 6 m is drilled to the right then 6 m to the left. Gray, 1998, advised that the main limit to hole length was the strength of the drill rod joints under tension during pulling of the rod string. Hole friction was a major factor. To reduce hole friction, he recommended that holes be drilled straighter using the oilfield technique of rotary-slide. This technique has been routinely used at Appin Colliery since early 1988.

Conventional rotary holes are still used, mainly for infill drilling and gas content core collection. Although BHP developed a monitored ProRam drill rig (Danell, 1999) and showed that it could detect outburst prone structures, the drill has not been used for detecting structures ahead of advancing faces, nor have other drills been fitted with monitoring equipment. This is a case of good research ignored by industry.

Drilling contractors and mine drillers are continually reviewing drilling methods to improve their methodologies and to reduce risks.

The information which is gained from any in-seam hole is still completely dependent on the vigilance of the driller. An experienced and dedicated driller can detect even small structures through minor changes in drilling characteristics. The mine can only use this information if it is accurately recorded then properly interpreted. The Australian coal industry now has many experienced in-seam drillers. Tahmoor Colliery and BHP-Billiton’s South Coast mines employ their own drillers and equipment for most of their drilling requirements. Other mines use drilling contractors. The industry is well serviced, but better communication appears to be required to improve results and satisfaction.

When most drilling was conducted with rotary drilling, structures were detected by bogging of the rods. With downhole motor drilling, the more powerful drill rigs allow the drilling through smaller structures without bogging. A vigilant driller is required to detect these zones. The industry still requires automated logging during drilling to detect structures. The brightest hope comes from the Sigra torque and thrust tool which is being developed under ACARP funding (project C7023). In laboratory tests, it has successfully detected minor differences in coal strength during drilling. Another potential is the borehole dielectric probe (Murray et al, 1999) which detects changes in moisture in the coal. To get such tools into the mines will require mine support for initial field proving then financial support for the construction and approvals. Mining personnel need to champion research projects or progress will be slow.

Although drilling technology has advanced, sticky drilling zones still challenge the best of drillers. These zones are typically associated with geological structures or stress concentrations. When the drill bit enters such a zone, the rods, through an uncertain mechanism, become stuck. Perhaps the stressed coal tightens about the bit and motor. Perhaps the soft coal caves and before being fully cleared by the circulating water, blocks the hole behind the motor. The result is hundreds of thousands of dollars worth of equipment left down the hole. Recovery attempts are time consuming and costly. If the equipment cannot be recovered, it does not take many losses to bankrupt a drilling contractor. Tower Colliery has a few such drill rod graveyards. The Sigra borehole pressurization system, developed under ACARP funding (Gray, 1998) and currently seeking a trial site for proving, offers a possible solution to drilling through sticky zones. It allows drilling to be conducted under applied fluid pressure which internally supports the hole wall. This technique is successfully used in surface

Coordinator of ACARP In-seam Drilling and Gas Research
drilling. ACARP are currently funding CMTE research (project C10016) into better methods for drilling through sticky zones and impermeable coal.

When drilling equipment is stuck in the hole, there are a few ways of attempting to recover it. Today, it is accepted that the best method is to overcore the rods and bottom-hole-assembly to free them. Recently, a set of gear was stuck at 960 m and then recovered by overcoring; this was a record.

In-seam drilling has advanced considerably in the last 10 years in respect of the equipment used for drilling and surveying the holes and the expertise of the drillers. There is still much to be done to improve the collection of data from the hole for identification of geological structures and reporting of the data. The development of downhole probes for the detection of structures while drilling has been frustratingly slow.
SURVEYING

Henk Verhoef

See following pages for Power Point Presentation

Discussion and Questions

Paul ?, Powercoal – Exploration issues still need to be addressed. Continuous roof and floor profiling while drilling would be useful.

Henk – AMT are developing a gamma tool which will be initially tested in a highwall. We are awaiting Queensland approvals to do the tests. There are some problems using gamma as a profiler as has been shown by research by CSIRO/CMTE. The gamma sensor can only detect roof or floor changes when the probe is within about 40 cm. CSIRO/CMTE are also working on a borehole radar system. But the solution to the problems is not yet in sight. A probe will have to be measure while drill. Some radar tools appear to work post drilling at this stage. IS approvals represent a major problem with regards to power issues. At this stage, seam profiling is still a dream.

