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**PROCEEDINGS OF THE 2004
COAL OPERATORS' CONFERENCE**

UNIVERSITY OF WOLLONGONG, NSW

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FOREWORD

The underground coal operators conference series held annually in Wollongong has been recognised as the main form of the exchange of ideas between mine operators, engineers and researchers in the diverse field of coal mining technology. For the last five years the conference addressed a variety of issues, focusing primarily on underground ground control and mine safety. In order to increase participation, attention has now been drawn to addressing various other issues in addition to ground control. The theme of Coal2004, Mine Planning demonstrates the true interest of the conference in promoting high output longwall operation. This year the conference is preceded by a halfday workshop on mine subsidence and surface environment impacts.

The COAL2004 Conference has been generously supported by the following organizations:

- BHPBilliton, Illawarra Coal
- Minova Australia

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 and manufacturing industries in Australia and elsewhere. The Aus.I.M.M Illawarra Branch and the organising committee are extremely grateful to all the above for their support.

The Organising Committee also extends appreciation to James Cook and his colleagues at the Wollongong Union Centre for the management and registration of the conference, Bruce Robertson for his assistance in Audio Visual maintenance, Leonie McIntyre the Faculty of Engineering for type setting the Conference Proceedings, Anthony Petre for assisting in designing the proceedings cover and the University of Wollongong Printery for printing copies of the proceedings.

Naj Aziz, *Associate Professor (Editor)*
Bob Kinnimoth, *(Co-Editor)*

PREFACE

The Illawarra Branch of the Australasian Institute of Mining and Metallurgy, (AusIMM) has great pride in continuing to conduct high quality conferences for the coal mining industries.

This year's conference, the fifth in the series of annual Coal Operations conferences, is a joint collaboration between the AusIMM, the University of Wollongong and the Mine Managers Association of Australia. The breadth of topics and speakers cover many of the most critical issues for the coal industry, including strategic planning, addressing OHS requirements, environmental aspects, personnel management, combustion and ventilation, hydraulic, geo-mechanics and roof support. Clearly, these topics are critically important for the long term success of the Australian coal industry.

As Chair of the Illawarra Branch, AusIMM, I wish to thank the organisers for the considerable effort they have put into making the conference a success and those who have participated in the conference. On behalf of the Illawarra Branch, I wish you a successful and rewarding conference.



Ray Tolhurst
Chair, Illawarra Branch
Australasian Institute of Mining and Metallurgy

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KEYNOTE ADDRESS

STRATEGIC PLANNING FOR THE FUTURE

Bruce Allan ¹

INTRODUCTION

Planning and development for both new and existing coal mines is a major process that has increased in complexity from one involving a small number of stakeholders that were primarily focused on the broad issues of:

- Resource
- Technical process
- Market
- Business result

to one today which is quite complex, involving many internal studies, corporate checks and balances, expert consultant studies and many stakeholders both small and large, government and non-government.

The aim of this AUSIMM 2004 Conference and Workshop is to bring together expert speakers who will seek to highlight and share some of their experiences of issues and aspects which impact on coal mine planning and development – both now and into the future.

In any business project or venture, and coal mining is no different, it is important to clearly understand all issues that will impact on a project and its viability and have in place appropriate strategies to manage and ensure the successful outcome of all goals that are set.

In using the word strategy in this paper it is used in the sense of pertaining to skilful management of any kind, involving structured plans or schemes that achieve a successful end result. Strategy used in a military sense, has down through the ages denoted rigour and discipline to an action, coupled to accurate evaluation and understanding of the risks associated with the outcome.

Mine planning is no different it depends upon accurate evaluation and risk understanding and quality management. Planning must display foresight and vision of a changing world, through leadership and direction. To support the importance for strategic planning, I share with you the case study of Illawarra Coal and the Company's quest to create World-Class Mines.

ILLAWARRA COAL CASE STUDY

BACKGROUND

In 1995, BHP Collieries, operating as part of the BHP Steel Group, owned and operated four (4) coalmines in the Illawarra, namely:

- Appin Colliery – 1960
- Cordeaux Colliery – 1976
- Tower Colliery – 1977
- Elouera Colliery – formed in 1990

¹ BHP Billiton – Illawarra Coal

Total raw coal production 7.6 Million tonnes/annum (mtpa) , 5.6 mtpa mined from the Bulli seam and 2.0 mtpa mined from the Wongawilli seam.

Coal washing and processing was at two locations, Appin – SADA coal preparation plant (2 mtpa) with all the remaining coal from the Bulli Seam and the Wongawilli seam, (5.6 mtpa) at a conglomeration of coal washing plants located within the Port Kembla Steelworks – BHP.

Transport of coal from the mines to the coal preparation plants and then to either the steelworks blended beds or to export, consisted of three modes of transport involving road haulage, conveyor systems and rail transport on both private and public networks.

Waste disposal of coal washery refuse was on mine property adjacent to the Elouera Colliery at Wongawilli.

The operations were costly and constrained in production capacity. Mine planning was limited in thinking, with little planning for the future, targeting mainly for the budget year. The mines were focused at continually removing costs from the business to stay competitive with limited new capital being invested back into the business for improvement. Illawarra Coal operations in 1995 were not world-class.

THE STRATEGY

In 1996 when West Cliff Colliery (owned and operated by Kembla Coal & Coke – KCC) became available for sale, it was considered the mine had some synergies to the adjacent BHP Appin Colliery, but these synergies were not clear.

The potential acquisition of West Cliff Colliery provided BHP with the opportunity to re-evaluate the current operating and planning strategies for the Illawarra Collieries Group.

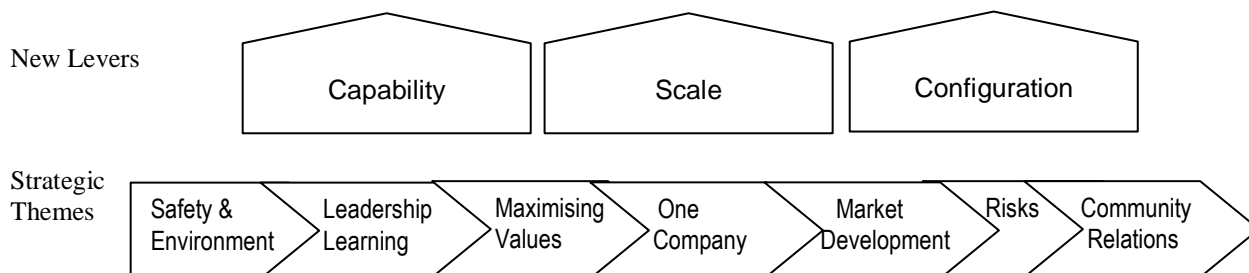
In mid-1996, BHP Collieries moved under the management of the BHP Minerals Group, this group having a totally different operating and planning philosophy to that of the BHP Steel Group.

In July 1996, a small strategic planning group was formed within Illawarra Coal, made up of senior management, some with existing BHP backgrounds and some from outside BHP.

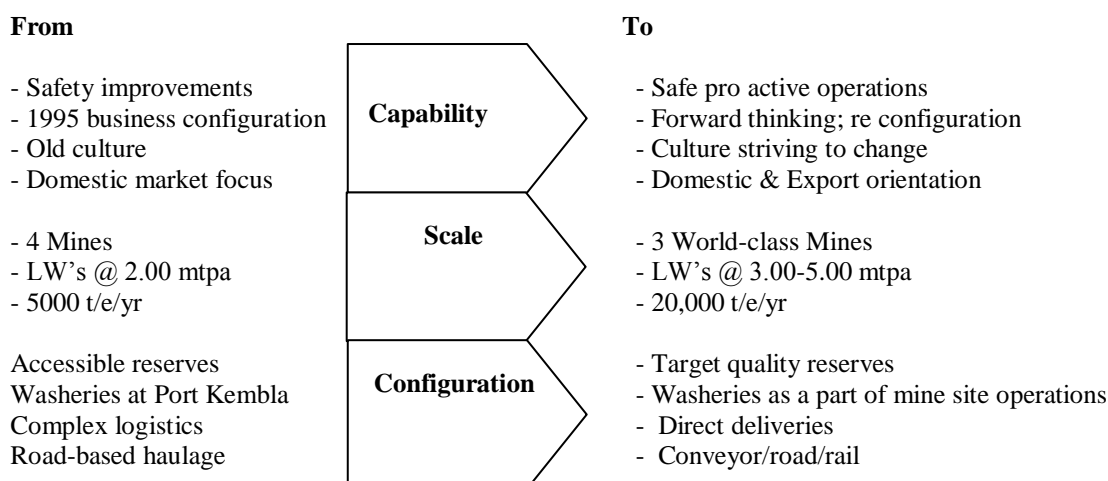
A vision of the future was developed by this strategic planning group, which would lead to the creation of value with future world-class mines for Illawarra Coal.

Key element of this vision centred on the following:

- Objectives
- Safety
 - Productivity
 - Financials



This new vision would require a step change across the business:



In assessing the potential of KCC's, West Cliff Colliery, the strategic group developed the view that West Cliff Colliery and it's associated infrastructure could significantly enhance the value of Illawarra Coal through the ability to provide:

- Lower cost, rapid entry to BHP existing quality reserves
- An infrastructure hub in the Appin area
- A significant logistic advantage leading to lower costs
- 6.5 mtpa coal washery facility that could be further enhanced
- 2.5 mtpa coal stock pile area at site
- 15-year coal washery refuse emplacement area at mine site
- Additional coking coal reserves for BHP Billiton and
- Low community impact – a remote site

Following a detailed merger and acquisition evaluation study by the strategy team, covering such areas as:

- Business strategy
- Risk (Business and Safety)
- Market analysis
- Mineral resources and geology
- Mining
- Coal processing and logistics
- Infrastructure and
- Health, safety, environment and community

West Cliff was purchased by BHP in April 1997 and incorporated into Illawarra Coal.

THE WAY FORWARD

To meet the future vision for the BHP Billiton Illawarra Coal Strategy of 3-5 mtpa mining operations, longwall dimensions at operating mines would need to be increased from 200 metres to 300-metres wide, with block lengths up to and exceeding 3 kilometres, containing some 3 – 5 million tonnes of minable coal.

The age of existing mines and the suitability of infrastructure was evaluated.

Elouera Colliery is the only source of Wongawilli seam coal to Illawarra Coal and is a vital component of the Company's, coal blending and marketing strategy and a key ingredient to the Port Kembla Steel Works for coke making.

Elouera Colliery, mining the Wongawilli Seam, will reach the end of its economic life in early 2005. Plans had been in place for some period of time to prospect and develop the Wongawilli Seam through the existing Cordeaux Colliery mine workings as a replacement source of coal.

The plan to re develop Cordeaux Colliery was re-evaluated by the Strategic Group. It was considered following detailed investigation, that it would be better to develop a new mine, of world-class standards, closer to the coast, to extract the Company's Wongawilli Seam reserves. This could be achieved from existing Company owned property located on the Illawarra escarpment at the old Nebo mine site. The decision led to the Dendrobium Mine Project, which came into being to provide up to 5.00 million tonnes per annum of raw coal, from the Wongawilli seam, by late 2004.

On the completion of Bulli Seam extraction at Cordeaux Colliery and following the removal of all valuable plant and equipment from the mine, the operations were finally sealed in 2003, capping a 27-year operation.

To achieve the same criteria of 3-5 mtpa longwall operations, in the Bulli Seam, at other mines in Illawarra Coal, a similar review was undertaken by the strategy team. One of the mines, Appin, Tower or West Cliff would need to be combined to create the appropriate longwall domains for the future. As West Cliff contained the "hub" of surface infrastructure and Appin had sound surface infrastructure, was located close to West Cliff and was equipped with a 2000 tph seam to surface coal clearance system, the decision was made to close Tower Colliery operations. This occurred in December 2002 and the Tower mine workings were progressively incorporate into the mine workings of Appin Colliery thus providing additional ventilation systems to manage the high methane seam gas contents and permitting increasing mine production from larger longwall domains in the near future.

The strategy behind the Appin and West Cliff mine re - configurations and the development of the new Dendrobium Mine led to a very significant rationalisation in the movement and transport of coal from Illawarra Coal mines to the Port Kembla Coal Terminal (PKCT) and to the Port Kembla Steelworks. Road haulage of raw coal carried along the main Picton Road from Tower and Cordeaux would eventually cease, significantly reducing transport and logistics costs from that region.

The private Kemira Valley rail line would be upgraded and become a dedicated rail line for Dendrobium Coal to an upgraded Dendrobium coal washed plant located within the Port Kembla Steelworks. The road haulage of wash Bulli Seam coal from West Cliff, would remain the only product being transported to the Port Kembla Coal Terminal and BlueScope Steel via the Appin Road, a Federal Government dedicated export road. This strategy has permitted Illawarra Coal to engage a road haulage contractor for this dedicated coal transport route with a purpose built road haulage fleet with enhanced safety features bringing improved cost of performance for group logistics.

REVIEW OF THE STRATEGY

Through a structured strategic planning process, commenced in 1996, BHP Illawarra Coal, now a part of the BHP Billiton Group (BHPB), has significantly changed the capability, scale and configuration of Company mines in the Illawarra.

The process outlined has been the positioning aspect of planning for the future. Success in the on going operation of underground coalmines is to identify, understand and manage the risks in a systematic way.

The strategy team identified the following key future risks to the Illawarra Coal:

- External Affairs
 - Residential, Industrial developments
 - National Parks
 - Transport
 - Mine Subsidence Impacts
 - Strategic Land acquisition

- Environment & Community
 - Transport
 - Future refuse emplacement
 - Mine subsidence
 - Emissions and waste water
- Underground Mining Issues
 - Seam gas contents – removal of gas to safe acceptable levels prior to mining
 - Employee safety and exposures
 - Geological uncertainty
 - Modern fit for purpose mining equipment
- Technological Change/Markets
 - Steel industry process alternatives
 - Supply and demand fluctuations
 - International exchange rate movements
- People
 - Future technically and statutory employees
 - Leadership development of key communicators
 - Involving all levels of the workforce in the understanding and achievement of the business objectives
 - Ensuing the full-unimpeded release of our people's potential.

CONCLUSION

The future of mining by BHPB, in the Illawarra, will continue to be sustainable through the rigorous planning processes outlined. Planning tools and networks are in wide use across the BHPB organisation and are regularly tested and re-evaluated at all levels to ensure appropriate suitability.

With all the other competing interest of the region, never in the history of mining in the Illawarra has planning, for the future, been so important. For the future success of underground coal mining in the Illawarra, we must ensure that our mine management teams plan and think strategically.

KEYNOTE ADDRESS

PEOPLE WITH PURPOSE CREATING PROSPERITY AT THE CSA MINE

Richard Morland ¹

INTRODUCTION

The CSA Mine has been an important part of the metalliferous mining industry in central western New South Wales for the last 40 years. Indeed, it is an icon of the community of Cobar (Fig. 1), some 10km distant, to the south of the mine and processing facilities.

Since its beginnings in 1871, the mine has had a colorful history, often beset by technical problems, experiencing intermittent industrial unrest during the 1970's, 80's and 90's, and subjected to a series of corporate changes which often prevented the operation from consistently achieving its full potential.

Despite all this, the operation has been an important contributor to the community of Cobar for many years, producing over 25 million tonnes of ore since 1965, and yielding \$1.5 billion of revenue over that period.

A watershed in the mine's history occurred in 1999, when Cobar Management Pty Ltd (CMPL) took control of the operation, following its closure by Ashanti Gold, in 1998.

A NEW BUSINESS MODEL

CMPL adopted a very different philosophy from that under which the mine operated until 1997.

For many years leading up to 1997, the mine had been producing over one million tonnes of ore annually, but at head grades consistently less than 3% copper (Cu). This reflected a 'tonnes mentality' very common in a number of mining companies, and reflected an acceptance of dilution as a natural consequence of large scale mechanized mining.

CMPL adopted a strategy which paid much closer attention to wall rock dilution and excavation stability, opting for a production rate of around 500,000 tonnes per annum, but at head grades of more than 5% Cu.

Hand in hand with this small tonnage philosophy, came a reduction in fixed costs, through a smaller equipment fleet, less people, but achieving metal output almost as high as that achieved with the higher ore production of previous years. The business model was based on utilizing existing plant infrastructure – very little new capital was required to restart the operation.

In addition, much good work was begun in the area of workplace culture. A new climate based on sustainability was created, with a focus on safety improvement, best practice environmental performance and a systems-based approach to doing business.

Contractors were employed initially to carry out the core mining and maintenance activities, followed by the ore processing area twelve months later. The use of contractors in the early stages of the operation's rebirth reflected the assessment of operational and business risks that existed at the time.

1999 and 2000 were good years with high metal output, high productivity and high metal prices.

But problems were looming.....

¹ General Manager, Cobar Management Pty Ltd

TECHNICAL DIFFICULTIES

The last quarter of 2000 revealed the first signs that the CSA Mine would live up to its reputation of being 'difficult'.

A high grade stope failed late in the year, continued to cave, and set off a chain of events that resulted in a steadily declining performance over the next 12 months.

This circumstance arose from stress related ground activity not anticipated in the feasibility study for the new CSA.

The 2001 year saw the operation just achieve its budget in terms of ore output, but development, back filling and operating costs were unsatisfactory, due to the ongoing effects of the stope failure, and protracted attempts to recover from its consequences. The mine failed to make a profit or any positive cash flow in the last six months of that year.

By the end of 2001, the mine was out of ore, and things were looking grim. Cash reserves were dwindling and the employees on the site were justifiably concerned about their future.

OUR PURPOSE – FOUNDATIONS OF THE TURNAROUND

It was evident at the end of 2001 that something needed to be done, and quickly, if the operation was to survive. The milling operations were suspended in late December 2001, due to major ground collapses in the mine, and the consequent inability of the operation to supply mill feed. These collapses had occurred as a result of pushing the mining operation beyond its capacity to deliver.

A new management team was brought into the operation. The team was composed of very experienced industry professionals, some of whom had been involved in significant turnarounds at other operations.

Whilst the problems in the mine were urgent, it was agreed by the team that there was no need to panic. It was clear that 2002 would be a year of recovery and consolidation. Profitability was a desired outcome, but was not the main goal.... it was more important to re-establish the capability to produce revenue, meaning that the restoration of sustainable mine output was the main priority.

A strategy was quickly developed, based on getting the process inputs under control and doing the basics consistently well... good results would follow. The strategy was communicated to CMPL's parent company, a large Swiss-based commodities trader, in order to ensure their support for the tough times ahead. This financial support would prove to be crucial in getting the operation back on its feet.

The foundation of the new approach was a commitment to formal business planning which integrated –

- A clearly articulated Vision
- A fundamental set of underpinning Values
- A defined set of Critical Success Factors
- Stretching, but realistic Business Goals.

At the outset, it was realized that the success of this approach would depend heavily on –

- The quality of the leadership provided by the senior management team, superintendents and supervisors.
- The effectiveness of the communication of Vision, Values and Goals.
- The amount of engagement or buy-in that could be gained from the workforce.

In addition, it was important to understand the root causes of the problems which had occurred in 2000 and 2001.

The issues which contributed most to the operation's problems were –

- Inadequate planning
- Inability to place backfill
- Inadequate rock mechanics input to the operations
- Inappropriate organizational arrangements
- Poor definition and execution of strategy.

It was also clear that short term thinking had permeated all parts of the organization, resulting in a succession of expedient decisions which had served to exacerbate the problems facing the operation.