Bruce Robertson, Anglo Coal – There are two developments from overseas which hold promise. They will be prototype tested in oil and gas holes. They are run down the rods and out the end for logging while the rods are pulled. It will be very expensive. Trial sites are required. Coal Bed Concepts are currently testing a RIM borehole radar system.

Phil Eade, BHP Billiton – Gamma horizon sensors are used on miners for horizon control.

Anon – We have provided drilling control for +1500 m holes in-seam based on surface 3D seismic. There are few marker horizons in the 6 m seam. Every fault of +1.5 m throw has been indicated by seismic and picked up by drilling.

Chris Harvey, NSW DMR - Outbursts are being managed from a systems safety approval basis. There is a need for computerized drill hole logging as opposed to relying on the driller. Consistent output which is not subject to human error is required.

Henk – All surveying tools record positional information. The information recorded cannot be deleted. The driller can only change the survey distance from collar. We can develop a system which counts the rods in the hole, but the industry does not want this yet.

Chris – Drillers should also be logging penetration rates and other parameters. What we have at present is too subject to human error.

Henk – To collect such data, a computer system is required. The DGS can do the job, but we are having some difficulties getting approvals. It takes far too much time to get from the research prototype to the finished product because of the approvals process.

Bruce Robertson, Anglo Coal – We have to distinguish between things which are of technical interest and those that are of commercial value.

Brian Lyne, Queensland Mines Dept – I see value in being able to drill flanking holes for the full longwall length of say 4 km. If they were of larger diameter, say 300 mm, they could be used for drainage then for face ventilation to improve existing mine ventilation.

Henk – There is a need to provide drilling data to the drill operator while drilling so he can make adjustments as necessary. Australian Standards limit the Mecca system to 2 km maximum. The Mecca system uses the drill rod as part of the cable. Other methods of data transmission are also restricted, so getting data from a 4 km hole is technically difficult.

1 Advanced Mining Technologies
TOWER COLLIERY DRILLING DIFFICULTIES

Miles Brown¹ and Phillip Eade²

See following pages for Power Point Presentation

Tower colliery has a northerly rending 250 m wide zone which is near impossible to drill or drain. It is associated with a fault which varies between a thrust fault and a bedding plane fault. The coal in the zone contains 15 m³/tonne CH₄. Sub-parallel to the zone are a couple of dykes. The fault lies in a structural low and the coal is thicker here than elsewhere. The area is highly stressed with a prominent horizontal stress which has created enormous roof problems with some intense bolting patterns required (8X8m fully grouted bolts per metre in the maingate roads. Attempts have been made to drill numerous drainage holes through the coal, but with little success. The accompanying figure shows the holes drilled. The zone is outburst prone and two outbursts have occurred while remote mining was conducted to cross the zone. The permeability of the coal is effectively zero in places. There is no gas flow from any holes which penetrate the zone.

The challenge has been how to reduce the gas contents in the zone without having killer delays (eg 5 months to mine one pillar)? Headings should be oriented parallel to maximum horizontal stress to get best heading conditions. Holes drilled on the eastern side of the zone have been drilled parallel to the maximum stress where possible and this seems to have helped considerably. As minor structures were intersected in the holes, we diverted to the roof and crossed the tight zones in the roof. Much drilling in the roof followed by drops into the coal was conducted. The holes drilled parallel to maximum stress produced 80 – 100 lps per hole. Holes drilled close to perpendicular to maximum stress, but of equal length only produced around 20 lps. To handle the difficult zones, the drilling methods of the last 10 years had to be abandoned and new tactics adopted by drilling at 90 degrees to the usual direction.

The coal permeability changes from zero to high in 100 m. Up to 5 branches had to be made per hole to get the required number of holes in the time available. Each hole can produce much water and coal fines. Drilling plans changed frequently with many of the changes made on the job by the drillers. So far there is $800,000 worth of gear stuck in the coal. Across the 134 m zone in one panel, 34 cores were taken for gas content testing.

Discussion and Questions

Marc Justen, South Bulga – Have you tried scroll drilling?
Miles – Yes, but they were not successful. When they hit the zone, they dived to the floor.

Bruce Robertson, Anglo Coal – What characterized the impermeable zone?
Miles – There was much calcite and thick mylonite. There were also dykes. We drilled 45 holes in one face but it refused to drain.

Jason Moultrie, North Goonyella – How did the coal and strata respond to longwall mining?
Miles – There were some major falls on mining. There was an OBMP developed for longwall mining.

John Hanes – Was the sheared coal in the zone soft? Were the ribs hard?
Miles – The coal hardness varied dramatically. Rib lines varied from hard vertical ribs to crushed. Drilling fluctuated from very hard (like rock) to soft. Often it was difficult to see the fault after mining. It looked like two packs of cards forced together overlapping and buckling up.

Anon – Was there any difference in water make from the coal?
Miles – Around the zone had been drained for some time and no water was noted. Variations in water were not noted.

Phillip Eade – The geology has not been quantified, but it appears complex.

¹ Tower Colliery
² BHP Billiton Illawarra Collieries
GAS DRAINAGE AND EXTRACTION
BOREHOLE MAINTENANCE

Bob Newman

Borehole maintenance is a very basic and non-technical subject and in these days of limited resources tends to be treated as a low priority, but if it is ignored it can result in a lot of time and money spent forming boreholes which serve no purpose.

Maintenance is required to fix problems of 3 types - blockages by solid material, water removal and leakage. It should be noted that many such problems can be avoided by applying good standards at the time holes are drilled and connected to the drainage system.

Blockages by solid material:

Effort should be made to avoid them:

- Minimise branches - tends to be a compromise as drillers want as many branches as possible to avoid set-ups and standpipe installation, whereas for gas drainage and monitoring purposes, we prefer none. At Tahmoor we try to limit to 1 branch per hole but are not always successful in this for various reasons. There is a potential for every branch except the last drilled to be blocked by fines from later drilling.
- Thorough flushing at completion of drilling (each branch) is important - when drilling is finished the hole should be flushed for at least 30 minutes, longer if the water is still not clean. Good flushing won't stop material getting into the hole completely - as holes dry out fine material can be released from the walls. Some degree of wall failure often occurs, depending on strength/structure of coal and stress conditions and there may be erosion by water flow.
- Large diameter holes are less likely to block
- Potential blockage sites (such as sharp bends) in the plumbing at the collar of the hole should be minimised (though blockages are easy to clear in this area).
- If possible design holes to be up-dip and try to drill on a steady grade (ie avoid swillies).

Identifying blockages

- Blockages in the plumbing are usually obvious during monitoring because there is no vacuum.
- Blockages at or near the collar of the hole are also usually easy to identify as flow is zero or close to it.
- Blockages elsewhere may be difficult to identify as we have to rely on recognising abnormal flow shown by monitoring. This may be fairly obvious in high permeability coals where flow rates are normally high, but at Tahmoor rates are low (peaks of around 1-2 l/m/min quickly falling to less than 1 l/m/min) and low flows may not be readily apparent. This problem is worsened as we can only afford to monitor, at best, once a week and it is not unusual to miss the peak flow altogether. Adding these factors to normal variations between holes, any partial hole blockage may not be apparent.

Clearing blockages

- In the collar plumbing or close to the collar is relatively easy - just disconnect and clean. It may be necessary to use some type of blow-pipe to clean down the hole, but please take care as the hole may well be under pressure - don't stand in line with it.
- Further down the hole is more difficult. One can try to just wash it out but this is not likely to have much success. Sometimes filling the hole with water then rapidly opening it to vacuum (shock loading) may work.
- The last resort is to bring back a rig and re-drill - a smaller rotary rig may be adequate. However, with branched holes, you're only going to get 1 branch opened, most likely the last one drilled which is the least likely to be blocked.

Effect of blockages

1 Tahmoor Colliery
• Lack of drainage - the effect on mining depends on when the blockage occurred (more likely early in the life of the hole) and how conservative the pattern design is (can adjacent holes cope with the loss?)
• Potential for intersecting holes under pressure - probably the more serious problem from a safety point of view.

Comment - another likely source of blockage is from debris produced after a hole has been intersected by mining. Hopefully by that stage its function as a gas drainage hole is complete, but there is still the potential for a build-up of pressure if the hole is still producing any gas.

Water

I don't have much to say on this for the good reason I don't know much about it. We have been fortunate up to now at Tahmoor that virtually all our drainage holes have been drilled up-dip or horizontal at worst, so dewatering has not been an issue. It is about to become one however as future holes are all going to be down-dip so I'm interested if anybody can provide some advice.

We have had occasional problems with water in the hosing to individual holes but this is usually avoided by avoiding big loops when hanging hoses.

We have a high vacuum and large diameter pipes in the gas drainage system so have not had major problems with water blockages in the pipes. The only problems have been with pipe movement because of the wave action occurring inside. Water problems in the mains can be solved by provision of adequate drain points, regularly emptied if not automatic.