A step back was required. The commitment to a longer term strategy included a recognition that things may go backward before starting to go forward. While this was clear to the site based management, the communication of this concept to the distant parent organization was critical. They quickly accepted the philosophy behind the new strategy, and provided their full support to ensure that the long term vision for the business was given the best possible chance of being realized.

TAKING STOCK

The mining operation had been under enormous pressure during 2001 to deliver an ore quantity which was simply beyond its capability; indeed the mine plan for 2002 was envisaging even higher production than 2001. There was no evidence of a structured planning approach to achieve this, despite clear evidence that the mine's technical complexity was increasing in tandem with the envisaged production increase.

Clearly, the target was unrealistic. The mine plan was immediately revised from 700,000 tonnes per year of output to a more realistic 550,000 tonnes.

The organizational structure was overhauled, and divided into three functional areas – Mining, Ore Processing and Site Services. Clear accountabilities for all facets of the operation were defined for the leaders in these areas.

Accountability for business decision making was vested in the hands of the CMPL management team, moving away from a “management by committee” approach which had been enshrined in some of the contractor arrangements until that time.

The importance of defining a long term vision for the operation was also recognized. Therefore, in parallel with addressing the immediate technical and operational issues, there was a very strong commitment to “in-mine” exploration drilling. This commitment was preserved in the face of a short term budget position that was very tight.

This was a very clear statement of confidence in the long term, and was fundamental to collecting vital information to assist in better planning processes.

OUR PEOPLE – GAINING ALIGNMENT

CMPL employs 200 people at the CSA Mine. Experience shows that properly selected people want to contribute to the success of a business... if they are given the chance.

Despite the problems besetting the operation, it was clear that the CSA Mine workforce was far from demoralized... they were concerned, but still showed a high level of enthusiasm for confronting and overcoming the challenges which lay ahead.

It became clear that leadership was going to be critical ... the energy of the employees would quickly dissipate if they were not aligned with the business objectives and properly involved with the plans to improve the performance of the operation. The recovery was to be leadership driven and people based.

A number of initiatives were implemented.

1. The management team quickly and clearly articulated its leadership philosophy, in the context of the newly developed vision. Management took responsibility for leading, and made a public commitment to the entire workforce, to provide a high standard of leadership at all times, and to hold itself accountable for the quality of that leadership.
2. The business plan was communicated across the site.
3. There was an immediate commitment to train all organizational leaders in "values based leadership". All leaders in the organization were required to commit to behaviors consistent with the Values – the effectiveness of this commitment was not left to chance.
4. The behaviors expected of everyone were clearly spelled out, in the context of the new organizational values.
5. The focus on achieving targets was clearly articulated. It was made clear that Vision, Values and Goals go together. In other words, Vision and Values are critical to ensuring that hard business targets are achieved.
6. There was immediate emphasis placed on housekeeping – getting the work environment right. A series of site clean-ups was held to get everyone's attention and to define the new standards. All employees participated in clean up of their work areas.
7. Improved work rosters were introduced to allow employees to balance fatigue, fitness and family. Rosters appropriate to fly-in, fly-out operations which had been previously implemented for all mine operations employees, were discarded immediately. The mill areas soon followed.
8. The management team made it clear that the site would move to a predominantly owner-operator status within a year, consistent with a philosophy of developing a highly effective relationship with its employees.
9. The equipment fleet in the mine was refinanced, enabling CMPL to take ownership of the fleet in 2003, at a fraction of the cost of a new fleet. This further reinforced the owner operator philosophy and reinforced the sense of confidence in the future.
10. Industry standards were introduced where there were obvious and significant benefits to the business. Examples include the introduction of backfill management standards and a JORC compliant resource statement.
11. Initiation of a Life of Mine concept study.

All of these initiatives provided the opportunity for management to demonstrate its credentials, to create a longer term perspective, to involve employees, create excitement about the possibilities and to promote confidence in the ability of everyone to contribute meaningfully to improving the business.

VISION, VALUES AND GOALS – PRODUCING RESULTS

Vision and Values provide a basis for distilling the philosophy of an organization into relatively simple words and statements. The judgement on the merits of that philosophy must be based on the assessment of the results achieved. That is why Vision and Values must be integrated with the attainment of the business objectives. If not, they are nothing more than "motherhood" concepts.

The results from the operation achieved over the last two years, speak for themselves –

- 70% improvement in safety as measured by Lost Time Injury Frequency Rate and Total Injury Frequency Rate.
- Best mine development performance by CMPL, in 2002 and again in 2003

- Highest ore tonnage milled by CMPL, in 2002 and 2003
- Best concentrate output achieved by CMPL, in 2003
- Second highest metal and concentrate production ever, in 2003.
- Concentrate sales 35% above long term (20 year) average, in 2003
- Return to monthly profits by June 2002.
- Return to annual profit in 2003.
- Significant cash generation in 2003, permitting self-funding of all capital expenditure.
- Significant repayment of long term debt achieved in 2003.

INVOLVED PEOPLE – CREATING A PROSPEROUS FUTURE TOGETHER

None of the achievements of the last two years would have been possible without employees who were prepared to commit themselves fully to the new business strategy.

There is a strong commitment, through observance of the organizational values, to

- working with common purpose,
- holding people accountable for their actions,
- acting with integrity,
- demonstrating respect by valuing everyone's contributions, and
- ensuring that plans are executed with a sense of urgency.

People enjoy working at the CSA Mine. All of the operational and corporate functions are carried out on site, with the result that accountability for running all aspects of the business remains on site. This has facilitated an atmosphere of getting things done, with relatively quick decision-making processes, and a philosophy of pushing responsibility to where it belongs in the organization.

People in the CSA organization have fulfilling and important jobs. Everyone knows that they make a difference.

MOVING FORWARD – THE BEST YET TO COME

The company is now looking ahead with renewed confidence. Medium term plans to 2010 are being developed, which will see the mine extended to a depth of 1.8km below the surface.

There is great confidence about the potential for further ore discoveries within the existing mine, and on the 700 square kilometers of exploration tenements granted to CMPL.

Far from being in its twilight years, there is a growing sense that the best years for the CSA Mine lie ahead.

The future is being built on people, working with purpose. Prosperity for the organization, its owners, employees and the communities in which it operates is now a reality.

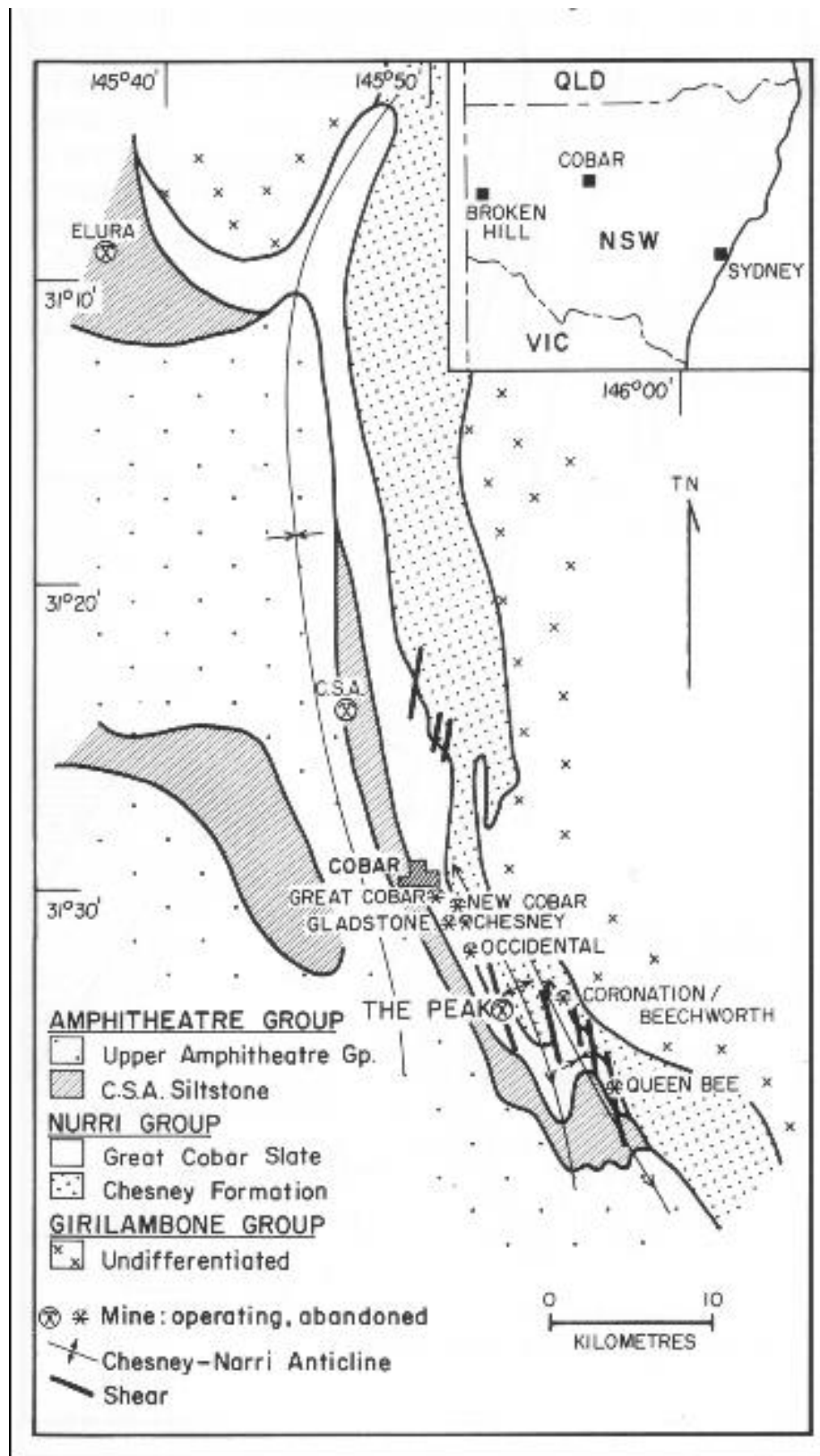


Fig 1 – Generalised geology of the Cobar area and location of the main ore deposits. [Source: Hinman, M.C. and Scott, A.K., 1990. The Peak gold deposit. Cobar, in *Geology of the Mineral Deposits of Australia and Papua New Guinea* (Ed. F.E. Hughes), pp. 1345-1351 (The Australian Institute of Mining and Metallurgy: Melbourne)]

DENDROBIUM MINE: FROM PAPER TO PRODUCTION

Peter Whittall ¹

INTRODUCTION

In July 2000, BHP Billiton Illawarra Coal announced its intention to conduct a feasibility study into establishing a new mine in Mt Kembla, New South Wales. The mine was needed to replace a specific grade of coal currently being supplied from the Company's Elouera Colliery. The Dendrobium Mine Project (incorporating the mine and the associated infrastructure), having undergone a Commission of Enquiry, was given approval by the Department of Planning in November 2001 and by the BHP Billiton Board of Directors immediately thereafter. The Dendrobium Mine became an official entity on December 23, 2001 by a redistribution of Kemira, Mt. Kembla and Elouera Mine holdings.

The project and the mine were named for the Dendrobium, a genus of orchid native to the area, the name of the parish in which the mine holdings are located and, for many years, the name given to the area of coal to the north west of the Wongawilli and Nebo workings.

The Dendrobium Project has been seen as a benchmark in a number of areas including community consultation. This paper, while referring to a range of areas, will predominantly focus on the planning of the mine, the mining and engineering challenges and successes the mine has achieved to date, and some of the learnings from undertaking to create a new underground coal mine on the Illawarra escarpment.

BACKGROUND

BHP Billiton's Illawarra Coal Holdings (and previously BHP (AIS Steel Pty Ltd)) has owned and managed many underground coal mines in the Illawarra area since the operations at the steelworks at Port Kembla began in the late 1920s.

Attrition of coal reserves along the escarpment has seen the majority of BHP Billiton's mines close or merge with neighbouring mines to obtain synergies in infrastructure and access. Prior to Dendrobium, BHP Billiton's last new mine openings were the Cordeaux and Tower Mines in the late 1970s.

BHP Billiton Illawarra Coal currently operates the Appin, Elouera and West Cliff Mines, with Appin and West Cliff mining the No.1, or Bulli, Seam, and Elouera mining the No.3, or Wongawilli, Seam. These mines produce premium coking coal for the steel making industry and it is a blend of the Bulli and Wongawilli Coals that produces the best product.

Elouera Colliery (created from the Wongawilli and Nebo Collieries) dates back to 1927 and has mined extensively by both pillar extraction and more recently by longwall extraction. Large faults to the north of the current mining area restrict further development and has forced the Company to look to an alternate source for the Wongawilli Seam product to meet the blending needs of the Bulli Seam mines. Elouera supplies about 20 - 30% of the Steelworks coal (approximately 1 Mt per annum of clean coking coal)

¹ Manager Special Projects, BHPBilliton Illawarra Coal

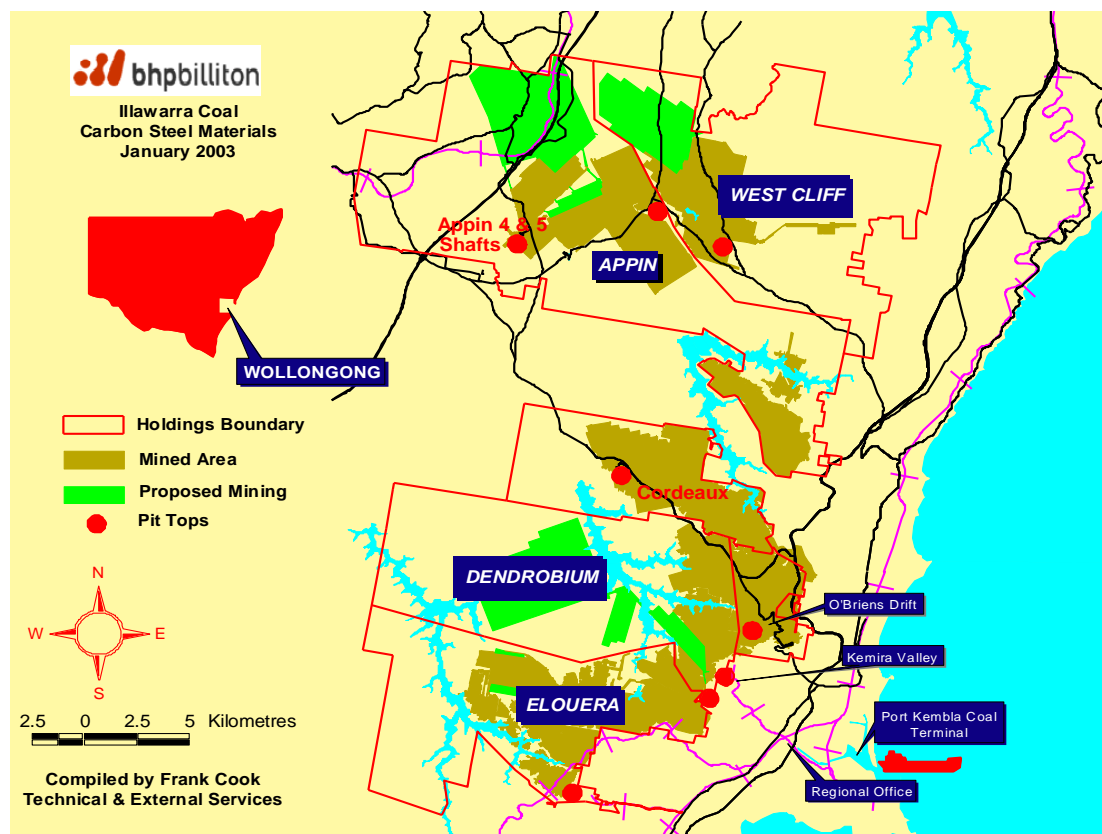


Fig 1 - Illawarra Coal, Mining and Infrastructure Layout

The concept and pre-feasibility studies for the Dendrobium Project investigated alternate sources for the Wongawilli Seam-type coal as well as alternative steel making opportunities that would reduce the need for this type of coal in the blend. The alternative steel technologies were not viable. The Wongawilli Seam is of similar reflectance to the Bulli Seam Coals (approximately 1.23%) but of higher vitrinite. At 80% vitrinite the Wongawilli Seam is among the highest ranked coals in Australia. The Wongawilli coal is also an order of magnitude lower in phosphorous (0.007%) than the Bulli Seam coals of Appin and West Cliff. Although similarly ranked coals were available from several sources in Queensland, those coals did not have similarly low phosphorous levels to compensate for the Bulli Seam in the blend. Sourcing coal from a remote source rather than locally also significantly impacted the Steelworks viability due to transport and handling charges as well as infrastructure upgrades required at the port.

The recommendation of the pre-feasibility was to pursue to development of a locally sourced Wongawilli Seam product. The Nebo Portal of Elouera Colliery (formerly the pit top for the Nebo Colliery) was chosen as the surface location of the new mine as it provided existing infrastructure and reasonable access to the resource.

The area of coal available to the Dendrobium Mine lies within the Wongawilli Seam of the Illawarra Coal Measures. Access to the larger reserves away from the escarpment is via a relatively narrow corridor of coal bounded to the south by Elouera's Nebo workings, to the north-east by the abandoned Kemira Colliery and vertically above by the abandoned Mt Kembla workings in the Bulli Seam.

The scope of the total Dendrobium Project incorporates:

- The development of the new Dendrobium Mine (including the upgrade to the old Nebo Colliery surface facilities as the Dendrobium Pit Top);
- The establishment of a 150,000 tonne raw coal stockpile in the adjacent Kemira Valley;
- Coal handling facilities in Kemira Valley for stockpile reclamation and train loading;
- Upgrade to the existing rail link between the Kemira Valley and the coal washery located within the Steelworks site;

- Upgrade to the coal washery for increased tonnage and a move to wash only Dendrobium Coal;
- Establishment of a Coal Drier at the washery to reduce the moisture content of the finer Wongawilli Coal;
- Management of the West Cliff emplacement area to receive Dendrobium coal wash (the Elouera emplacement area will discontinue with the closure of Elouera).

Following BHP Billiton Board Approval in December 2001, work commenced on the new mine site in February 2002, tunnels commenced underground on 28th May 2002 and longwall production is targeted to commence in late 2004.

FEASIBILITY STUDY DOCUMENT

BHP Billiton Capital Investment System

The Dendrobium Project was evaluated using BHP Billiton's Capital Investment System of review based on the Investment Modules for Concept Study, Pre-feasibility Study, Feasibility Study and Execution Phase. Each module sets out the requirements for analysis and the level of detail and accuracy for both technical and financial assessment. The Feasibility Study comprises 21 chapters which are required to be addressed:

- Executive Summary and Recommendations;
- Strategy;
- Market Analysis;
- Risk (Business and Safety);
- Geology and Mineral Resources;
- Mining;
- Mineral (Coal) Processing;
- Infrastructure;
- Human Resources;
- Project Execution;
- Asset Management;
- Information Management;
- Safety;
- Environment;
- External Relations;
- Capital Costs;
- Operating Costs;
- Ownership, Legal, Contractual and Finance;
- Financial Analysis;
- Project Status and Reviews; and
- Work Plan to next decision.

The Capital Investment System also requires the use of "Peer Review Teams" and "Tollgate Reviews". The Peer Review Teams comprise internal Company subject experts who are charged with reviewing a project for both the technical and financial evaluation methodologies and also the deliverables from the project. The Tollgate Reviews are an opportunity for the Peer Review Team to evaluate the progress of the Project Study.

More specifically, the purpose of the tollgate process is to ensure:

- That critical decision and project parameters are addressed prior to committing funds, and
- Each project meets the strategic, technical and investment requirements of BHP Billiton;

The Tollgating process incorporates a progressive review of project development and facilitates:

- Utilisation of appropriate expertise from other areas of the Company, or from external sources, to enhance project delivery;
- Re-assessment of project deliverables/financial criteria for investment in the light of changes in external circumstances, such as market conditions, and
- Cessation of a project subject to requests for more risk management work.

Ultimately, at the conclusion of the Feasibility Study, the Peer Review Team's endorsement and recommendations are required before a project can be submitted for funding by the Company.

Dendrobium underwent a number of Peer Reviews on every aspect of the project utilising varying groups of reviewers. The mining and mine engineering reviews for the project were held in November 2000, February 2001 and June 2001, prior to final tollgate.

Mining And Mine Engineering Study Scopes

During the previous stages of the Dendrobium Project, a mine plan had been conceived and evaluated to a standard sufficient to allow the Dendrobium Mine option to be chosen over other options discussed earlier. The task of the Feasibility Study is to take the selected option, in this case a new mine and associated infrastructure, develop the option from a technical perspective and also to address all of the issues presented within the chapters outlined above. The Feasibility Study is required to be evaluated to a +/- 10 - 15% range for both capital and operating costs prior to project sanction.

The scope of the Mining study included:

- Mine Planning;
- Critical Safety Risk Review of Mine Plan and Operations;
- Mine and Production Scheduling;
- Portal Construction and Drivage of Mine Access Tunnels;
- Ventilation Design;
- Shaft Construction;
- Main Fan Selection;
- Gas Drainage;
- Roadway Development;
- Hydrology - inrush prevention and prediction, mitigation of mine water accumulation risk;
- Subsidence;
- Strata Control;
- Management Plans - Statutory and Critical Risk;
- Mine Operating Costs; and
- Underground "issues" not otherwise covered.

The scope of the Mine Engineering study included:

- Mine Services - compressed air, fresh water supply, underground power distribution;
- Surface Conveyor and interaction with Kemira Valley Coal Handling Facilities;
- Pit Top Facilities and Building works program;
- Surface Power Substations and distribution;
- Mine Equipment Requirements
 - Mining Equipment - Development and Longwall;
 - Underground Conveyor System;
 - Internal company availability, new equipment requirements, equipment sizing.

The scope of this paper precludes an in depth analysis of all of these topics, although each one has required significant work in designing the mine and bringing it to production level. The intent is to look at some of the more critical issues that have affected the implementation of the operations to date.

MINE PLANNING AND OPERATIONS

Mine Planning

As previously identified, the pre-feasibility mine plan was designed sufficiently robust from a mining engineering perspective to allow evaluation as a viable mine plan.

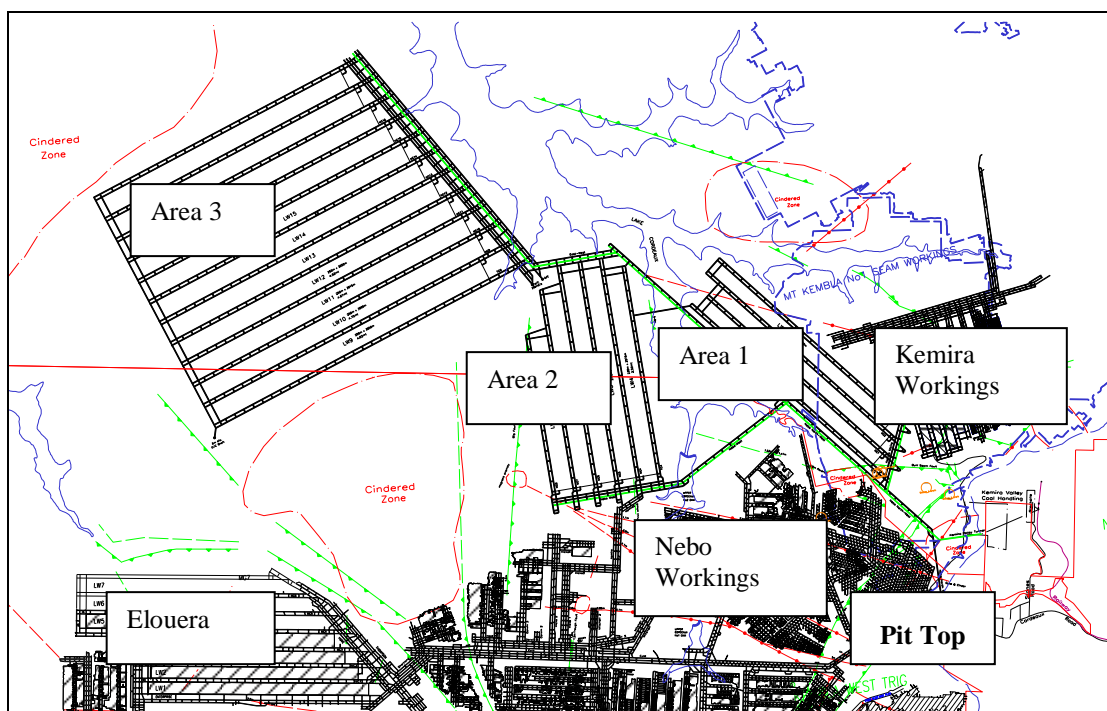


Fig 2 - Dendrobium Mine Plan November 2000

Figure 2 shows the Dendrobium Mine Plan effectively as it was at the commencement of the feasibility. The key physical constraints on the mine geometry are:

- Surrounding colliery workings of Kemira, Elouera (Nebo) and Mt Kembla Collieries;
- Cordeaux Dam and storage;
- Igneous Intrusions - Nepheline Syenite Intrusion (between Elouera longwalls and Dendrobium Area 3), Cordeaux Crinanite (a basalt like intrusion which will restrict Area 2); a large intrusion to the north west of Area 3; and a sill and dyke structure to the north of Area 1 which bisects the Kemira workings and Area 3; and
- Structures - large (approx 10m) fault to the west of Area 2; fault clusters close to the escarpment in Area 1.

A significant difficulty experienced during the mine planning process was that the review of previous exploration data, had not been completed and an active exploration program of boreholes and surface seismic was being conducted as part of the feasibility study. The upside was no doubt improved knowledge of the holding and greater confidence in the end result. The downside, for the mine planning and scheduling process, was the constant "truncations" to the mining areas caused by identification of significant structures.

The restricted mining areas has meant that geophysical parameters such as stress direction, roof strength variability and coal quality variations, have, for all intents and purposes, had to be dealt with as outcomes of the chosen mine layout rather than drivers of it. Area 3 is the only area with sufficient flexibility to allow optimisation against stress orientation. Figure 3 shows the principal stress directions across the mine. Mining to date has supported these stress predictions although there has been significant variability in direction and magnitude around the "pit bottom" area associated with localised structures.

Area 1 is oriented in what would normally be deemed to be the worst stress direction for both the gate roads and the longwall retreat. A mine schedule was run whereby Area 1 was bypassed and Area 2 mined first. This option was rejected based on poor financial return as the lead times and start-up costs before longwall coal were prohibitive. The alternative has been to budget for significant amounts of secondary support in the Area 1 gateroads.

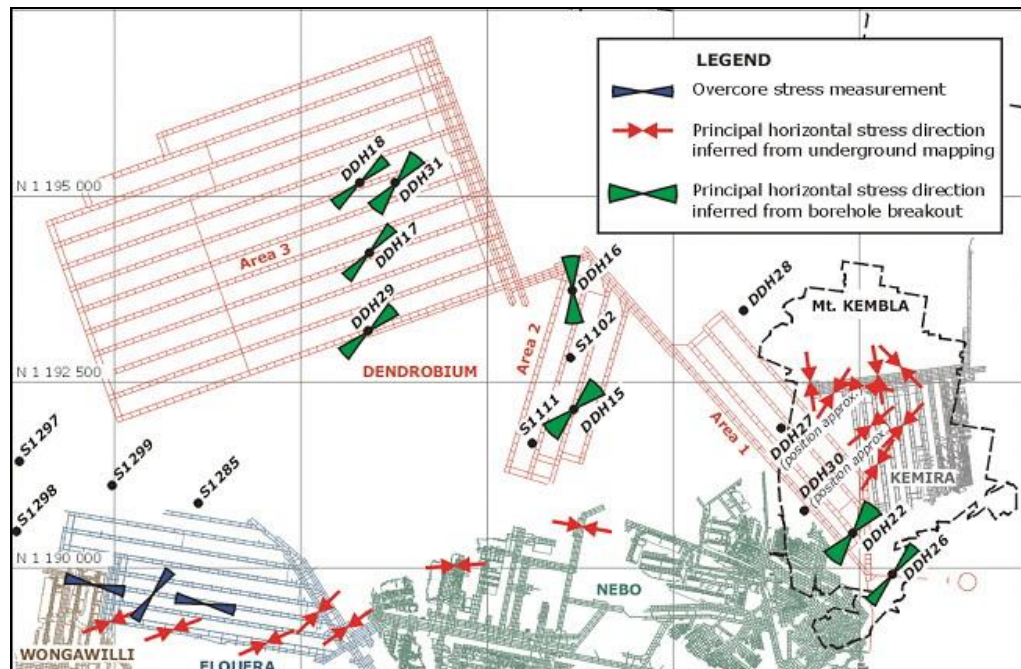


Fig 3 - Principal Horizontal Stress (source Strata Control Technology 2001)

Figure 4 shows the mine plan as it is currently being executed. Significant changes between the plan shown in figure 2 (November 2000) and this plan (August 2003) and their reasons are:

1. Portals and Drifts
 - a. Dendrobium Tunnel (Personnel and Materials)- the location of the portal has been relocated some 100m closer to the pit top to accommodate a drift section to the American Creek Seam;
 - b. Kemira Valley Tunnel (Belt Drift) - the portal for this tunnel had several options but the final location produced a tunnel which commenced approximately 60m below the Wongawilli Seam and drifted upward at 1 in 20 to intersect the seam. The orientation of the tunnel allows the main conveyor to deliver from pit bottom to the top of the stockpile rill tower.
2. Area 1 - Kemira Mains, servicing the Area 1 longwall entries, has moved away from pit bottom because the longwall takeoff points are defined by the Dam Safety Committee restrictions on mining under the Cordeaux Storage. From a cost benefit it was a fairly neutral decision although there were increased belt installation costs, the reduced overall drivage took days off the longwall start date which is the critical path for the project.
3. Area 1 - Longwall reduced from 3 x 180m blocks to 2 x 240m blocks. This saved one set of gateroads and gave a better subsidence profile for the same extraction thickness.
4. Longwall 2 truncated to the north due to dyke and sill;
5. Area 2 - no longer connected to Nebo Workings. The longwalls have each been shortened due to the presence of the Cordeaux Crinanite and a significant amount of net development drivage has been saved. The trade-off has been that an additional 20km of roadways need to be driven towards Area 2 prior to longwall 1 startup to ensure continuity going from longwall 2 to longwall 3.
6. Area 3 - exploration confidence has allowed these longwalls to be lengthened to 5200m with some reorientation to optimal stress direction.

The mine plan will no doubt continue to evolve at a micro level to accommodate geological anomalies as well as scheduling imperatives. As the plan shows, however, there is not a large discretion for significant changes to the layouts of Area 1 or Area 2.

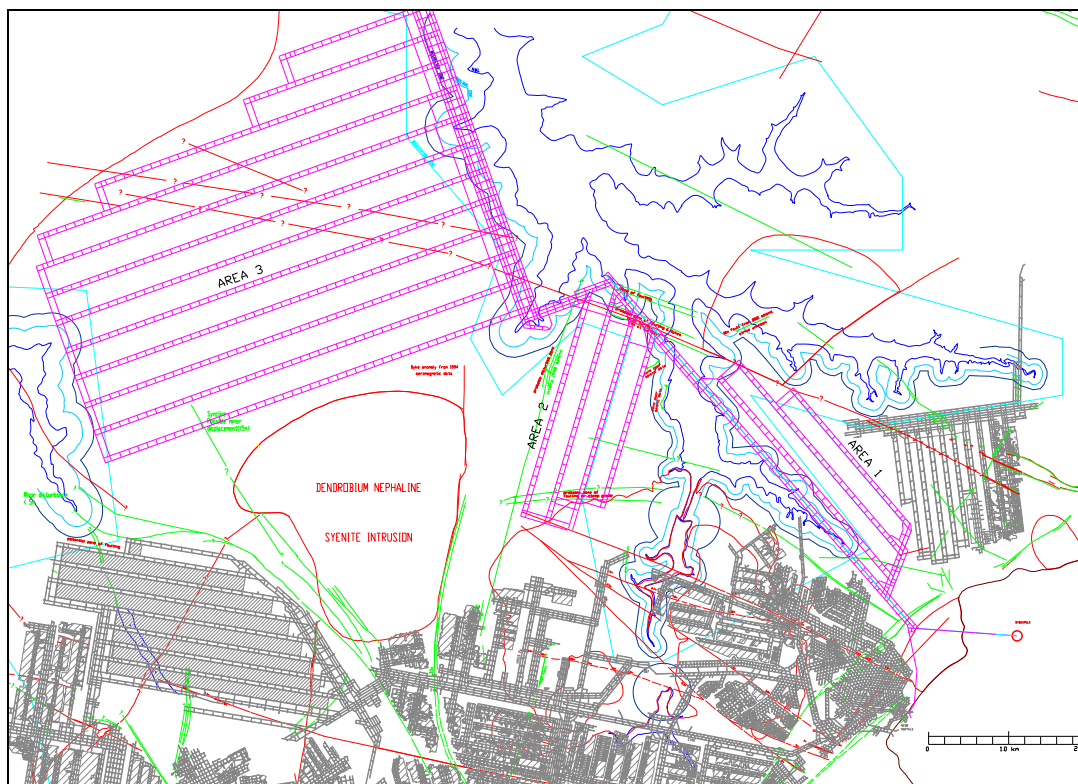


Fig 4 - Dendrobium Mine Plan - August 2003

Critical Safety Risk Review Of Mine Plan And Operations

In February 2001 the Project Team conducted a comprehensive safety risk assessment of all mine operations. The review started with the construction phase of the surface infrastructure and portals and covered all aspects of underground operations through to longwall extraction.

The purpose of this Critical Risk Assessment was:

1. Identification of critical hazards
2. Assessment of the risk of Dendrobium Mine to BHP Billiton;
3. To provide qualitative analysis of single or multiple fatality potential events and thereby identify key hard barriers for financial evaluation during the feasibility study;
4. Identification and verification that risk reduction strategies are in place to render the critical risks to an acceptable level;
5. To identify management systems which need to be developed at the mine as a matter of priority in preparation for commencement;
6. To identify any features of the mine plan which may, by their nature, be introducing significant risk to the mine where an alternative may be available;
7. To identify key risks and groups of risks to allow quantitative evaluation against Company, District, State, National, and International industry and multi-industry data;
8. Development of project Individual Risk Per Annum (IRPA);
9. Comparison of IRPA values with BHP acceptability criteria, other projects and historical probability of fatality; and
10. Documentation of management systems relating to individual risks by means of a risk register.

The primary objectives of the Critical Risk Assessment were:

1. Demonstration that the project has identified critical risks and has adopted risk reduction strategies and systems which will manage those risks; and
2. Verification that the risk reduction strategies will be effective in managing those risks to a level which is acceptable to BHP.

The risk assessment involved both Company and Consultant representatives. An initial review identified the main categories for assessment such as gas, ventilation, inrush, fire, strata control, underground transport, rail operations, asphyxiation, single entry and emergency response. Individual sessions were then run specifically for each risk category with a targeted review group. Only the Mine Manager, OH&S Manager and the facilitator were common to every session which included between eight and fifteen people depending on the session.

This was an excellent process to undertake early in the mine design phase. In one two-week period (plus processing time), a safety roadmap was established for the construction and operations phase with comprehensive involvement from a broad group of operational and technical experts.

The key safety mining risks were targeted for further analysis by Quantitative Risk Assessment (QRA). The objectives of the QRA were:

1. Assessment of risk on a statistical basis of probability;
2. Development of IRPA data for the project;
3. Comparison of quantified risks against BHP Billiton acceptability criteria;
4. Comparison of relative risks across different projects;
5. Comparison of relative risks across different industries;
6. Comparison of relative risks before and after implementation of risk reduction strategies; and
7. Tracking of changes in the risk profile with time.

The QRA process involved:

1. Assessment of the annual probability of an initiating event based on industry statistical data eg. gas explosion from 1993-2000;
2. Analysis of the downstream events which will dictate survival or otherwise by means of an event tree;
3. Assignment of probability to after the event circumstances, decisions, survival equipment and actions;
4. Determination of outcomes (fatality or non-fatality);
5. Combination of probabilities in order to derive probability of fatality (IRPA)
6. Comparison of site IRPA value with BHP Billiton acceptability criteria.

Acquisition of suitable data was difficult in this process and deriving meaningful results which enhanced the safety of the operations is a point for further discussion. In summary, the qualitative risk assessment provided an excellent platform for the development of safety systems and the management risk at Dendrobium well into the operational phase. The quantitative risk assessment allowed the project to be evaluated against other investment options for the Company based on their relative risks to our employees and contractors.

A strong culture of risk assessment has been established at Dendrobium.

MINE PRODUCTION AND SCHEDULING

At the time that Dendrobium was being evaluated, Illawarra Coal was operating an in-house developed scheduling package based on Excel macros. Each of the mining engineers within the Group were proficient on this tool and Illawarra Coal was still evaluating an alternative commercially available scheduling package. The whole of the project through to current operations has been scheduled on this package which has proven over the years to be reliable, powerful and flexible. To ensure consistency through the implementation phase, Dendrobium is still running its schedules on this package although, like the rest of Illawarra Coal, Dendrobium has commenced using a commercial scheduling package in parallel.

Key features of the mine's schedules have been:

1. Each mine plan layout has been run on an expected best (P10), expected (P50) and expected worst (P90) case;
2. Every schedule must be accompanied by accurate notes on assumptions and strategies to allow comparison of outcomes between options (also helped as several engineers were used over the early stages of the project to assist with scheduling);
3. Longwall continuity was kept at +100 days in all scenarios for all layouts;
4. Development rates were outcomes of industry benchmarking and heavily weighted to outcomes of neighbouring mines in similar conditions. No "upside" was allowed for "new mine, fresh work force" types of intangibles. Consequently development rates were perceived as conservative;

5. Development panels modelled on Dendrobium projected advance rates irrespective of Contractor development predictions;
6. Insufficient ramp up time was allocated in the schedule for new panels/workforce starting up in confined panels with no "pit room". Dendrobium has experienced significant delays in early stages of development due to inefficiency of operations. These were not adequately allowed for in the schedule.
7. Similarly outbye tasks such as pumping and secondary support impact directly on face operations due to lack of outbye labour and the proximity of outbye and face operations. This has improved as the mining faces have progressed inbye.

PORTAL CONSTRUCTION AND DRIVAGE OF MINE ACCESS TUNNELS

The Dendrobium mine is accessed via two cross measure drift and in-seam tunnel combinations, Dendrobium Tunnel and Kemira Valley Tunnel.