Leakage

Causes
• Most common cause is a poor standard of standpipe installation with inadequate grouting. Must have clear procedures and properly trained personnel on the job
• Rib fractures - if this is a constant problem then the standpipes need to be longer. If holes are generally satisfactory there may still be locations where geology is different or local stresses are higher, where problems can occur
• Leaks in plumbing - usually from poor standards of installation which should be easily fixed
• Damage to plumbing including standpipes - try to avoid the likelihood of damage by machines (have collars in drill stubs and keep vehicles out; try and put pipelines in roads not normally used by vehicles, etc)
• Intersection of holes by mining or other boreholes

Fixing
• Plumbing leaks are usually easy - just re-do it
• Sealing a leaky standpipe or rib fractures is more difficult. We have drilled (using rib borers) and grouted around hole collars, using special grouts (eg Combextra) with mixed success. It is a very time consuming and labour intensive process.
• Sealing a damaged standpipe can be very difficult - usually a "bodgie" job using anything from plaster to electricians tape. If you're lucky you may be able to cut it off and re-thread it
• If intersected by mining, it may well be that the hole is then no longer required and can be disconnected. If it is still required, or if making gas which is a problem, it needs to be sealed at the intersection point. At Tahmoor we usually seal temporarily using gas bags (often put too close to the rib line and therefore not very effective), later replacing this using another gas bag as a plug placed well into the rib with plaster poured/pumped in behind. These seals also have mixed success.
• If intersected by another borehole, it is usual to identify the hole concerned and close or plug it at the collar to stop the air leakage.

Effect of leaks
• Effect on drainage - provided the leakage does not result in a complete loss of vacuum, drainage should not be affected. The accepted wisdom is that the vacuum level does not affect the desorption rate and, provided the gas can flow, degassing should continue at the same rate.
• Loss of efficiency - with a lot of leakage we will be spending money running fans to drag air into the mine and more running vacuum pumps to drag it out again. I have never sat down and worked out what the costs are but I wouldn't think they would be very big.
• Possible explosive mixes in the line - gas drain lines usually carry mixes which are kept (we hope) above the explosive limit. Any air leakage into the line carries the potential of creating an explosive mixture at some point in the system, so is best kept to a minimum. In most cases the existence of such a mix would not be detected, as monitoring is at best only carried out at a few points (only 1 at Tahmoor).

• Leakage out - if air can leak in under vacuum, gas can leak out into airways if the vacuum is lost and the system comes under positive pressure. At Tahmoor at least, this could be in intake airways carrying people, diesels and non-flame proof electrical equipment.

Discussion and Questions

Chris Harvey, NSW DMR – Have you had any problems with erosion in the main pipelines?
Bob – Not as far as I know, but we are conducting a survey soon on pipes which have been installed for around 20 years.

Brian Lyne, Queensland Mines Dept – Do you grout standpipes?
Bob – Yes. We use different types of cement such as those from Posroc. The standpipes are 6 m long.

Jason Moultrie, North Goonyella – How often do you measure flow rates?
Bob – In the early part of the hole we try to measure at least once per week. It then reduces to about once per month.

John Hanes – Would more frequent monitoring of flows be useful?
Bob – The ideal would be automatic monitoring for the life of the hole.

John – Under ACARP funding, Ian Gray of Sigra has developed a flowmeter. He has just completed a project to install automatic monitoring to the flowmeters which can be connected to the pit’s data highway. Therefore, the product is available for automatic monitoring of gas flow from boreholes.

Chris Harvey – Have you got a good handle on water separation out of the main lines?
Bob – Yes. It is a manual operation. The water falls into a reservoir and it is periodically drained by opening a valve. I believe there is an automatic device developed by Mark Menagazzo.
CONCLUSIONS
SUMMATION AND CONCLUSIONS

Facilitated by Bruce Robertson

Seeing so many people turn up to this workshop helps confirm that the industry has many challenges in the gas and outburst field. We have not solved all the problems and it is through these meetings that we go forward as an industry and learn from each other’s mistakes and successes.

Gas Reservoir Assessment

Most gas content measurement variations are understandable. The inherent variability of sampling is innate to the basic control system that is in place in the threshold system. The only issue outstanding seems to be the potential effect of the partial pressure due to differences in volumes in the Q3 bomb. There is possibly room for more standardization across the service providers if it is thought necessary to address the potential differences between labs.

Permeability seems to be well understood in reservoir engineering terms, but it is still poorly understood in the coal industry and it is such a critical parameter. There are not enough permeability tests done. We let permeability get lost in the empirical approach we have to gas drainage. It is when we encounter high gas or low permeability that a better fundamental understanding of permeability would come into its own. In green-fields or brown-fields studies we really need to spend enough money to conduct interference tests to obtain the correct information for assessing and planning drainage requirements. Single borehole permeability tests are not enough.

The implications of cleat infilling are just starting to come to light. With further ACARP funding in the follow up project, more will be discovered. An interesting part of the discussion was the attempt to understand the post depositional fluid flow and geology and precipitation of the carbonates. This appears to be a fertile area for research and if it is not covered in the new ACARP project, it should be looked at to provide the link between microscopic and macroscopic features. Use the microscope to help understand the causes of low permeability then link back to visible evidence which the mine geologist can use in a practical way.

Nitrogen seems in increase in the gas emitted from coal as the other gases decrease. Measuring nitrogen content of the gas should indicate the gas content level and should therefore present a simplified method for outburst assessment.

Discussion and Questions

Frans Bos – Wireline logs could help characterize the differences in coal seams which affect permeability. It would be interesting to run these logs in horizontal holes.

Ray Williams – Could someone comment on the relative costs of an interference test and a simple permeability test.

Bruce – An interference test requires at least 3 observation wells and a central pump well. On top of this is the cost of a pumping system and a monitoring system. Depending on depth and drilling costs, the total cost would be between $100,000 and $200,000.

Rob Jeffrey – As well as the cost of drilling, there is also the cost of someone monitoring the holes up to 30 days.

Bruce – The beauty of the test is you get the real permeability of the seam. The cost of borehole permeability tests ranges from (on top of the cost of the exploration hole) around $5000 per individual sample and the multiphase test about $25,000. It depends on the number of tests done in the one hole. But you only get the permeability of the seam tested, not the directional permeability. If you have many of these across a site, it is useful. But if you need reliable information for a reservoir simulator, or to design a gas drainage system, it is necessary to get real permeability testing.

Ray – Permeability testing needs to be staged. The spot permeability tests provide information to best locate the interference test and its monitoring equipment.

1 Anglo Coal
Phillip Eade – If we are talking about reservoir characterization, we should not restrict testing to the coal seam which we are mining. In the Bulli seam, only around 10% of the gas comes from the mined seam. To characterize the reservoir, the study should include all seams and even some sandstone reservoirs.

Bruce – The details required for the other seams is probably less than for the working seam as there is no concern for outburst risk, but the gas emission risk is important.

Brian Lyne – Would interference testing provide relevant information to solve problems such as those at Tower which occur in only a small area of the mine, say 100 m wide?

Bruce – You use the cheaper permeability tests in many holes to characterize the whole area. If there are consistent results (with depth) with that sort of information, you then chose where to do an interference study. Fortunately the problems at Tower are not common across the industry. There are ways to ensure that the interference test is not conducted in an anomalous area like Tower’s.

Phillip Eade – There is more variation than what we might think, eg Tahmoor, Tower and West Cliff have all had low permeability areas.

Rob Jeffrey – It is very useful to do a stress measurement with a permeability test because the permeability is so closely controlled by stress. It adds about an hour and a half to the test time by doing a step rate test in the seam after the permeability test. The results then can be correlated with the stress and this will give a better understanding of what is happening and why permeability might change between holes.

Bruce – With the reservoir tests we only concentrate on water permeability. Is there an air curve test we can do for gas or is that easily assumed from the results?

Rob – Multi phase testing can be conducted, but it is considerably more expensive. Numerical modeling is then required to analyse the results. Multi phase testing does extract a field relative permeability.

Ray – The relative field permeability is important for modeling. In our current ACARP project on modeling, we have had considerable troubles with the various models we have tested fitting curves for relative permeability. We then got Dave Casey to extend his multi phase test from 8 hours to 5 days so we had more data to work with. There is still much work to do, but relative permeability is very important. In a case where the water permeability falls to zero and the water saturation is still 60%, there is a huge affect on what can happen on mining.

Bruce – How close to measuring the required parameters can you get in a closely monitored in-seam drainage hole?