Dendrobium Tunnel

To access the Dendrobium Holdings from the pit top, it was necessary to drive a tunnel in approximately a northern direction along the escarpment to skirt the old Nebo Colliery workings. Several options for the portal location and mining horizon were investigated. A drill rig was located at the extremity of the pit top bench with the intent of drilling along the tunnel orientation and proving the ground. The drill rig encountered impassable ground after 400m of drilling. No amount of grout or branching was able to progress drilling past this point. A few surface bores on the hillside above the portal added to the conclusion that at least the first 400m would be in dubious ground with the strata above the seam eroded and replaced by talus slope. Lack of access to the surface above the remainder of the tunnel due to terrain and government approvals led to the decision to drift down to the American Creek (AC) Seam, approximately 8m below the floor of the Wongawilli Seam, via a 1 in 10 drift commencing at the portal on the floor of the Wongawilli Seam.

The portal was preformed against the hillside and then forepoles were drilled in a ring around the portal. Removal of the initial material was by excavator until the roadheader could be positioned. Arches were used and shotcreted to form a smooth lining for the first 15m.

The Dendrobium Tunnel shown in Figure 5 commenced portal construction in April 2002 and went underground on June 3, 2002.



Fig 5 - Dendrobium Tunnel - June 2002

In this area the AC Seam is banded coal over about two metres of shales and is not commercially recoverable. The weathering was found only to extent to the top of the Wongawilli Seam. The interburden consists of 80 - 100Mpa sandstone and provides an excellent roof for the tunnel. The tunnel was driven to keep approximately 1m of this sandstone in the crown. Maintaining a constant downgrade over the length of the tunnel for long term water drainage meant the variations in the strata impacted on the this horizon and often large faces of sandstone slowed progress. The majority of structures in the tunnel were able to be projected from Nebo Colliery plans in the Wongawilli Seam. Development of this tunnel was by S200 roadheader and diesel ramcars.

The tunnel stayed in the lower AC horizon for some 800m before drifting back up at 1 in 10 to the floor of the No 3 seam approaching pit bottom.

Kemira Valley Tunnel (KVT)

The Kemira Valley Tunnel shown in Figure 6 connects the underground operations with the coal handling facilities in Kemira Valley and contains the mine's trunk conveyor. The portal is located on the hillside about 60m above the valley floor and allows the trunk belt to exit the mine and be delivered to the top of the 150 Kt stockpile rill tower in a single flow. Unlike the Dendrobium Tunnel, the KVT commenced from a box cut. The underground portion of the tunnel commenced on May 28, 2002 and is approximately 1150m in length. The first 900m is a cross measure drift rising at 1 in 20 and then flattens out in the Wongawilli Seam towards pit bottom.

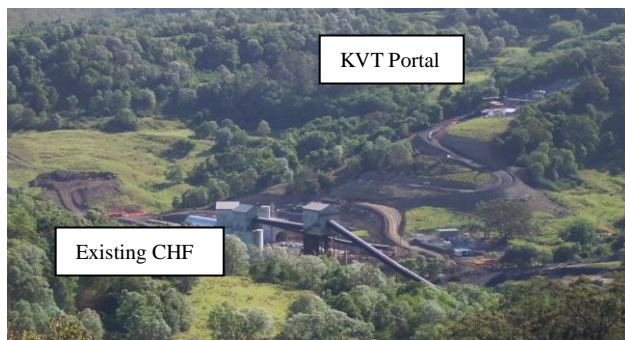


Fig 6 - Kemira Valley Works

The coal industry, especially in the Southern Coalfields, is becoming expectant of onboard bolting and positive temporary roof and rib support systems.



Fig 7 - AM 105 Roadheader - KVT July 2002

Hard rock industry roadheaders are not designed to provide this, nor are coal mining machines designed to mine sandstone tunnels. This presented a high risk area of work for the mine and also the first serious incident when an operator, supporting the roof of the KVT from the boom of the AM105 shown in Figure 7 was struck by falling rock from a greasy back on a thrust fault facing. Several attempts were made to retrofit hydraulic rams to the boom for temporary support without success.

Applying roadheader equipment and support patterns to coal measures, especially when driving cross measure, provided a long term stable roadway but presented challenges for the safe operation of the panel and support of the roof.

The two tunnels were eventually holed within the coal seam by continuous miner without further incident but having installed a significant number of additional "intermediate" bolts to ensure workers were not exposed to unsupported roof during construction.

VENTILATION DESIGN AND FAN INSTALLATION

The change to the mine plan excluding connection to Nebo in Area 2 also meant that Dendrobium would not have access to the Nebo shafts for upcast or downcast ventilation. The first step in designing the ventilation for the mine was to understand the in situ gas environment. The gas reservoir parameters used for the gas emission modelling were derived from analysis of field and laboratory testing conducted by BHPIC, Sibra, Multiphase Technologies Pty Ltd (Multiphase) and CSRIO. Geological data was provided by BHPIC.

The gas content and composition characterisation for the area was limited to 8 boreholes. Compared to normal industry practice, this is a small data set.

Using their gas emission models, GeoGAS ¹ provided gas emission estimates for Areas 1, 2 and 3 of the Dendrobium Project. The gas content in Area 1 was determined to be 2 - 5 m³/t CH₄ and in Area 2 4 - 5 m³/t

¹ GeoGas Pty Ltd report to Dendrobium, July 2001

increasing in CO₂ composition (approx 30%) in the north. Area 3 presents a difficult challenge in gas environment. A dyke structure traversing the area literally splits the mining area into two zones. The Area south of the dyke indicates a gas regime of up to 6m³/t CH₄. The area immediately to the north of the dyke indicates up to 17m³/t of CO₂. There appears to be little transition between the two with the dyke acting as the delineating feature.

The ventilation modelling was conducted by Australian Coal Mining Consultants Pty Ltd (ACMC) ¹.

The total development split requirements for the dilution of gas emission are determined by the greater of:

- Satisfaction of statutory auxiliary fan requirements. The minimum quantity to be available at the fan site is the sum of the open circuit capacity of each auxiliary fan operating in a panel and 30% of the open circuit capacity of the largest auxiliary fan in operation in the panel.
- Dilution of the intake emission to the last cut-through to the design basis
- Dilution of the total panel emission outbye in the return to the design basis.

These are examined independently in the following sections.

Development Ventilation Requirements

Development Face Area Gas Emission Ventilation Requirements

Auxiliary ventilation simulations were established for a twin entry 120m x 50m development panel with the return standing heading driven and the cut through (driven from the intake heading) almost holed. Permutations of 610mm and 762mm diameter fibreglass ducting and 13 m³/s and 18m³/s open circuit auxiliary fans were examined.

A face area design basis of 0.8 % CH₄ for the gas emissions predicted by GeoGAS, would require an 18m³/s open circuit auxiliary fan coupled to 610mm diameter ducting to be specified. This combination would deliver 6.3 m³/s to the face heading with the standing heading regulated to 4 m³/s. The auxiliary fan would be operating at a duty point of 13.5 m³/s at 4.3 kPa.

A minimum of 23.4 m³/s would be required at the fan site to support an 18 m³/s open circuit fan therefore 25 m³/s delivered to the last open cut through was adopted as the minimum design basis for the primary ventilation system.

Development Panel Gas Emission Ventilation Requirements

The following were used as the design basis for determination of the development panel ventilation requirements:

- General body contamination at the start of the hazardous zone of 0.25% CH₄
- General body contamination in the returns of 0.8% CH₄

Ventilation quantities were determined as being required at the last line of cutthroughs for a number of milestones in the mine's life. Between Maingate (MG)2 and MG6 this requirement increased from 40m³/s to almost 80m³/s.

The resultant ventilation quantities were those required to manage all gas emission. It may not be possible to deliver such ventilation quantities to the working places due to excessive airflow velocities and ventilation pressure losses. Gas emission not able to be sufficiently diluted would be managed through seam gas drainage or capture.

Longwall Panel Ventilation Requirements

The following were used as the design base for determination of the longwall panel ventilation requirements:

¹ Australian Coal Mining Consultants Pty Ltd report to Dendrobium, July 2001

- Face general body contamination - 0.8% CH₄
- Longwall return general body contamination - 0.8% CH₄
- Bleed return general body contamination – 1.5% CH₄.

It was assumed that a minimum average ventilation velocity of 2 m/s is required for the control of airborne dust. Assuming a longwall airway effective cross sectional area of 15 m², a minimum longwall face ventilation quantity of 30 m³/s would be required for airborne dust control per se.

Volumes for Longwalls 2 to 6 ranged from 37 to 40 m³/s on the face and 20 to 30 m³/s for return quantities.

Ventilation Airflow Velocities

The design of ventilation systems have traditionally been undertaken with the resultant ventilation airflow velocity maximums being 2.5 m/s in belt roads, 4 m/s in intake airways road and 6 m/s in return airways. This is for reasons of minimising dust generation and pressure losses.

The use of these velocities effectively limited the Dendrobium mine intake quantity to 156 m³/s whilst the Dendrobium Tunnel and Kemira Valley Tunnel are the only intakes to the mine.

In order to mitigate against this limitation, ACMC used maximum velocities as a design basis, which are generally 50% greater than those traditionally used. These are 3.8 m/s in belt roads, 6 m/s in intake airways and 9 m/s in return airways. In addition 6 m/s was used as a maximum for homotropical belt roads, being the lesser of the maximum intake roadway airflow velocity and the sum of the maximum belt road airflow velocity plus an assumed belt speed of 4.1 m/s.

Conclusions of Ventilation Modelling

Table 1 gives the pressure/quantity outcomes for a number of milestones throughout the mines life.

Table 1 – Pressure Quantity Values

Milestone	Description	Collar Ventilation Quantity m ³ /s	Collar Static Pressure Pa
1	LW1 at start up, #1 shaft as upcast	190	1,396
2	LW2 at start up, #1 shaft as upcast	220	3,347
3	LW2 at start up, #1 shaft as upcast and #2 shaft as downcast	296	4,454
4	LW3 at start up, #1 shaft as downcast and #2 shaft as upcast	212	2,490
5	LW4 at start up, #1 shaft as downcast and #2 shaft as upcast	305	3,545
6	LW6 at start up, #1 shaft as downcast and #2 shaft as upcast	188	1,804
7	LW11 at start up, #1 shaft as downcast and #2 shaft as upcast	248	5,407

The conclusions of the review were as follows:

1. The results of the GeoGAS emission modelling and the ACMC ventilation modelling indicate the following for development panels:
 - Face area gas emissions in all areas would be managed using an 18 m³/s open circuit auxiliary fan with 610 mm diameter ducting without the requirement for gas drainage. Where a super unit panel is to be employed, a second auxiliary fan would be required to support the second unit.
 - Gas capture and/or drainage would be required where 55 m³/s cannot be delivered to the last cut through of Area 2 development panels. Ventilation modelling indicates that this would occur in TG3 and MG3 prior to the commissioning of the #2 shaft as a down cast shaft. Prior to the commissioning of the #2 down cast shaft, the ventilation specification has allowed for 25 m³/s to be delivered to the last cut through in Area 2 gate roads. This is sufficient to dilute 62.5 l/s of intake emission and 200 l/s of return emission to the design basis.
 - Gas capture and/or drainage would be required in Area 3 for gateroad development. The ventilation specification has allowed for 50 m³/s to be delivered to the last cut through of the Area 3 gate roads. This is sufficient to dilute 125 l/s of intake emission and 400 l/s of return emission to the design basis.
 - A deviation in the actual development and extraction rates from those used in the GeoGAS gas emission modelling would result in variations in the actual compared to modelled gas emissions. If the variations were positive, additional airflow or gas drainage/capture capability would be required to meet design bases.
 - For milestone 3, increasing the airway frictional coefficients of resistance by 25% would reduce the ventilation quantities delivered to TG3 and MG3 by 15%. Increasing the resistance to leakage of the ventilation control devices by 25% would result in a 4% reduction in the ventilation quantities delivered to TG3 and MG3. Combining both variations on the design basis would result in 19% and 18% reductions in the ventilation quantities delivered to TG3 and MG3 respectively and an 8% increase in the mine resistance.
2. GeoGAS longwall extraction emission modelling (for the most southern Area 3 panel) and the ACMC ventilation modelling indicated that with design bases of 0.25%, 0.8% and 1.5% CH₄ longwall intake, return and bleed general body contaminations, no post drainage would be required for longwall extraction.

The assumption that the emission estimates from LW6 represent that of successive longwall panels to the north is likely to be an optimistic one. Gas emission in excess of that which would be manageable by means of the ventilation airflow quantities would be handled through the use of post drainage.
3. Airway velocities are generally acceptable for all models with the exception of that in the Kemira Valley Tunnel during the extraction of Longwall 2 whilst Nebo Mains D heading is being mined. It is likely that some means of velocity mitigation would be required around transfer points and a conservative view would be to make provisions for the shrouding for the entire tunnel conveyor at a later date if proven required.
4. Whilst the main fan is located at the #1 shaft collar, the collar duty would vary from 190 m³/s at 1.4 kPa to 300 m³/s at 4.5 kPa. This would require 370 kW to 1.83 MW motor power capacity. Whilst the main fan is located at the #2 shaft collar, the collar duty would vary from 190 m³/s at 1.8 kPa to 250 m³/s at 5.4 kPa. This would require 471 kW to 1.86 MW motor power capacity. Motor power calculations assume an overall efficiency for the fan installation of 0.72 in converting electrical power to air power (including impellor, transmission and motor losses). As can be seen from the range of duties required, the fan or fans need to be able to perform across a wide range during the mine's life. The successfully tendered fans are twin centrifugal Flakt-Woods fans with Variable Speed Drives (VSD) (850Kw, 690V). Installation of the VSD drives allows reduced speeds for lower duties and therefore reduced operating costs.

Because of the low duty point required during start up only one fan has been installed with the second required during the extraction of Area 1. The two fans will be installed as stand alone units with their own control systems (interlocked during twin fan operation) to allow easy relocation to the No 2 Shaft during the LW 2 - 3 changeout. The first fan shown in Figure 8 is operating well at the low duty during initial drivage although there have been some problems successfully getting the fan to perform at maximum duty during testing for full commissioning. This is as yet unresolved.



Fig 8 - Dendrobium No. 1 Fan

SHAFT CONSTRUCTION

Three types of shaft construction methods were tendered for the construction the No.1 Shaft: conventional drill and blast; raise bore; and blind bore.

The conventional method was discounted on a number of criteria including workforce exposure during construction and use of explosives in close proximity to community residences. The decision was ultimately made to construct the shaft using a blind bore method tendered by Ardent Underground Pty Ltd. This method was more expensive than the tendered raise bore methods but has significant advantages which were more attractive to Dendrobium:

1. The shaft could be sunk, lined, and allow the fan to be installed prior to holing into the shaft from the mine roadways. This effectively took the shaft off the mines critical path for longwall start up;
2. The shaft stays full of water until ready for lining and therefore there is less opportunity for the wall of the shaft (some shales and claystones) to deteriorate;
3. Under most foreseen circumstances there would be no need for any person to enter the shaft below collar level. This was a safety objective of both Dendrobium and Ardent; and
4. The shaft material did not have to be dealt with by the mine's coal flow system.

Ardent supplied a fully refurbished and newly reconfigured drill rig and purpose built drill head for the shaft. Ardent opted to drill without a pilot hole and following the completion of the 8m pre-sink, the shaft was drilled to full depth of 180m (floor of the Wongawilli Seam) at 4.25m diameter in a single pass. The method required that the shaft be maintained full of water during drilling. A small diameter steel pipe running down the inside of the drill string delivered compressed air to the bottom of the drill string above the drill head and the released compressed air bubbled up inside the drill rods creating a negative pressure. This negative pressure, couple with the hydrostatic head of the shaft water was sufficient to lift all cuttings from the hole and "float" them up the inside of the drill.

Once the shaft was completed the drill head and rods were replaced with a shotcreting frame suspended in the shaft. The frame consisted of a rotating arm, fixed stabilisers, IS lighting and video cameras. The shotcrete was then applied by an operator in a cabin adjacent to the collar who was able to watch the shotcrete being applied via the video camera while operating the crane. Approximately 15m of head was bailed from the shaft at a time exposing 15m of shaft wall for lining. On completion of this section the next 15m was bailed.

This process did initially present some technical difficulties in getting a consistent application of the product but this was rectified with different pebble sizes and different nylon staple sizes. The most significant problem arose when an aquifer was intersected at approximately 48m below collar level. The water outflow was sufficient to wash the shotcrete off during application and the operation had to be suspended pending a solution in stopping the water flow. Several options were reviewed but a desire to not allow people into the shaft forced a solution of remotely stopping the flow.

The eventual solution was ingenious. A Pro-Ram was suspended in the shaft on a makeshift frame with telescopic stabilisers. Lights and cameras were mounted on the frame to allow remote location of the drill rig and remote operation via extended hose lines. A modified 1200mm drill was drilled into the heart of the water outflow using water flushing. Polyurethane was then pumped through the drill rod into the surrounding strata until the water flow ceased. There were a number of engineering "tricks" that needed to be employed to do what sounds like a simple task. The result was that the lining was able to be continued without further complication, no-one had to enter the shaft and the shaft was successfully lined to several metres from the base.

The shaft was completed behind schedule due to some drilling and lining problems but it was still far enough ahead of underground operations to allow the fan to be installed and pre-commissioned. The fan was turned on just prior to the miner holing the bottom of the shaft.

The only injury incurred during the sinking and lining of the shaft was a cut hand to an employee handling a piece of checker-plate. The shaft project was awarded a Merit Award in BHP Billiton's annual Health Safety Environment and Community Awards in 2003.

HYDROLOGY

There are four major sources of potential water ingress to the Dendrobium workings. These sources have inflow potential varying from the ability to create nuisance inflow requiring a pump out system to life threatening inrush potential. The four sources are:

1. Wongawilli Seam and escarpment ingress due to rain water and seam water migration (eg 3.8 l/s/km + 10l/s seam + high rain)¹;
2. Pondered stored water in the overlying Mt Kembla Bulli Seam workings;
3. Surface Stored Waters administered by the Sydney Catchment Authority and the Dam Safety Committee; and
4. Stored water against the extremity of workings in the Kemira and Nebo Wongawilli Seam workings.

Each of these sources and potentials must be evaluated based on the inflow or inrush potential of the resultant water to the mine and therefore the consequent risk to people, equipment and the operation.

Strategies must be designed to successfully mitigate both the life threatening and production hampering risks. During the Mining Peer Review process, inrush was identified as the single greatest risk to the Dendrobium personnel and operations. The Inrush Management Plan was required to be completed as part of the feasibility study prior to project approval.

The high water mark in the Nebo workings could be discerned by direct observation in Elouera. The high water mark in Mt Kembla can be inferred from a piezometer in a surface borehole located over the Mt Kembla workings. This hole had been used to successfully monitor water levels and to correlate underground measurements during research into the Kemira Colliery inundation in 1990².

The inundation at Kemira was due to a connection between the Kemira workings and the Mt Kembla flooded workings. The water level in Kemira can therefore be assumed to be consistent with the Mt Kembla water level.