Ray – When it comes to coal mining, the curve matching we can do is not exactly brilliant because we can never get decent water flows. Work done recently with CH4 in Queensland curve matching their production from surface to in-seam has been very good as they had good water data and pore pressure. We were able to apply a few different relative permeability curves to get good matches. You can curve match but the input parameters are not necessarily unique. From interference testing, you can get porosity and compressibility, two numbers we play around with in sensitivity studies.

Rob – The problem is there are so many parameters involved that there is a uniqueness problem in obtaining the one and only relative permeability curve from the data available. The only alternative is to do core testing which is with very small samples which are not necessarily representative.

Ray – From CBM testing arises the skin factor controlling desorption pressure. You can go to great lengths to get all the input parameters right then stuff it up when you drill the hole by the way you drill it. You can drop the desorption pressure too quickly for example. In-seam drilling is conducted mainly at or below gas desorption pressure. Gas is desorbing uncontrollably while drilling and this damages the hole. When we curve match those productions, we might use a permeability that makes it all fit, but it might not be true. The Sigra borehole pressurization tool could be very useful here. When drilling in soft coal in-seam, there can be active gas desorption while drilling which leads to bogging of the rods and other problems. If you pressurize the hole while drilling, the environment is quite benign. In-seam drilling from the surface uses water pressure to stabilize the hole and drilling is easy relative to drilling under ground. Then, when the water is drawn down sufficiently to allow gas desorption, all hell breaks loose.

Tim Meyer, CMTE – What is the mechanism for hole damage when the pressure in the hole is not controlled?
Ray – When drilling from the surface into permeable coal, the weight of the drilling fluid can force fluid into the formation. Once the pore pressure is reduced to around gas desorption pressure, (it is worse with more permeable ground) if the gas is migrating a long way to the hole and is carrying a lot of coal fines, too quick a drop in pressure can cause the coal pores and fractures to block and thus reduce the permeability. In testing we try to drop the pressure very slowly to reduce damage. In coal mining parlance, you want to drop the gas pressure down as quickly as possible, get the gas out and get in and mine it as soon as possible. But in surface to in-seam drilling, it is known that a quick reduction in pressure will damage the hole.

Bruce – If you depressurise too quickly, you reduce the pore pressure and reduce the permeability before the coal has a chance to shrink.

Bruce – Has anybody had significantly variable sorption isotherms across their site? Is variability something we need to worry about?

Ray – Isotherms are a bit like permeability in that you do a lot of gas content tests, only a few permeability tests and only one or two isotherms. We are now doing more isotherms in our ACARP project in conjunction with CSIRO and James Cook University. The variations do not appear to be maceral related, but seem to be controlled by ash and the temperature of the test. There are issues with isotherms which will be clarified in our ACARP report.

Outburst Assessment
Outburst modeling has advanced considerably and is now looking at the mechanism of the outburst. We look forward to how much more CSIRO can develop the model so we can apply it to our individual circumstances.

Chris Harvey pointed out we have taken a singular approach to outburst control with respect to reservoir parameters. He encouraged us to open the debate to parameters other than gas content. The view of the session was that most of the other parameters are pretty difficult to practically and realistically obtain. This means they are difficult to build into outburst management plans.

Issues were raised to where outburst research ought to be directed. There has not been a high focus on outbursting by the ACARP Underground Sub-committee over the last few years, but in the priority setting session held last year, outburst research was rated as the top priority. We funded more outburst related projects in 2002 than in previous years. One of the projects is a scooping study that John Hanes will lead to define a road map to focus outburst research funded by ACARP over the next few years. We will be looking for input from the researchers as well as from the operators in the field.

The modeling of development gas emissions by GeoGas shows what can be done in predicting gas emissions during mining. Rao described the modeling at Dartbrook.

Discussion and Questions
Chris Harvey – There is a need to document any outburst incident in a proper fashion. Some time ago, the DMR tried to establish an outburst data base. As soon as we started the data base, we stopped having outbursts. The problem than became how do you measure something that is not happening? I urge that operators should clearly document what they have on outbursts then distribute that information. We will fall into traps if we believe what we will experience in the future is controlled by what has happened in the past. We are not going to understand what is happening unless we thoroughly understand what has happened in the past. There is a template for reporting in the 1996 book by Ripu (Editor's note - the form approved by the Outburst Technical Task Force after the South Bulli fatalities is included with these notes) But there is no single repository for outburst reports.

Rob Jeffrey – A web page could be established with ACARP so people could fill in the information via the internet.

Jason Moultrie, North Goonyella – At North Goonyella, we had an incident which, when compared with south coast incidents, could be called a minor outburst. But how do you define a minor outburst?