Two critical components of the Inrush Management Plan are:

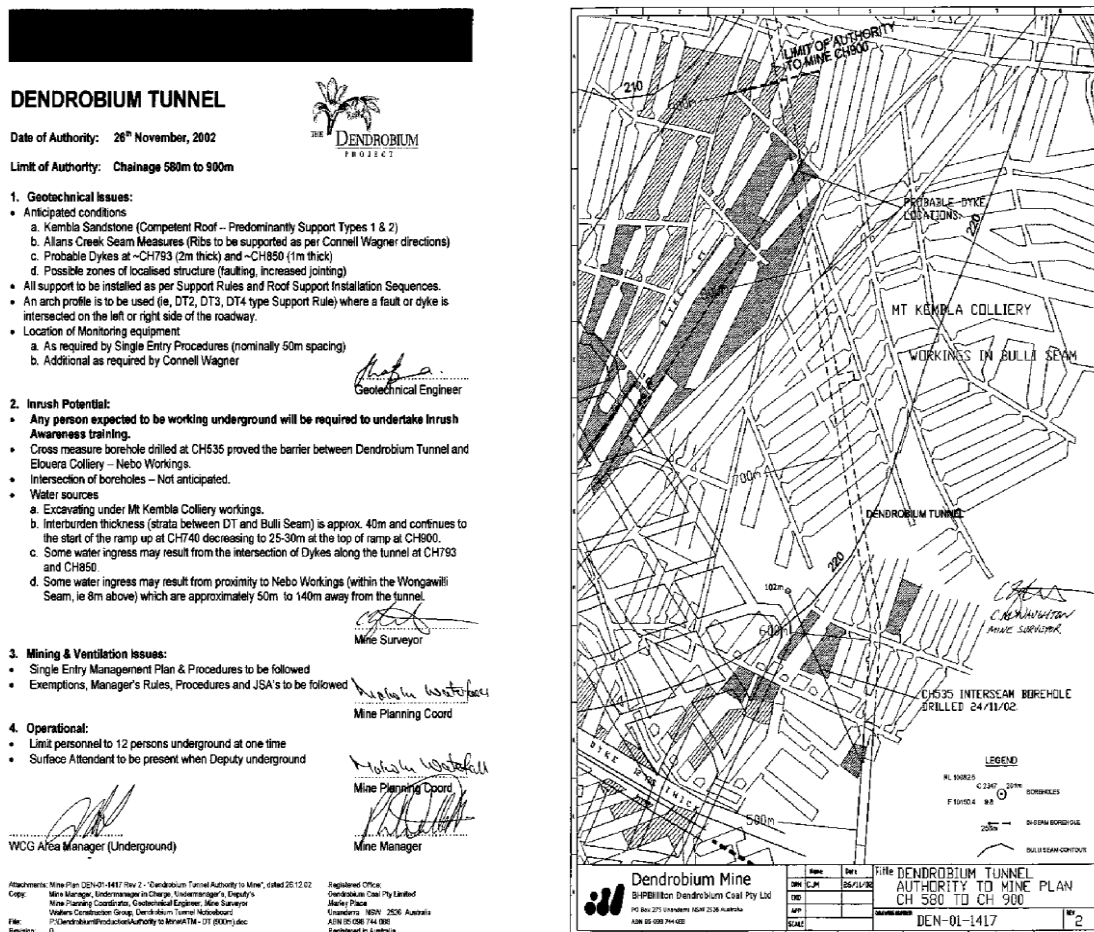
1. The Authority to Mine (ATM) process and document; and

¹ Seedsman Geotechnics Pty Ltd report to Dendrobium Mine July 2001

² Whittall, Peter - Hydrogeological Investigation and Analysis of a Major Inundation at Kemira Colliery, University of Wollongong Thesis, 1990.

2. Routine periodic drilling from the Dendrobium workings to the Mt Kembla workings to determine water level.

The Authority to Mine process is modelled on the Outburst ATM process used in the Illawarra No 1 Seam mines. The process requires a quorum of key people to review all relevant data and determine the extent of safe mining given the current data and then to sign off an ATM document specifying drivage configurations and distances covered under the Authority. No mining may take place without a current ATM in place in the panel.



All holes which intersect an open roadway in Mt Kembla are pressure tested to determine the head of water in the hole and samples of any water found are analysed to provide a fingerprint of the water sample to allow the water source to be determined.

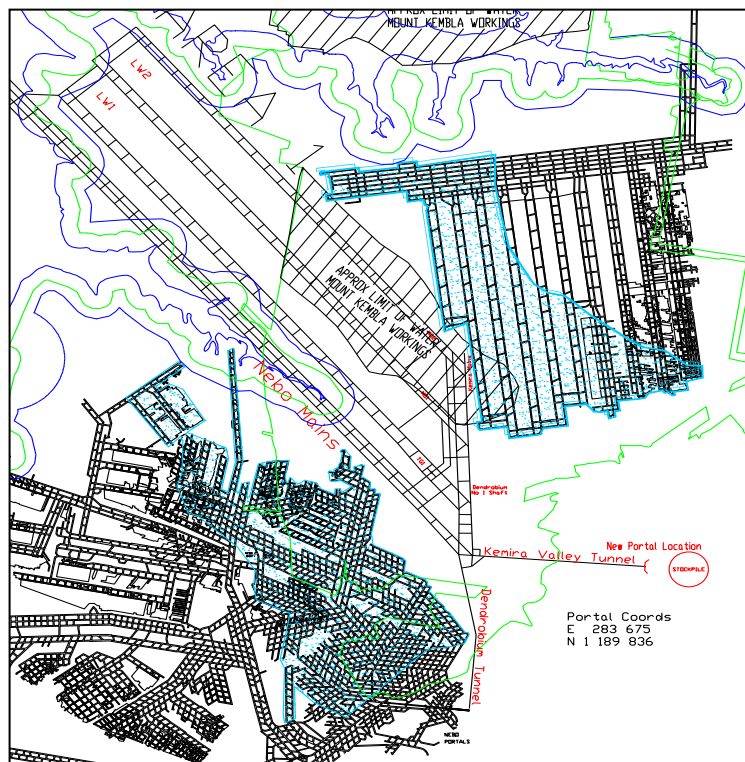


Fig 10 - Flooded Workings to 198.4m RL in Mt Kembla and Kemira Workings

A number of holes have been drilled into Mt Kembla which have indicated localised ponding in the workings but none have identified the high water mark of the main water body which is still ahead of the current workings. In December 2003 the first hole was drilled to connect with the Kemira workings and pressure testing confirmed the predicted water levels. This puts the volume of ponded water in the Mt Kembla workings at approximately 0.9GJ which will need to be drained prior to longwall extraction under the Mt Kembla workings. Figure 10 shows the position of flooded workings in the Mt Kembla and Kemira workings.

MINE ENGINEERING

The Mine Engineering scope of study in the feasibility study incorporated all underground engineering, surface conveyor and stockpile interaction, Dendrobium pit top facilities, main ventilation fans and surface power substations.

The philosophy taken in evaluating new and second-hand components of the engineering scope was that all fixed installations such as conveyors, power supply and fixed services should be right sized with contingency for future growth and purchased new. All transitory equipment which could be made to fit the shorter term needs of the mine and cycled out as their life or applicability diminished should be sourced internally from other BHP Billiton operations where possible.

Three models were developed and applied in determining the equipment sizing for the mine. The longwall model was developed by Australian Coal Mining Consultants (ACMC) to determine maximum outputs from the longwall to achieve the mines projected tonnages. The outputs of this model were used in both the conveyor design model and also the mine scheduling package. Lastly the conveyor model outputs were used to determine stockpile parameters and design.

Conveyor System

Continental Conveyors in partnership with Walter Construction Group successfully tendered for the supply and installation of the conveyor system at Dendrobium.

The key features of the conveyor system are:

1. Trunk Belts 1800mm 4500 tph capacity (6000tph peak) with bolt on drive units. Drive units are a combination of up to four 375KW and 450KW units which can be combined and mixed and matched to down or upgrade the belts as required;
2. Gateroad belts are 1500mm 3500 tph capacity (4200 tph peak). The decision to maintain gateroad structure at 1500mm was based on ergonomics. The 1500mm is still man-handleable whereas larger structure would require all machine operation for installation and recovery;
3. System designed on Voith PKL fluid couplings with PLC controlled fluid fill to give variable speed. VVVF drives were investigated during the feasibility study but discounted because of reliability issues. The PKL couplings give creep speed for inspections and slow start up to reduce load spikes. Half speed during development only is achieved using a different gear box ratio;
4. The rill tower forms the structural support for the tail of the KVT trunk conveyor. The rill tower is required to reduce coal degradation and reduce coal dust in the Kemira Valley. The design of this structure proved to be more complex than first conceived. The final installation appears to be operating satisfactorily to date.

Overall the conveyor installations have been successful and are performing as planned. The belts have not carried longwall coal yet.

Power Supply

Dendrobium is serviced by three separate power supplies:

1. The pit top is serviced by 6.6KV via the existing site switchyard which is connect to the steelworks power supply. The switchyard was upgraded with a new transformer and all of the surface distribution boards have been replaced during construction;
2. The No 1 Shaft has its own switchyard and transformer connected to an Intergral Energy 33KV supply. The main fan runs on an 850KW 690V motor. 690V is not a common voltage and therefore the mine has purchased a spare motor and spare transformer for the site against the event of in service failure; and
3. The underground operations are supplied via Kemira Valley. The new switchyard is supplied from the Integral Energy 33KV line into two off 1500Mva transformers.

The panel transformers are 2.Mva units supplying 1000V outlets to the mining equipment. The longwall will be 3.3KV. The 2.5Mva transformers were chosen to allow longer runs on the low tension side with modelling indicating up to 1400m without a transformer advancement.

The power supply for the mine has been sized for the extraction of Area 3 (approximately 2027 at forecast extraction rates of 5 Mtpa).

Mining Equipment

The first three years of development at Dendrobium will be in strata conditions which require reasonably intensive primary support of between 6 and 8 by 2.4m fully encapsulated roof bolts as well as 1.2 to 1.8m rib bolts every metre of advance. For this reason the Project team decided to overhaul existing ABM20 machines available within Illawarra Coal rather than purchase new equipment. The available data indicates that from a roof strength and stress orientation, Area 3 will be more amenable to a 4 bolt per metre pattern and therefore a machine better suited to high speed cutting than bolting. The ABMs have been upgraded to ABM20 Mark IIIs with ABM25 conveyor and loader blade. The rib protection is essential in the Wongawilli Seam operations.

Walter Construction Group (Walter), the principle contractor at the mine, has two 12CM30 PPM machines in operation at the mine. The machine allows for changeout of the loader blade and platform area to get the machine closer to the face in poorer conditions with the trade-off of slower load out rates.

The mine has also gone with second hand shuttle cars, PJBs, Eimco 915s and EJ130, MPV, trailers, DCBs, auxiliary fans, monorail, pumps, compressors, and the TG conveyor system (which will only carry out the first tailgate development and will not be used for longwall extraction). All of the equipment has been overhauled prior to delivery to the site and is performing well.

Four new FBL loaders are currently on order as there are insufficient second hand units available.

Dendrobium Pit Top

Nebo Colliery commenced operations in 1946. The majority of buildings on the mine site date to 1948. At the commencement of the Dendrobium Project the buildings were just over 50 years old and therefore came under the NSW heritage listing. This precluded changes to the buildings unless sanctioned by the relevant government authorities and consequently presented some challenges in upgrading the 1948 facilities which serviced a pillar extraction mine with up to 500 employees to a modern pit top servicing a longwall mine with 180 employees and more service providers.

Some of the key issues were:

1. Location of an early 19th century kerosene oil shale works on the site which had to be partially excavated, investigated, mapped and covered over again for future excavation;
2. Inability to change the façade of the main bath-house office building. This led to compromises such as raising the floor in the control room to be able to see out of the windows rather than lower the window;
3. Surface services were reasonably well documented but very out of date and in need of upgrade - power supply and distribution, pipework;
4. Lack of pit top room for gear set down areas and car parking facilities;
5. The bathhouse/main office building is not located adjacent to the car parking area but rather at the back of the site which makes demarcation of personnel/visitors and heavy equipment very difficult.

In general the pit top works have gone very well although over budget. The pit top is now a functional, if tight, service area for the mine. Delivery of the longwall later this year will however stretch its capacity.

CONCLUSION

Since the Concept and Prefeasibility studies identified the establishment of a new mine in Mt Kembla as the best alternative to guarantee Illawarra's ability to supply a premium coking coal blend to the Port Kembla Steelworks and the world steel making industry, the Dendrobium Mine has set out to be a benchmark operation in all areas. As previously stated the environmental and community works, which have not been discussed in this paper, have set standards which are now being applied more broadly both within the Company and to other State significant projects. The mine, as the first new longwall mine in the district in more than 20 years, has been given the opportunity to put mining and engineering systems and practices in place from the outset which will seek to establish the mine at the forefront of the industry.

Dendrobium, however, is still a coal mine operating in a sometimes harsh underground environment. The management systems and planning prepared over the last several years is now being implemented. Some early successes such as the ventilation shaft, pit top upgrade, coal handling facilities and power supply will need to be repeated in benchmark development rates, longwall production rates and zero harm to the workforce if Dendrobium is really to transfer its paper aspirations into safety and production realities.

A STRUCTURAL SYNTHESIS OF THE SYDNEY BASIN – WORKING TOWARDS IMPROVING GEOLOGICAL CONFIDENCE AND PRODUCTIVITY

**Chris Woodfull¹, Stuart Munroe², Sally Griffin³,
Amanda Buckingham⁴, Andrew Ham⁵, Michael Etheridge⁶, Patrick Hanna⁷**

ABSTRACT: In a globally competitive environment, coal mining operations are continually focusing on improving longwall mine production and productivity levels, and at the same time enhancing safety practices. Commonly operators look to reduce costs and increase development rates by looking to improve equipment performance, change work practices and reduce unplanned downtime. This can be achieved in part by complementary assessments into potentially disruptive geological and geotechnical issues such as faults, intrusives and poor strata conditions, using advanced geological exploration and imaging techniques, in conjunction with their existing site-based geological knowledge. However, the location of, and controls on, local structural risk issues are not always well understood. Geometric patterns of small and local scale structures typically are a reflection of structures that are developed at regional scales. Developing an understanding of regional structure can enhance the insight into local mining risks. Regional structures are important in controlling basin initiation, depositional centres, fold and fault development, fault reactivation, and the loci for volcanic and intrusive activity.

INTRODUCTION

A structurally-based basin-scale study was undertaken for the Sydney Basin and the eastern part of the Gunnedah Basin and was principally designed to provide a GIS-based regional structural framework as the basis for more detailed assessments of structural risk issues for coal resources. This study was undertaken by a team of consultants with the support of a number of coal mining companies and the NSW Dept. of Mineral Resources (DMR). The structural framework was developed by integrating a wide variety of geological and geophysical information, gathered from not only the public domain but also using confidential and high resolution data offered by supporting companies. Use of the GIS database facilitated the interrogation of a range of geological parameters such as intrusive, fault trend relationships and/or regional to local fault patterns and regional stress data.

A structural model that includes 4-D interpretation of the Basins as well as a series of maps which highlight the known and interpreted regional basement geology and structural features has been developed. Using this information, regional-scale structural corridors and domains can be identified and ranked according to a range of regional risk parameters, such as the relative nature and/or abundance of basement structural features, basement geology, depth to basement, and the distribution of intra basin intrusives. The interpretation forms a base onto which more detailed information can be added and analysed in a regional context.

Importantly, this study provides a data platform as well as offering participating companies the ability to interrogate a more integrated and comprehensive dataset of the entire Sydney Basin for their own long term planning and risk analysis. By improving recognition of structural patterns in the Basin both regionally and locally, an understanding of the relationship between basement features (structure, composition, depth) and known structural risk can help reduce geological uncertainty and therefore be more effective in managing

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structural risk. The Springvale and Angus Place operators are a case in point, where an improved regional understanding of influences relating to local/operational-scale anomalies was the first stage in a structural review and risk analysis study that has assisted in improved operational efficiency for these longwall mines (Knight and Teasdale 2001).

BACKGROUND

Coal mining operations are continually focusing on improving productivity levels in a range of areas. One of the critical areas is the geology. Geological issues include coal quality, changes in seam thickness, seam splitting and identifying or resolving potentially disruptive geological and geotechnical issues such as faults and intrusives. From a company's competitive perspective, a reduction in geological uncertainty should lead to improved mining decisions, and commensurate productivity and cost-related benefits over time. Industry-supported integrated regional-scale studies which aim to deliver practical outcomes can also help provide positive impacts on the overall competitiveness of the region (Woodfull, Munroe and Hanna, 2003).

To help resolve the potentially disruptive geological and geotechnical issues such as faults, intrusives and poor strata conditions, coal mining operations use forward structural interpretation (i.e., ahead of mine planning and development). Forward structural interpretation studies and the structural framework and models that develop from this work, can be derived from a range of geological data sources including geological mapping (surface and seam), drill hole information and the interpretation of remotely sensed data (SRK, 2004, Woodfull, Munroe and Hanna, 2003). The location of, and controls on, local structural risk issues are not always well understood. Geometric patterns of small and local scale structures typically are a reflection of structures that are developed at regional scales.

More detailed interpretations and structural models developed from these studies typically result from the integration of a combination of geological field data (drill hole, mapping) and potential field data such as airborne and seismic surveys. While potential field data may be used as assessment tools for near surface to at-the-seam mining risk issues, these data sets (mainly magnetics and gravity) can also provide a window to the basement. Once calibrated to geology, this data provides information that allows the development of a predictive structural model based on basement composition and structure.

The basement of any basin provides the foundation onto which the sediments are deposited. The inherent composition and fabrics within the basement play a major role in the manner in which the crust deforms during major periods of extension or compression. Basement structures are important in controlling basin initiation, depositional centres, fold and fault development, fault reactivation, and the loci for volcanic and intrusive activity.

By improving our understanding of the relationship between regional-scale basement features (structure, composition, depth) and known local structural risk (such as recognised structural hazards at the mine), we can help to reduce some areas of geological uncertainty and, therefore, be more efficient and effective in managing risk at a range of scales.

OBJECTIVES

The two main objectives of the study were to:

- provide GIS-based, integrated regional-scale geological data that is relevant to structural geology studies, principally in the area of structural risk management, whereby future short to longer term sub-regional to mine-scale studies can build on the results of this synthesis work, and
- improve the understanding of the basement geology and regional fault kinematics during basin development, through the interpretation, at regional scales (1:500 000 to 1:100 000 where practical), of the data sets and the development of a 4-dimensional structural geological model for the Basin area.

Additionally, the study aims to assist the change from an empirical to a more analytical approach to structural risk management. Understanding why potentially disruptive or changing geological conditions occur, can be a powerful tool in improving predictive studies and helping to manage risk areas / issues, (SRK, 2004). The study results may also have broader application within one organization as the regional structural model and/or

integrated data sets should have practical benefits to broader natural resource and risk-based studies including gas exploration, mitigation and/or sequestration initiatives, groundwater and land use planning / environmental studies.

APPROACH

The methodology used to develop a comprehensive structural model relies on the integration of all appropriate geophysical and geological information. Individual datasets alone can be ambiguous and when interpreted in isolation often produce poorly constrained results. Through integration, the model can be better constrained. Integration provides the means with which to calibrate each dataset to the other. Figure 1 presents an overview of the methodology. A more detailed discussion of the approach can be found in two related publications (SRK, 2004 and Woodfull, Munroe and Hanna, 2003).