Chris Harvey – At Tower Colliery, a small outburst occurred while remotely mining through the tight zone. The outburst potential was known as the gas content was 13 m3/tonne, so a plan for mining under outburst conditions was in place with nobody at the face. The mining of this relatively small outburst was considered a success.

Bruce Robertson – It is dangerous to distinguish between small and large outbursts as small outbursts can be precursors to large ones.

Ray Williams – If a gas content sample yields 9.1 m3/tonne and the threshold is 9, then there is a problem about what to do. More detail is needed for these conditions.
Chris Harvey – A system safety approach is required for outburst management. There are few examples to follow from overseas as they do not seem to have any idea of safety barriers.

Bruce Robertson – There is a distinction between structure free conditions and structured conditions with the latter being more common. If you can reduce the gas content to a manageable level, you should not get an outburst from structure free coal. The structures are the focus for potential outbursts in otherwise drained coal. Techniques to reliably detect structures should be advanced.

Brian Lyne – The Central outburst was associated with a structure and similar structures had been previously mined through without incident. I have not heard how such a structure can be predicted ahead of the face. The last fatal outburst at West Cliff occurred from a drilled face and these holes did not detect the structure.

(Editor’s note: The West Cliff fatal outburst occurred from a structure on the right hand rib. The advance drill hole that should have covered this rib, as with the other holes drilled in the advance, veered to the left and was located in the left rib. It entirely missed the structure.)

Bruce Robertson – If a structure is known, it must be well tested ahead of mining by advance drilling. The central structure was unknown and the intersection with the outburst was the first intersection of this structure.

Ray Williams – At Tahmoor, the tight coal is relatively free from structures and has low permeability probably as a result. Therefore, it should be safe to say that in this case, if no structure is detected, it should be safe to mine at higher than the normal threshold gas content. If the permeability is low because of stress, then there is need for concern.

John Hanes – There is a need to ensure that all personnel, especially face crews are well trained to recognise changes in conditions at the face. Such changes include structures, cleat changes, water changes, gas changes or mining strain changes. Noticing changes is the first barrier.

Modelling – No discussion was required.

Drilling Technology
Contractors such as Mitchell Drilling, CH4, Phoenix and others are interested in surface to in-seam drilling.

Henk showed how survey tool technology has advanced and how more value can be added. ACARP and suppliers are active in trying to improve what can be got out of a hole. We should be able to a lot better than we are, especially in long in-seam exploration holes by getting geophysical logs from these holes.

John Hanes gave a summary of where in seam drilling technology is today.

Discussion and Questions
Henk Verhoef - Work being done by CMTE in the development of the natural gamma probe.

Bob Newman – Is any research work being done to improve drilling through sticky ground?
Tim Meyer, CMTE – We are about to conduct trials using waterjet to replace the downhole motor, with a bent sub and high pressure rods. We did a trial about a year ago into the escarpment at Nebo Colliery. There was little success partly because of equipment design problems, but also because of the poor ground of the trial site. The next trial will be into a highwall at Moura. We aim to drill 300+ m demonstration holes. CMTE has also developed survey capability, mainly for use in TRD work, but also in the rigid drill string work. At Tahmoor, about 2 years ago, we re-entered about 5 holes that had been draining for a few months. The aim was to slot the holes in a number of orientations to try to stimulate extra gas flow. We did not get any extra gas. What we have heard today about carbonates in the cleats might help explain why we did not get much success. We are keen to try slotting in normal coal to see if slotting will accelerate drainage, especially in mines that are behind schedule.

Phillip Eade – Has anyone done any work on whether increased suction will improve drainage. Also what has been done with hydraulic fracturing of in-seam holes?
Bruce Robertson – Rob Jeffrey did some hydraulic fracturing trials at Central Colliery with ACARP funding 6 years ago. The packers could not effectively seal the hole and there was pressure bypass. Some fracking by Ian Gray has been tried at Dartbrook recently to overcome a low permeability problem. We got an immediate increase in gas make, but are now dewatering the zone prior to gas make to see what the success was. It was a
quick efficient process to generate 3 m fractures in the hole. The coal is hard and brittle so we had no real problems fracturing it with modest treatment pressure. We did not have to use high pressure pumps.

**Tim Meyer** – We are trying to get some novel drilling technologies tested. To do this we need test sites and some equipment to support the experiments. That requires coordinating with a contract driller or one of the mine’s drill crews. As drilling is a high priority at most mines, it is near impossible to get support or a site for the trials. It is very frustrating trying to schedule for any experiments. I do not know how this can be solved.