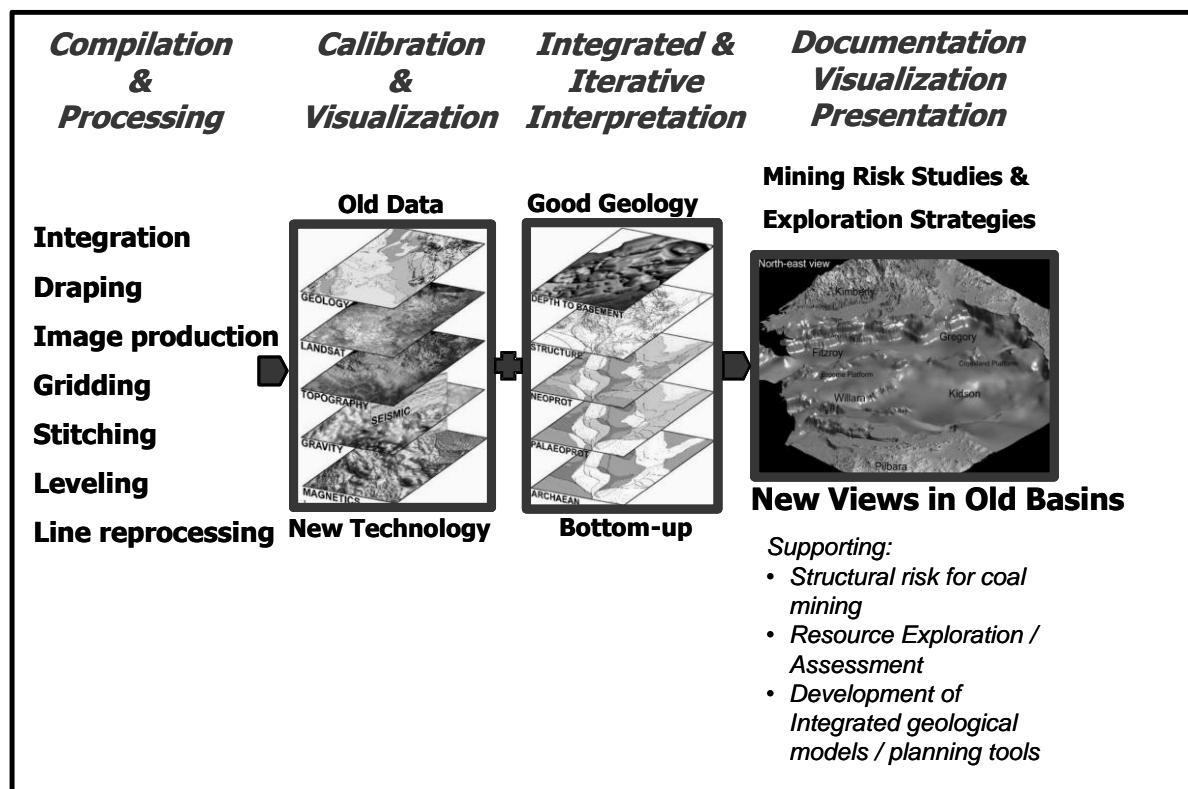


Fig 1 - Simplified pictorial flow chart showing the process followed in the development of the regional structural model for the Sydney Basin

Data Sets and Regional Structural / Basin Model

More than a dozen geological and potential field data sets were compiled, processed and/or developed for this study. Table 1 aims to provide an overview of the key data sets and the series interpretive maps available for use as a result of this study.

As an example of how the various data sets were used, Figure 2 presents a simplified flow chart of the main data sets used in the development of the basement geology and structure interpretive maps, the modeled cross sections. Key interpretive information (basement geology, structural layers, regional geology) in conjunction with modeling results (e.g., cross sections, magnetic and gravity anomaly depth estimates) and critical data from selected data sets (e.g., drill hole, seismic data) were then used to develop the 4-D regional structural / basin model shown at the bottom of Figure 2.

Benefits / Uses and Potential

One of the primary benefits of the study, and presently recognised by supporting organisations as a core strength of the project, is the collation and integration of a range of disparate data sets relevant to geology and

geotechnical risk studies (SRK, 2004). With the development of the GIS-based datasets, organisations and individuals can now more readily:

- place mining and exploration leases, or risk issues in a more regional context, and
- undertake iterative data analysis and assess their areas for data deficiencies / resolution issues as part of exploration and /or risk management planning.

Another important benefit of the study is the development of a 4-D structural / basin model. The development of this regional model does provide a:

- framework for ongoing detailed structural risk studies, or for re-evaluating risk issues, at a range of scales, and
- structural framework for coal companies to incorporate into their existing coal geology / resource models or to build district-scale coal resource models.

It should be noted that while the collation and integration work has been quite extensive, it does not necessarily represent all the geological or potential field data that could be included in the data sets, at this point in time. It is envisaged that the data sets and the 4-D structural / basin model will evolve over time. Furthermore, the structural model should benefit from improved calibration, via the inclusion of new data, such as higher resolution remotely sensed surveys and factual mine site data or feedback as the model begins to be used on-site.

However, while needing to keep in mind impacts on the coal seam, the study is regionally focused, therefore:

- The results of this study do not aim to be interpretive or predictive at the coal seam level.
- The project does aim to be predictive at a regional / district scale, particularly where there is a higher level of data quality and resolution.

CONCLUSION

A 'Sydney Basin' regional structural synthesis study and basin model has been undertaken for a number of coal exploration and mining entities with interests in the Sydney Basin and the eastern part of the Gunnedah Basin. The study contains:

- a compilation of relevant data, processing (where appropriate) and integration of correlating data sets for a structural geological study,
- the provision of the data sets in a GIS (or equivalent) format,
- a number of interpretive data sets that have been generated from the initial data compilation,
- the development of a 4-D regional structural / basin model, using the compiled data set, and
- synthesis of the results of the study to understand the spatial distribution of observed geological features.

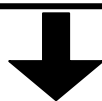
The starting point for the Basin study is the 'container', or basement, into which the Basin has been deposited, by the reactivation and activation of geologically controlled features. Pre-Basin geology and structure has had an important control on the geological development of the Basin. Syn-Basin and Post-Basin tectonics have also reactivated basement structures as well as activating intra-Basin features (SRK, 2004).

The data sets and accompanying regional structural / basin model developed for this study, provide coal exploration and mining companies with a more integrated geological framework for ongoing sub-regional to mine-scale geological risk based studies. The results of study can be immediately used to:

- More readily start to place their mining and exploration leases, or local risk issues in a more regional geological context,
- Broaden and quicken the process of iterative data analysis,
- Use the data sets as an aid during future exploration planning, such as for sub regional to local scale data acquisition programs , and / or
- Use the model as a platform for (or synthesis the structural components into existing) more detailed coal geology / resource models.

Table 1 - Summary tables briefly highlighting and describing A) the key data sets compiled and developed for the study, B) the Time and Space Event History Chart and C) the interpretive maps developed for the study, based on the key data sets and Event History Chart.

Datasets	
Airborne Magnetic	Provide information on: <ul style="list-style-type: none"> • Structure and composition of the magnetic basement • Exposed magnetic bodies such as igneous intrusions Compilation of over 40 surveys from the public domain and higher resolution private company surveys Area-based improvement in data coverage for the following coalfield areas: Hunter/Newcastle, 26%; Southern, 20%; Western, 10%; Gunnedah, 3%
Radiometric	Sourced from recent public domain and a number of smaller private company surveys Provide information on regional structure at the surface
Gravity	Useful for basin architecture, structure and geology
Digital Elevation Model (DEM)	Surface expression of geology and structure The main sources were the NSW DMR 250m DTM grid (onshore) and the GA 2003 bathymetry and topography grid (offshore)
Surface Geology	Provide calibration for interpretation of DEM, gravity and magnetic data Provided by the NSW DMR as a 250K scale digital data set In addition, published 100K scale coalfield geology maps are included as scanned images. Digital data of structure and igneous bodies have been captured from the 100K scanned maps
Satellite - Landsat 7 and ASTER	Surface reflectance data is used for identifying surface geology and structure - reflected in outcrop, vegetation patterns or soil types Two satellite imagery data sets were compiled: <ul style="list-style-type: none"> • 30 m pixel size - Landsat 7 mosaic of bands 7 (Red), 4 (Green) and 2 (Blue) derived from the NASA Earth Science Enterprise coverage - MrSID (circa 1990) • 15 m pixel size - ASTER mosaic (with near complete coverage for the Basins), for improved resolution over the
Published Seismic Data	Provide regional constraints on the structural geometry of basement blocks and basins, and movement histories on major structures Selected published line profiles have been used to help constrain basement depth estimates for the regional structural
Stratigraphic Drill Holes & Petroleum	Calibration of basement depth and basement lithology A key drill hole data set was developed to help constrain basement geology and depth estimates for the structural
Stress	Compilation database incorporating data from the Australasian Stress Map Database and other independent stress measurements
Mine Site Intrusion & Structure Data Base	Compilation map of intrusion and structure data mapped at the seam, from a number of companies, including sponsors
Other Data	<ul style="list-style-type: none"> • Intrusion geochronology from a number of published sources • Earthquake epicentres from USGS global data base • Surface heat flow from Global Heat Flow data base (Pollack et al 1991) • Southern Coalfields structural compilation (ACIRL, 1989) • Current mine and lease boundaries • Cultural layers (state boundaries, towns, roads, national parks)



Outcomes & Applications	
4-D regional Structural Model and Regional Basin model	Identifying regional geological risks and opportunities for future operations
Basement Geology and Structure	Implications for structure and intrusion behaviour at the seam level
Basin Architecture (SEEBASE™)	3-D view of the base of the Permian and Triassic sequences which incorporate all information on geology, tectonics, palaeogeography, intrusions and structural risk
Structural Interpretation	Structural zones interpreted to have been active during key tectonic events Provides some indication of overprinting relationships, likely growth faults and later through-going structures
Regional Surface Lineament interpretation	Interpreted structures / lineaments based on Landsat and DEM data sets; interpretive focus on coalfield areas
Regional cross sections	Iterative modeling of basement geology using gravity and magnetic profiles and relevant SEEBASE™ data
Time and Space Event History Chart	Compilation of key lithology, structure and deformation, tectonic and magmatic data for the Sydney and Gunnedah Basin areas over time, that highlights key kinematic events that have controlled the geological development of the areas, including the Basin geology and structure
Bibliography	A summary of useful technical publications / documents used in this study Has a bias toward structural geological, tectonic / kinematic, stress and / or intrusive-related papers

SEEBASE™: Structurally Enhanced View of Economic Basement.

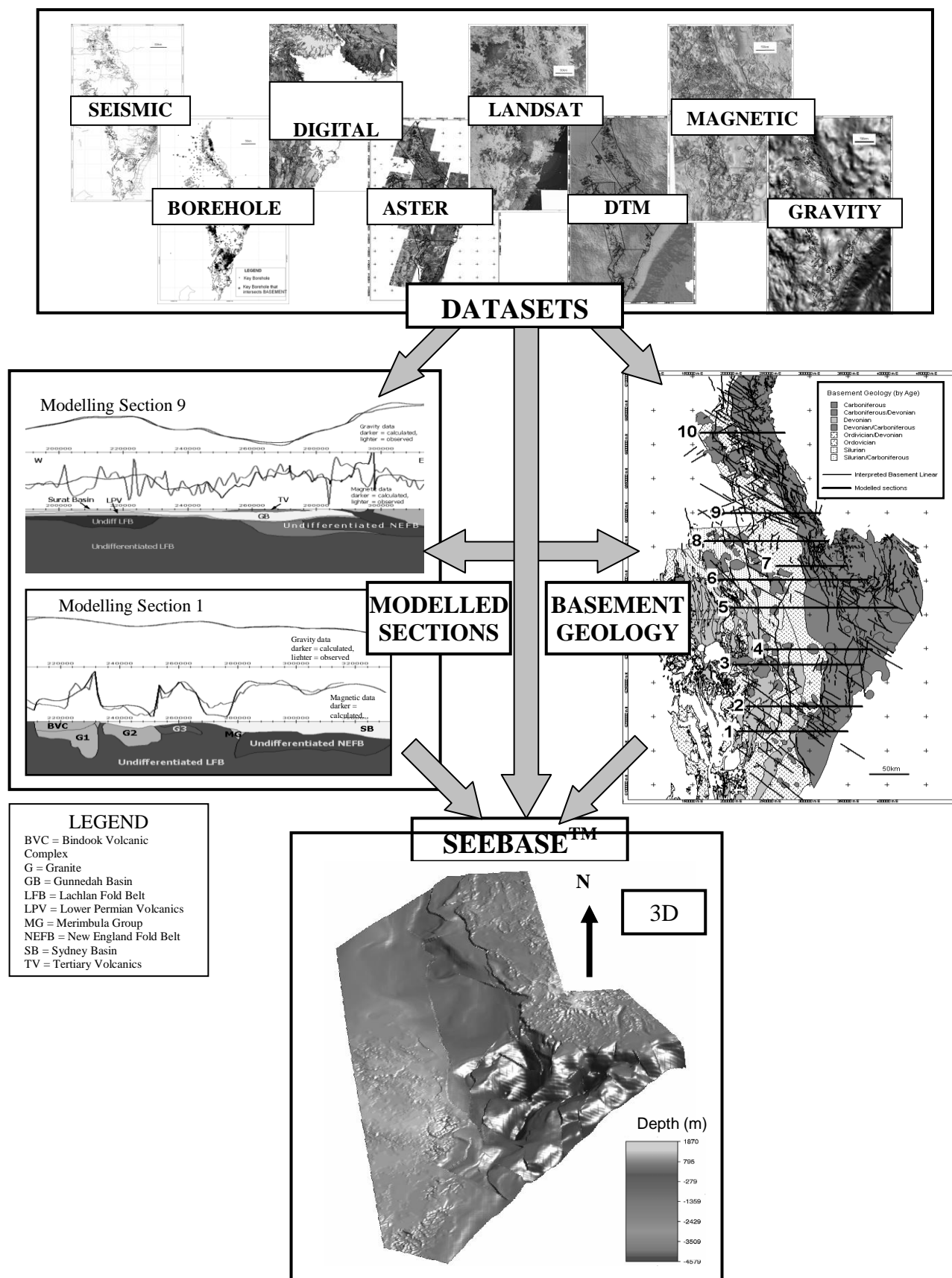


Fig 2 - Simplified flow chart showing the type of data sets that were used in the development A) the basement geology interpretive map and modeled cross sections, which formed a basis for B) the 4-D regional structural model

Based on the results of this regional study, more detailed district-scale interpretive studies, (complemented with supplementary data set contributions and / or acquisition), can also be expected to provide a more comprehensive local data / model package for improving geological certainty and thereby a more effective performance in predicting and managing geological risk.

It is planned that at least one update to the data sets will occur over the next 12 months, prior to a general release of the study results, with the aim of including any new public domain data that becomes available and / or additional private sector data contributions, that will improve the quality of the data sets or allow improvement in the calibration of the regional structural / basin model.

ACKNOWLEDGEMENTS

The authors wish to thank the management of Xstrata Coal, Anglo Coal, Excel Mining and Illawarra Coal for providing financial support and data to this project. The authors also wish to acknowledge the support of the NSW DMR including data contributions and advice, and the interest and mine-site data contributions from Muswellbrook Coal, AMCI, Southland Coal, Bloomfields Mine and Rix's Creek Mine. Additional, important data contributions include Geoscience Australia, Dr John Shepherd (Shepherd Mining Geotechnics) and Reynolds, S.D. and Hillis, R.R. 2003, The Australasian Stress Map Database (www.ncpgg.adelaide.edu.au/asm).

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PLANNING FOR A HEALTHY FUTURE

Bruce Ham ¹

ABSTRACT: The move from prescriptive to 'duty of care' style legislation will move the coal industry towards the world of litigation and soaring insurance premiums. Evidence from Queensland suggests that coal miners' injuries are a \$4million Workers Compensation cost but the personal cost to miners, their families and community is well over \$43 million per year. Under 'duty of care' there is a high risk that much of this cost plus legal expenses could be transferred back to the employers through litigation and increasing insurance premiums. The sustainability of the industry may come under threat.

While 'fitness for duty' assessments are a key part of all well managed health surveillance programs, 'duty of care' style legislation dictates that employers must implement safety management systems for hazardous exposures. This may include long-term assessment and management of the risks of hazardous exposures in the occupational environment. An analysis is undertaken to identify issues of trigger levels for health interventions. Parameters such as health condition, change in health status and cumulative occupational exposure may be used as triggers that initiate health interventions prior to unacceptable or compensatable harm being suffered. It is argued this and nothing less will effectively meet the employer's obligation under legislation.

AIMS AND OBJECTIVES

While 'coal mine planning' is largely a geotechnical, engineering and economic evaluation activity, some consideration needs to be given to the human factors in production. One of the most important human factors and legislative issues is to have a workforce that is competent and confident in working without risk to their safety or health.

The objectives to be achieved in managing the risks to ensure a healthy workforce are:

1. Ensuring all the potential health hazards, work requirements and relevant medical conditions are identified.
2. Establishing a system to compile data on occupational health and exposures in the work place by implementing a health surveillance program that includes 'fitness for duty' assessment and on-going monitoring and management
3. Examining the evidence as to the likelihood, consequence and time frame of the impact of occupational health hazards,
4. Examining the controls or interventions that may be used in managing the risks,
5. Identifying trigger levels that initiate an occupational health intervention
6. Implementing occupational health interventions.
7. Ensuring an adequate system of documentation is established, and
8. Developing a review process.

DATA COLLECTION

There are several dimensions to the collection of data for the identification and management of occupational health risks. These include:

- Identify health hazards and potential controls,
- Collect and compile health surveillance data,
- Monitor exposures,

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- Examine injury data,
- Examine health outcome data and
- Examine mortality data

Through an understanding of the operation and occupational health practices, a process can be established to ensure all the potential health hazards are identified (Grantham 1992 and LaDou 1997). While injury data is a useful starting point, it is limited to identifying the effects of hazards with short term impacts. The diagnosis of injury also has some failings such as the strain and sprain injuries which incorporate a proportion of long term overuse and cumulative damage of soft tissue injuries.

Coal Services Limited and the Queensland Department of Natural Resources and Mines undertake comprehensive health assessment programs (Bofinger and Ham 2001, Coal Services 2004 ¹, Ham 2004 ². The number of entry and periodic health assessments is shown in Table 1.

Table 1 – Health Data Collection in Queensland and New South Wales

State / District	New South Wales				Queensland		
	North/North West		South & West				
• Medical Assessments	1999/00	2000/01	1999/00	2000/01	2000	2001	2002
Entrants	1010	1533	255	428	2221	5411	5512
Employees	1841	1373	726	556	2306	1419	1672

The number of employees in each Queensland and New South Wales has been about 10,000 persons. The greater number of assessments done in Queensland since 2001 is a reflection of the need to incorporate short-term contractors in the health scheme under the new 2001 legislation.

A program to gain reliable exposure estimates was reported by McFadden and Davies (2003). The study examined the issue of the amount of data to be collected to obtain a reliable estimate of exposure applying criteria developed by Grantham (2001). The study concluded that most face workers were exposed to high levels of inhalable dust and almost all underground mine workers were exposed to noise levels well beyond the statutory limits. The study identified a range of occupational health issues that warranted improved controls in a 12 month time frame. These included hazardous substances, microbial agents, organic vapours and various gaseous contaminants.

In the absence of a program of systematic exit medicals, some useful data has been obtained by examining the premature superannuation claims data from the Queensland Coal and Oil Shale (QCOS) Superannuation Fund in Queensland (Ham 2003). Claims are made for early payouts due to death or Total Permanent Disability (TPD). The numbers and proportions of persons who leave the industry with cancers, heart disease and traumatic injuries are shown in Table 2.

The work of Bofinger and Ham (2002 a), used the register of coal miners in New South Wales and Queensland to cross match and extract death data held by the Australian Institute of Health and Welfare. While this study focussed on the risk of heart disease in coal miners, it opened an opportunity to consider the coal miners' mortality data more widely

¹ Coal Services Limited 2004. Web Site – www.coalservices.com.au/health

² Ham B. W., 2004. Health Surveillance Web Site – www.members.optusnet.com.au

Table 2 – QCOS death and total permanent disability data

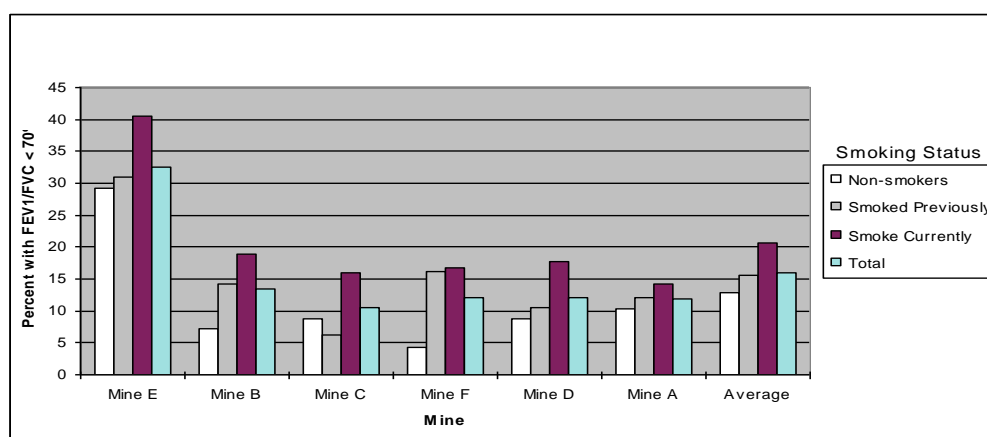
Cause	Deaths	TPD	Totals	Av. Age
Cancer	14	20	34	51
Circulatory disease	12	21	33	53
Ear disorders	0	3	3	
Endocrine disorders	0	3	3	
Infectious diseases	0	5	5	
Musculo-skeletal disorders	0	83	83	47
Nervous / mental disorders	9	43	52	48
Respiratory disease	0	4	4	
External causes	13	32	45	41
Other	3	2	5	
Total	51	216	267	48

ANALYSIS OF DATA

The Industry Commission report on Work, Health and Safety' (1995), accepted evidence being compiled for the Worksafe 'Report on Best Estimate of the Magnitude of Health Effects of Occupational Exposure to Hazardous Substances' (Kerr et. al., 1996) that identified widespread under-reporting of occupational related death through illness. The Commission concluded that systems needed to be put in place to monitor long-term exposures and provide a mechanism for collating long term health outcomes for persons working in environments of elevated risk. The analysis of the data collections discussed previously is a first step in addressing this need

An analysis of cross-sectional Queensland coal workers health data by Ham (2000) demonstrated the procedures for extracting and analyzing health data held in the Health Database maintained by the Health Surveillance Unit of Safety and Health Division (Mines) of the Department of Natural Resources and Mines. The study included stratifying the health data by mine, age group and gender. The health data was also cross matched with injury records held by the Department of Natural Resources and Mines.

This study also demonstrated how the health database data can be extracted and statistically analysed while maintaining confidentiality of personal medical records. Figure 1 shows the incidence of reduced respiratory function in 45 to 55 year old workers in underground Queensland coal mines. As electronic recording of medical data commenced in 1993, the longitudinal data should be available from 2003.

**Fig 1 - Respiratory function loss in underground coal miners (after Ham 2000)**

In order to give some perspective on the importance of health issues, the QCOS Superannuation early claim data was analysed by Ham (2003) so as to account for the cost of lost productive capacity. The results comparing published injury and lost time data are shown in Table 3.

Table 3 – Estimate of costs of poor health and injury from QCOS data

Data Source	Days lost	Man-Years lost	Total Cost \$M
Reported Lost Time Injury Data	3627	18	1
Injury Lost Time from Production Returns	5475	27	2
Sickness Lost Time from Production Returns	31158	156	12
Workers Compensation Report			4
Lost Wage Estimate from QCOS data		660	43

In relation to the health outcomes, it is important to distinguish between fatal and non-fatal outcomes. For example, hearing loss is non-fatal, but in severe cases, it represents a diminished quality of life. One dimension of hearing loss is damage versus impairment. Practical tests may be applied to demonstrate that a worker with hearing loss is suitable for working in a specific function. Generally, the first indicator of noise induced hearing loss occurs at the frequency of 4000 hertz where a loss of 40 decibels is considered significant. Impairment (40 decibels or more) in the voice ranges of 500 hertz to 2000 hertz may limit the ability of a person to safely function at work and in daily life. This might be considered an unacceptable outcome.

The second dimension is what is the level of hearing loss at which a health intervention should be undertaken. Interventions may vary from education and insistence on the wearing of hearing protection to removal from high noise environment. A worker may clearly be fit to undertake the work required, but continued exposure will result in an unacceptable level of harm – (and how is this level defined). If the retirement age is taken as 60 years and damage is proportional to cumulative exposure, then a trigger level for high level interventions may be set differently for young verses old workers. The case for different risk profiles may have to be argued in an anti-discrimination context.

A second non-fatal outcome is the lower back damage due to cumulative exposure to whole body vibration. McPhee, Foster and Long (2001) identified several types of mobile plant where international whole body vibration standards are exceeded in less than 8 hours of exposure. As a community, our habits and life style choices are often not conducive to the long term health of our backs. A proportion of the community is prone to degenerative back disorders. Our current screening tools are not very effective in providing early warning of the onset of this condition. This provides an opportunity for future research.

With whole body vibration and several other hazards, some significant methodological issues need to be addressed just in relation to the reporting of health outcomes and measuring cumulative exposure. This must be done before the social debate on what represents an unacceptable risk of an adverse health outcome and what are reasonable intervention strategies. The debate will have to resolve commercial, anti-discrimination and personal rights conflicts.

In terms of long term fatal outcomes, the challenge in analysis of health outcome data is to develop a method to compare it with community and other sector data-sets. The general community data format reported by the Australian Institute of Health and Welfare examines health disorder rates per 100,000 head of population in various age groups. Other approaches are to examine the proportions of different causes of death between groups and to compare the median age at death for particular causes

The work of Bofinger and Ham (2002b), used the register of coal miners in New South Wales and Queensland to cross match and extract death data held by the Australian Institute of Health and Welfare. The data from the heart disease study examined the proportions of heart disease to other causes of death to draw the conclusion that heart disease was no more a risk to coal miners than the general population. Death rates – deaths per 100,000 population per year were also calculated to confirm this conclusion. There was some concern that the calculated death rate was an underestimate as it was based on a birth group (cohort) without a correction to provide death rates based in the surviving population

Subsequent reports on the general population by the Australian Institute of Health and Welfare (AIHW, 2002), revealed that different causes of death have different median ages at death. For example, death by traumatic injury is a characteristic of a younger group, while cancer and respiratory disorders are characteristic of an older age group.

The age profiles on the register of Queensland miners, the New South Wales coal miners are substantially different from the general population as shown in Figure 2. As the New South Wales miners' register is contains a profile that is older than the general population, a high degree of validity can be attributed to the mortality data in terms of death rates and median age at death for various causes as shown in Table 4.

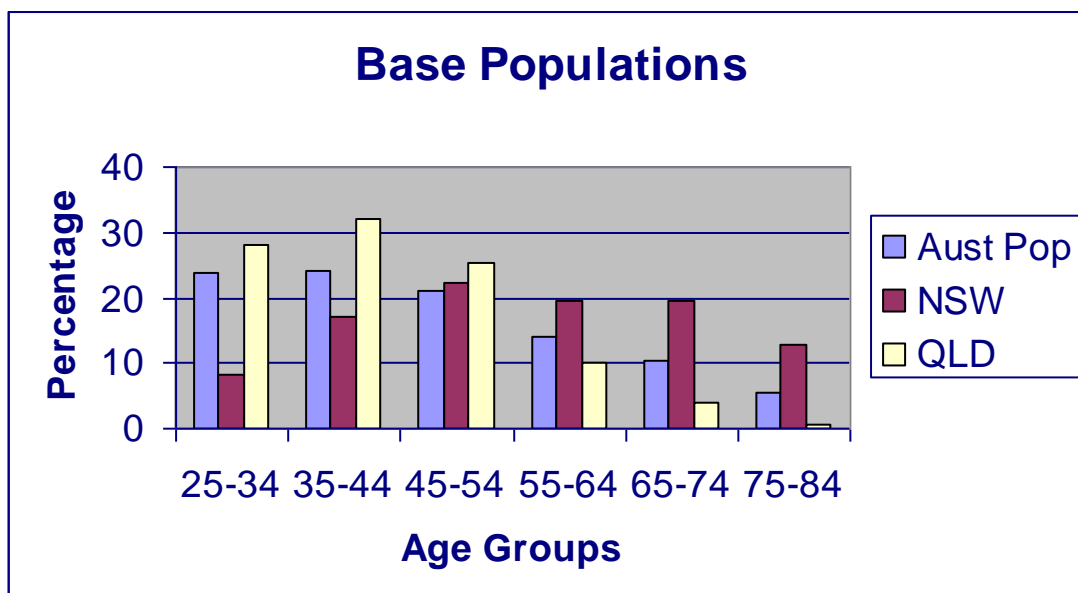


Fig 2 – Analysis of coal miners register data compared to general Australian population

The median age at death for various diseases indicates that the retired New South Wales miners generally have similar life expectancies to the general population. The same cannot be said for Queensland miners for whom median age at death is much lower than the general population. The comparison of median age at death is not valid in this case as older workers are under represented in the data set. In the case of Queensland, analysis of age specific death will be required to provide a reliable comparison. Changing technologies (eg. longwall mining) and work arrangements (eg 12 hour shifts) cause the use of historical data to be questioned. A better approach is to examine trends in age specific death rates particularly where death data can be linked to exposure data. This provides an opportunity for future research.

If the strategy applied by the UK coal miners dust disease case (Rudd, 1998 and Coggan and Taylor, 1998) is applied, then an unacceptable level of harm is defined when the occupational disease (or death) rate for a particular occupational group significantly exceeds that of the general population for the equivalent age group. The underlying dose-response studies for cases such as this, require long term and reliable estimates of exposures and health parameters and outcomes. In general terms, this is captured in the *Queensland Coal Mining Safety and Health Regulation 2001*, Section 53 (*Qld Gov. 2001*), but there is need for some guidance as to appropriate standards in data collection.

Extended shift work is known to be associated with difficulties in making ideal lifestyle choices, and can be allied with poor diet, high blood pressure and the risk of fatigue related injury (particularly journey injuries). Inter-mine comparisons and benchmarking are needed to clearly identify risk levels and promote best practice risk management in this area.

LEGISLATIVE FRAMEWORK

The collection, analysis and interpretation of health data and performance measures, takes on a whole new meaning under the new style duty of care legislation as compared to the previous prescriptive legislation. This is reinforced in the Queensland Coal Mining Safety and Health Regulation 2001(Qld Gov. 2001), in which Section 49 requires employers to have a safety management system to periodically monitor the level of risk from workers exposures to hazards. This is contained under the health scheme regulations which provide for medical practitioners to maintain confidential medical records. Through section 49 (safety management system for occupational exposures) the medical practitioners should provide advice for the identification and management of 'at risk' employees – not just advice on employees who have a medical condition which restricts their work activities.

Table 4 – Analysis of Mortality Data

ICD Code Number	Cause of Death Category	Number of deaths		% of deaths			Median Age at Death		
		NSW Miners	Qld Miners	NSW Miners	Qld Miners	Australian Population	NSW Miners	Qld Miners	Australian Population (1)
II	Neoplasms (Cancer)	821	113	34	39	31.1	70	61	71
IV	Endocrine, nutritional & metabolic diseases	70	7	3	2	2.5	72	64	74
V and IV	Mental Disorders and diseases of the nervous system	64	6	3	2	2.9	71	na	42
IX	Diseases of the circulatory system	940	75	39	26	35.5	74	59	76
X	Diseases of the respiratory system	232	12	10	4	6.4	73	72	77
XI	Diseases of the digestive system	69	4	3	1	na	68	na	na
XIX and XX	Injury etc – external causes	149	66	6	23	2.1	48	33	39
	All Others	83	10	3	3	19.5	75	67	na
Total	All classes	2428	293	(1) AIHW, 2004					

Driven by the findings of the 1994 Moura Disaster, the need and application of safety management systems is both entrenched in legislation and well understood in the control of major high energy hazards such as methane and spontaneous combustion. The application of these principles to long term, chronic and possibly fatal conditions is more complex because of the extended time frames and the difficulty of collating and analysing long-term exposure data and confidential medical data.

This problem is made more complex by the variability of human reactions to various hazardous exposures, the need to protect sensitive individuals and an array of legislative, industrial relations and social issues. Among the social issues, are the awareness of mine workers and employers to the medium and long term consequences of deteriorating health as a result of hazardous occupational exposures.

The first step toward developing a culture of health in the mining industry is to provide adequate training in the understanding, collection, analysis and implications of health and injury data. It is only very recently that the National Mining Training Advisory Body has recognized that the mining industry needs to be trained to at least the same standard as general industry in terms of such disciplines as ergonomics, occupational health and occupational hygiene. Over several years this understanding of health issues will be integrated into the coal mining education processes through levels of the Australian Qualification Framework (AQF 2 to AQF 6).

The current programs for coal industry health assessments coordinated by Coal Services Limited in New South Wales and the Department of Natural Resources and Mines in Queensland are well placed in terms of international best practice. Changes in legislation in New South Wales and Queensland, require hazards to be controlled through structured safety management systems. The well developed causation and damaging energy models used in the control of dynamic safety risks fail to meet the more subtle factors required for safety management systems for occupational hazardous exposures. Current practice is to identify most hazards, control them within the limits of current technology and provide personal protective equipment for high exposures. When an occupational disorder becomes severe in the workplace, a health assessment is undertaken to certify the employee is unable to work safely. The employee is terminated and may claim an insurance payment. It is difficult to reconcile this practice with the legislated obligations of care to be managed through a safety management system

At least in theory, the assessment of risk of an occupational exposure is based on a data set derived in a dose-response study. These studies form the basis for the development of exposure limits and related regulations. These standards ensure that an unacceptable level of harm occurs to an acceptably small proportion of the population.

A core concept of good occupational health and safety practice developed by The International Labour Organization (ILO) is that workers should be entitled to a career in the workplace without adverse long term effects to health. While there is often uncertainty in compiling long term occupational health statistics, programs of data collection and analysis currently need to be developed to enable employers to demonstrate to government that workers are not being adversely affected by occupational hazards.

CONCLUSIONS

There is a significant difference between trigger levels for fitness for duty criteria and early health interventions. Under 'Duty of Care', safety management systems should provide for trigger levels for an early health interventions that effectively manage the risks that a worker subject to occupational hazards might eventually suffer an unacceptable health outcome.

There is a long term need to undertake dose-response studies that identify either cumulative doses or critical changes in body function that signify that some health risk is moving into the unacceptable range. This raises two questions that can be complex in relation to some specific hazards. These are: 'how is the dose measured?' and 'what is a health outcome that is unacceptable. Unfortunately, comprehensive data sets and, in some cases, even measurement tools are not available in relation to cumulative doses for a number of common occupational hazards.

The industry should be concerned that by failing to establish that best practice exposure based health surveillance, it cannot substantiate the safety management systems for occupational exposures that have been implemented. The consequence is that former workers (or their families) with any one of a number of medical conditions that may be associated with occupational exposures have a substantial claim for compensation on the grounds that a safety management system as required by legislation which may have prevented the adverse health outcome, had not been adopted.

While the solution is not clear, the first steps are obvious. Health surveillance needs to include exit medicals and follow-up mortality studies. Exposure monitoring needs to be upgraded to ensure all workers at risk, have the exposure data maintained in a database that is transportable across the industry (as with their medical data). Historical and prospective dose response studies need to be undertaken. These and equivalent overseas studies need to be examined in the context of defining an acceptable level of risk of harm. While this may define levels for removal of exposed workers, it opens a new generation of questions in relation to the obligations and rights of both employees and employers.

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PLANNING FOR MINE CLOSURES

Hank Pinkster¹

ABSTRACT: Mine closure is not typically something associated with planning to develop a new mining operation, but it is at this stage as well as aayall other facets of it's life cycle that it must be considered.

BHPB Illawarra Coal operates 4 underground Coal Mines on the South Coast of NSW. In addition to these operational mines Illawarra Coal has responsibility for a number of disused mine entries that have to be rehabilitated. Some of the infrastructure associated with these operations dates back over 150 years and this in itself raises unique issues when it comes to rehabilitation.

Under the Mining Act 1992, environmental protection and rehabilitation are regulated by conditions included in all mining leases to ensure that all mining operations are safe, the resources are efficiently extracted, the environment is protected and rehabilitation achieves a stable, satisfactory outcome. The state and status of the numerous Illawarra Coal owned mine entries varies widely but approval from the Department of Mineral Resources (DMR) is required in all cases prior to their rehabilitation and subsequent removal from our asset base.

The future land use is a key driver in many cases as to what should occur with the respective sites and as to the relative timing for the rehabilitation to commence.

INTRODUCTION

BHPBilliton (BHPB) Illawarra Coal operates several collieries that due to the age of the operations, did not contemplate in detail rehabilitation until the operations were well advanced or in some cases close to closure. In planning for rehabilitation a protocol has been developed to achieve the desired rehabilitation outcomes for all the relevant stakeholders. This paper overviews the processes, interactions and timings to be considered when planning for mine closure. By way of example two current rehabilitation projects are used to demonstrate the issues, procedures and interactions necessary in order to develop and implement an acceptable rehabilitation plan.

Mine closure is a continuous series of activities that begins with pre planning prior to the project's design and construction and ends with the achievement of long term site stability and the establishment of a self sustaining ecosystem (WMI, 1994). Given this, an important objective would be that all organisations be encouraged to develop comprehensive mine closure plans that return the mine sites to where practicable viable, self sustaining eco systems and that these plans are thoroughly and appropriately communicated, financed, resourced, implemented and monitored.

Legislation for mine rehabilitation is under the guidance of the Department of Mineral Resources. Under the Mining Act 1992, environmental protection and rehabilitation are regulated by conditions included in all mining leases, including requirements for the submission of a Mining Operations Plan (MOP) prior to the commencement of operations, and subsequent Annual Environmental Management Reports (AEMR).

Collectively, the MOP and AEMR constitute the Mining, Rehabilitation and Environmental Management Process (MREMP), which has been developed by the Department of Mineral Resources. The MREMP aims to facilitate the development of mining in New South Wales and to ensure that all mining operations are safe, the resources are efficiently extracted, the environment is protected and rehabilitation achieves a stable, satisfactory outcome. Documentation on the subject of mine rehabilitation can be found from Government, industry and private sector groups. (ANZ Min Council 2000, DMR 2002, BHP Billiton 2002).

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The closure planning process is summarised diagrammatically in Figure 1. This diagram clearly identifies that the planning for mine closure starts with the concept or pre feasibility stage of a project, where the level of uncertainty is high. Closure planning spans the life cycle of the project and as such the level of uncertainty continually reduces as the closure becomes more of a reality.