**John Hanes** – Ian Gray and others are having the same problems. There is some good equipment just awaiting field trials so their development can be finalized and the new technologies introduced to the mines. They need a mine site for trials and industry champions. We can dream all we like about solutions to drilling and other problems, but unless industry is prepared to give the projects the support they need, new technology will not get off the ground.

**Henk** – AMT trialled the DGS system at Tower Colliery with excellent cooperation from Tower.

**Tim** – We have also had excellent cooperation from some mines, but have also been frustrated by long delays or cancellation of support.

**Chris Harvey** – We were to have a paper from EDL on gas utilization. If the industry wants to survive beyond this century, it has to utilize the gas it drains rather than waste it.

**Bruce Robertson** – I agree. But at the moment it is not profitable. It will be in time.

**Drainage and Extraction**

We benefited from Bob Newman’s experience in borehole maintenance. We can only commiserate with Miles Brown’s Tower experiences. Although we have a good handle on drilling and other technologies, Tower has shown that Mother Nature can turn around and bite us from time to time. In some cases we have to walk away from problems.

Thanks to all for attending, presenting and contributing.
APPENDIX A

OUTBURST REPORT

1. LOCATION
Mine: ........................................ Panel: ..............................
Roadway location: ........................ Co-ordinates: N……………… E………………

2. TIMING
Date: ........................ am/pm
Time: ........................ am/pm
Day of Week: ........................
Shift: ........................

3. HEALTH AND SAFETY
Injuries: .............................
Fatalities: ...........................

4. GEOLOGY
Presence of Structure: ........................ Type of Structure:.............
Fault: Type: normal/reverse/thrust/strikeslip/ .................................
Throw: .............. mm Dip:..................... Dip Direction: ........................
Dyke: Thickness:............. Hardness: ........................
Stress Direction: ........................
Frequency & Direction of Jointing: ............................
Mylonite (Gouge Evident): Thickness: ........ mm
Changes in Coal Properties: ............................
Changes in Roof: ............................
Changes in Floor: ............................
Water: ............................
Seam Gas: Methane:……….. % Carbon Dioxide:………….. %
Content ………. m3/tonne
Drill Log: Comments: ............................

5. VENTILATION
Air quantity: ............. m3/sec. Location: ..............................
Gas Readings:
Before o/b CH4 ......... % CO2 .............%
After o/b CH4 ......... % CO2 .............%
Variations detected: ............................

6. MINING DETAILS
Outburst mining method in use: Yes/No
Method of mining: Cont Miner/Road Header/Shearer/ ..............................
Mine Layout: Heading/Cutthrough/ Pillar/Split/ ..............................
Distance Driven: On Shift ........... m Last 24 hours ....... m Last 72 hours ....... m
Face Description: Cut out LHS ........................ Cut out RHS ........................
Undercut ........................ Face square ........................

7. INCIDENT DETAILS
Nature of Discharge: Lump Coal: .............................
Fine Coal: .............................
Roof Stone/Coal: .............................
Location of Outburst: Face centre ........................ Face LHS ........................ Face RHS ........................ Left Rib ........................ Right Rib ........................
Quantity discharged: ........................ tonnes
Distance solids thrown: ........................
Gas Liberated: CH4 ........... % CO2 ........... % CH4 ....... m3 CO2 ........... m3
Coning Evident: ............... Haze: .............. Brown/Red Colouring: ........
Effect on Machinery: Trip-outs: .................... Distance dislodged:..............
Comments: ........................................................................................................

8. OBSERVED
Audible Signs: Rumble/Bumping/Knocking/ ..............................................
Loud/Soft/ ...........................................................................................................
Visible Signs: Coal Spitting/Slabbing/Mass
Ejection/Surging ................................................................................................
Sensed Signs: Air warming/Air Cooling/Smell/ ..............................................

9. WRITTEN REPORTS (Attached).
Witnesses ...........................................................................................................
Deputy ................................................................................................................
Undermanager ............................................................................................... 
Check Inspector .............................................................................................
Others ............................................................................................................... 

10. PLANS
Mine Location (1:10,000)
Panel Location (1:2 000)
Gas Drainage (1:50)

Prepared by .......................................................... Date ............................. 

Signed: .......................................................... Position: ............................

Authorised / Countersigned ........................................ Position: ........................
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