The following diagram describes an overview of closure planning:

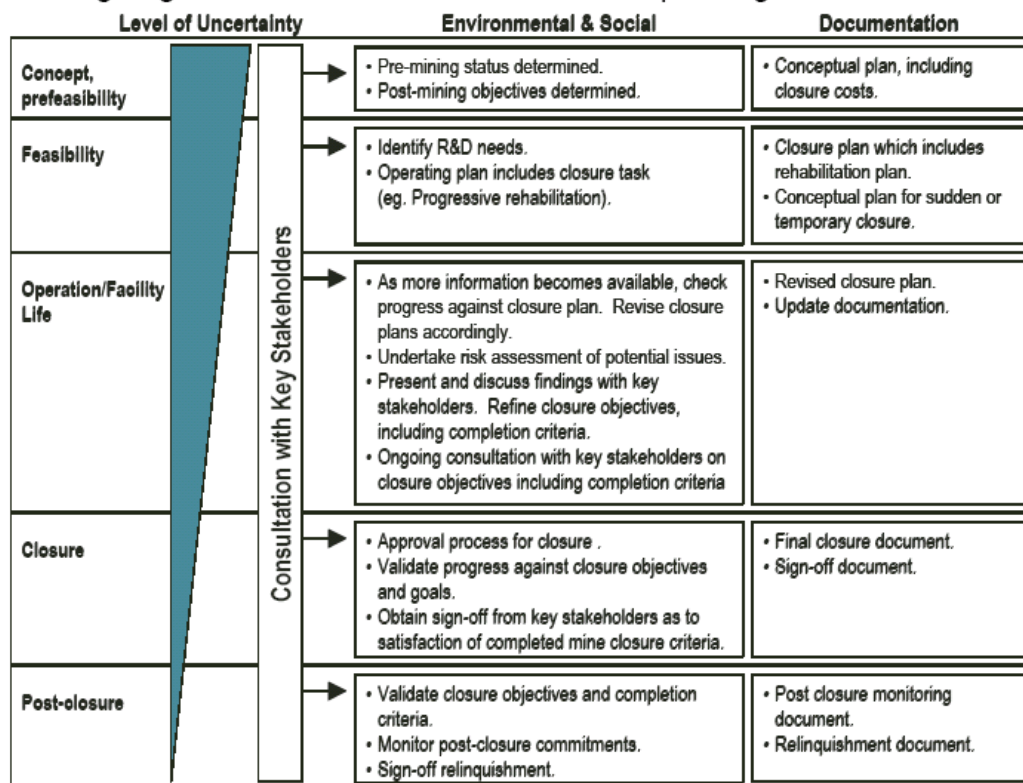


Fig 1 - Diagrammatic representation of the Closure Planning Process

Courtesy BHPBilliton HSEC Guideline No G08, rev11

REHABILITATION PLANNING METHODOLOGY

It is important to develop a plan for rehabilitation. An example of a staged rehabilitation process as used by BHPB Illawarra Coal is given diagrammatically in Figure 2. The following is a “blueprint” for the compilation of a Rehabilitation plan that has been developed during the experiences to date between BHPB and it’s consultants. It covers the considerations, documentation, interactions and detail required when formulating a rehabilitation plan from the conceptual stage through to the final relinquishment of the mining lease.

MOP/AEMR/MREMP

Issues to be considered are:

- Care and Maintenance Plan;
- BHPB developed options for future usage and initial Risk assessment; and
- Mine Rehabilitation and Closure Plan and the Mining, Rehabilitation and Environmental Management Process as per DMR Guidelines

Initial Stakeholders Presentation (ISP)

Firstly internal stakeholders need to develop the initial project concept with representatives from site, Environmental Dept, survey and media department. Subsequently the following stakeholders need to be involved.

Government Agencies

Government agency meetings including site inspections so that identification of relevant studies to assist the rehabilitation process can be determined. Agencies could involve NPWS; DMR; Local Council; EPA; DIPNR; RTA; SCA; Department of Fisheries etc.

Community

Consultative committees can be formed where relevant. Alternatively media releases can be used to inform and seek community input.

Unions

Involvement and communication with site unions.

Initial Rehabilitation Plan (IRP)

Undertake Relevant Studies

These can incorporate for example the following:

- Site Survey;
- Site Soil Contamination Assessment;
- Geotechnical;
- Aboriginal Heritage;
- European Heritage;
- Flora & Fauna;
- Acoustic;
- Traffic investigation; and
- Drainage.

Prepare Initial Rehabilitation Plans (individual plans for each site)

Typical format and content for a rehabilitation plan could include:

Section 1

Introduction;
Background;
Purpose of report;
Consultation undertaken;
Land Zoning; and
Existing Approvals.

Section 2

Proposed Works

Section 3

Existing Environment, Impacts and Safeguards focusing on items such as:

- Land Use;
- Topography, Geology and Soils;
- Flora and Fauna;
- Drainage and water quality;
- Ecology;
- Visual;
- Infrastructure;
- Cultural and heritage;
- Traffic;
- Air Quality;
- Noise; and
- Waste and hazardous materials.

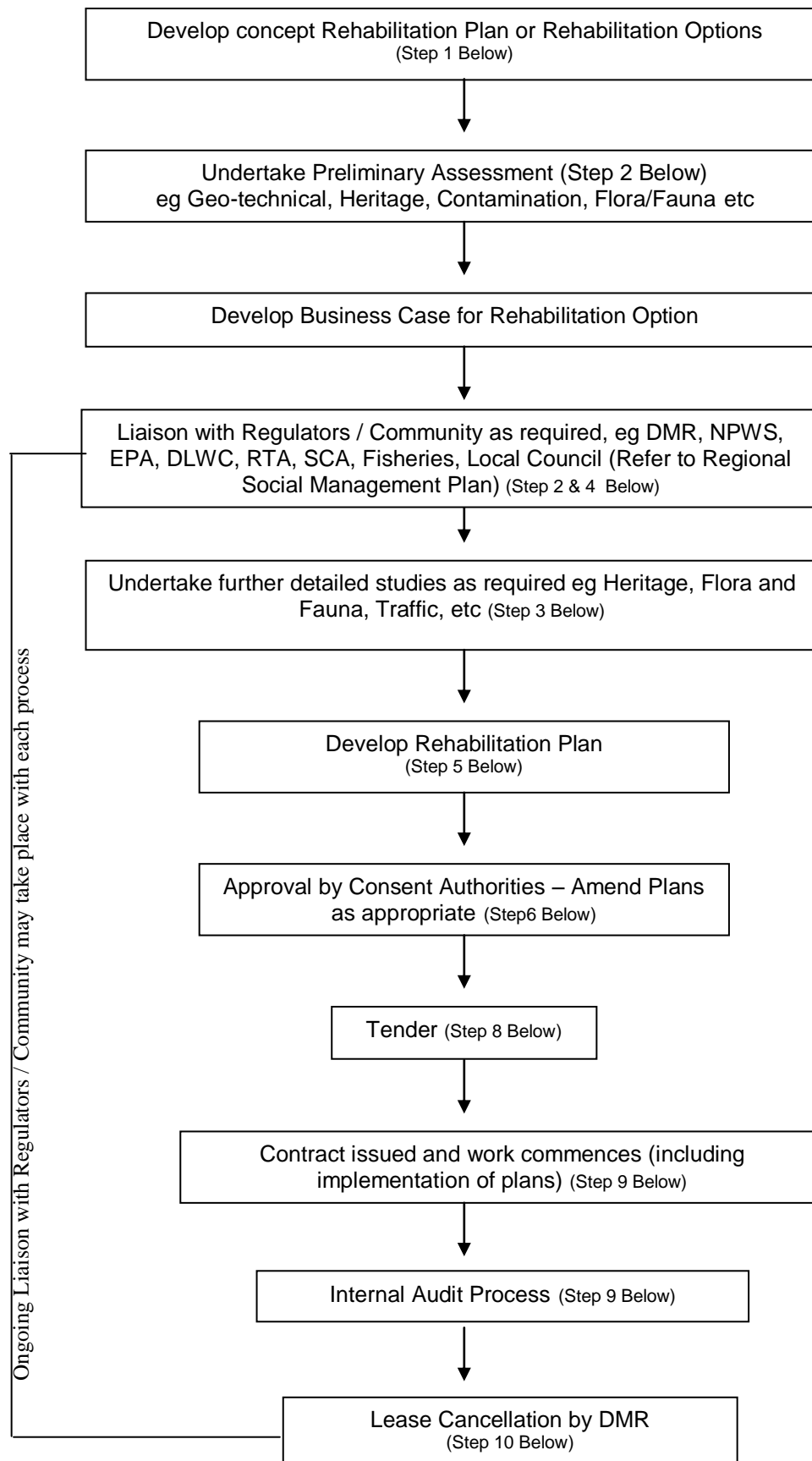


Fig 2 - Illawarra Coal Rehabilitation Process Flow Chart

Section 4

A summary of the following areas:

- Environmental Management;
- Management structure;
- Community liaison;
- Revegetation maintenance; and
- Risk Management.

Ongoing monitoring

Stakeholder Review Of IRP

This stakeholder review of the initial or draft rehabilitation plan can be achieved by the following mechanisms:

- Inter government agency presentations;
- Community presentations;
- Stakeholder responses addressed by BHPB; and
- Additional or expanded studies as or if required.

Revised Initial Rehabilitation Plan (RRP)

Following the stakeholder review of the initial rehabilitation plan it may be necessary to revise the plan based upon an assessment of stakeholder responses.

RRP Approval

Once the review of feedback is undertaken and modifications, if required are complete the plan can now be submitted for approval to Dept Mineral Resources.

Prepare Detailed Construction Plan (DCP)

With formal approval received and a review of the conditions of consent undertaken it is then necessary to:

- Incorporate conditions of consent into documentation,
- Prepare works procedures;
- Prepare construction drawings;
- Prepare contract documentation;
- Prepare Management Plans;
- Seek stakeholder input of DCP and subsequently address any responses; and
- Amend DCP where appropriate.

Tender

- Seek expression of interest from contractors for construction works
- Prepare list of selected tenderers
- Obtain competitive tender prices
- Assess tenders
- Let contract

Construction

- Undertake Risk Reviews
- Establish Work Procedures
- Carry out construction works
- Prepare works as executed documentation
- Certification of works by Engineers
- Sign off of works by DMR

Relinquishment Of Mining Lease

- Rehabilitation monitoring complete
- Sign off by DMR.

Transfer Of Land Ownership (As /If Required)

EXAMPLES OF PLANNING FOR MINE CLOSURE

Illawarra Coals assets include operations that date back over a hundred and fifty years as well as newer operations that are only about thirty years old. A brief summary of the issues associated with these two differing cases are set out below.

EXAMPLE 1 – KEMIRA COLLIERY

History of Operations

The Kemira Colliery was initially called the Albert Mine and mining commenced in 1848 when two tunnels were driven into the Balgownie and Wongawilli seams. In 1856, the Albert tunnels were abandoned, and a new tunnel was opened into the Bulli Seam (later called No. 1 Seam) a short distance higher on the slopes of Mt Keira. In 1861, a tramway was constructed which connected Mt Keira Mine to Wollongong Harbour. In 1883, No. 1 Shaft was sunk to the No 2 Seam to improve ventilation.

In 1937, Australian Iron and Steel purchased the Colliery to satisfy the increased demand for coal generated by the rapid expansion of the Port Kembla Steelworks. In 1955 the name changed to “Kemira”, derived from combining *Kem*-bla and *Ke-ira*. In 1956 the sinking of No 1 Calyx Shaft begun and in 1957 the sinking of No2 Calyx Shaft begun.

In 1969 the mine reached its peak manning level at 497 men. The Peak Yearly production was achieved in the year ending November 1979 with 770,684 tonnes of coal mined.

In 1982, the market conditions in the steel industry resulted in two-thirds of the workforce being retrenched and in 1991 mining ceased. At the time of closure the Colliery was Australia's longest operating underground coalmine. By the end of 1995 sealing had been completed at the mine's two tunnels and four shafts. Australian Iron and Steel became a wholly owned subsidiary of the BHP Group which later formed to BHP Billiton, the current owner of the property.

Site description

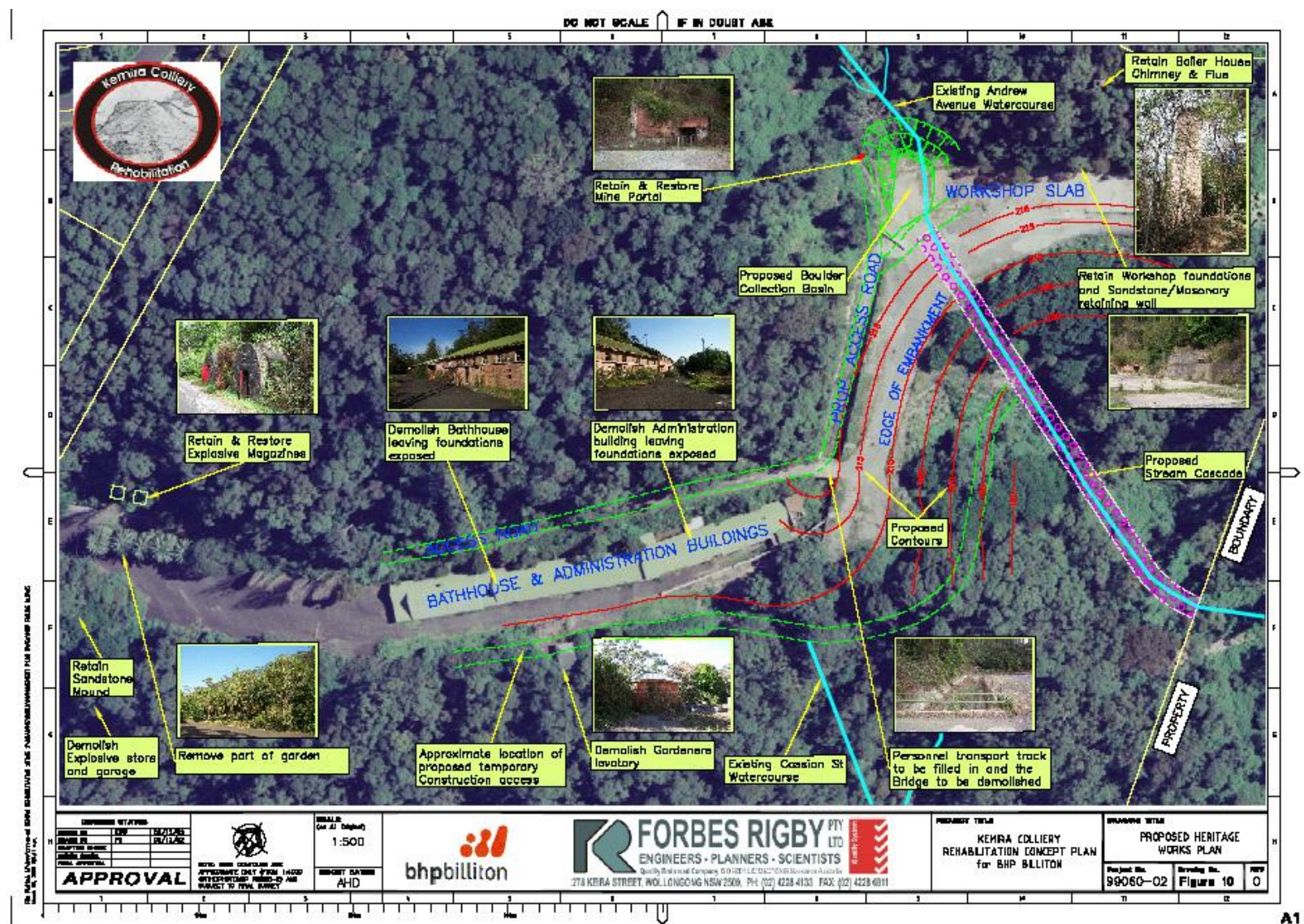
Figure 3 shows the layout of the site. The colliery pit top and platform are the primary focus of the rehabilitation works.

The mine platform consists of a long narrow bench with several buildings constructed on the platform, which serviced the mine during its operation. The now disused mine portal exits from the side of the hill approximately mid way along the platform.

Kemira Rehabilitation Strategy Objectives

The major objectives of this rehabilitation plan are as follows:

- Redirect and modify the drainage on the site so as to stabilise the site and control the flow to minimise scour;
- Recontour slopes to gradients that significantly improve long term stability;
- Continue the natural flow regime of the water courses which cross the site;
- Preserve the industrial and cultural heritage items of significance on the site;
- Minimise visual impact of the rehabilitation by revegetating the area disturbed by mining and recontouring activities;
- Restore and enhance the ecological integrity of the site as habitat for native vegetation and fauna;
- Complete the rehabilitation to the satisfaction of the Department of Mineral Resources so that BHP Billiton can relinquish the Coal Lease on the site;
- Achieve a sustainable outcome to make possible transfer of ownership of the site;
- Remediate the site to a standard appropriate for the recreational open space; and
- Ensure the site is clear of contaminants of concern that may pose an unacceptable risk of harm to human health and/or the environment.



qq.

Fig 3 - Kemira Site Location and Summary of Rehabilitation Scope

Strategies employed to achieve these objectives will include, but not be limited to the following:

- Minimise potential of cross catchment flows between Andrew Ave and Cassian Street catchments;
- Stabilise the embankment area (northern area) by reducing the slope, removal of fill, establishing appropriate subsurface drainage and vegetation;
- Construct a stream cascade down the embankment to prevent scour;
- Revegetate all disturbed areas and implement a maintenance programme to ensure the longevity of established vegetation;
- Minimise impacts to downstream vegetation by managing on site and reducing sediment and pollutant loads leaving the site;
- Involve relevant stakeholders including government departments and members of the community in contributing to the plans for the site; and
- Implement a long-term maintenance strategy;

Project Works Overview

The major tasks involved in the proposed rehabilitation programme are summarised below:

1. Site establishment including establishment of erosion and sediment control structures;
2. Demolition of existing buildings and structures and removal of debris;
3. Clear and mulch existing vegetation in a staged manner;
4. Excavate material from top of embankment and basin (starting from top and working down). This to occur in a staged manner with fill material sorted as required and taken to an off site location;
5. Construct gabion lined stream cascade;
6. Final landform profiling, and construction of permanent drainage works (eg contour drains);
7. Site re-vegetation and regeneration. This to be commenced as soon as possible once final land profiling to each area is completed;
8. Restore heritage structures remaining on site as appropriate; and
9. Remove temporary erosion and sediment controls once revegetation established.

It is estimated that the construction works will take approximately 30 weeks to complete.

The gabion stream can be seen in the left hand side of the picture in Figure 4 whilst the centre of the picture shows the finished land profile after placement of topsoil and geo fabric matting for stability. Subsequent steps will be the spraying of a seed mix and planting of selected tube stock.



Fig 4 - Photo showing works in progress.

The Risk Assessment identified 38 risks that needed specific management actions. The most significant risks that need to be managed during the construction works are associated with:

- The safety of road users while removing fill from the site;
- The safety of construction personnel on site during the works; and
- The effect of storm run off from the site on downstream property owners and residents.

The risk assessment identified the need for a variety of Management Plans to address controls over the identified impacts. The Construction Management Plans that have been developed include:

1. Surface Water and Erosion & Sediment Control;
2. Construction Traffic;
3. Remediation Action Plan;
4. Revegetation and Regeneration;
5. Conservation; and
6. Environmental Management Plan.

In addition a Maintenance Management Plan has been completed. A diagram illustrating the inter-relationship of documents is given in Figure 5.

KEMIRA COLLIERY REHABILITATION DOCUMENT SUMMARY

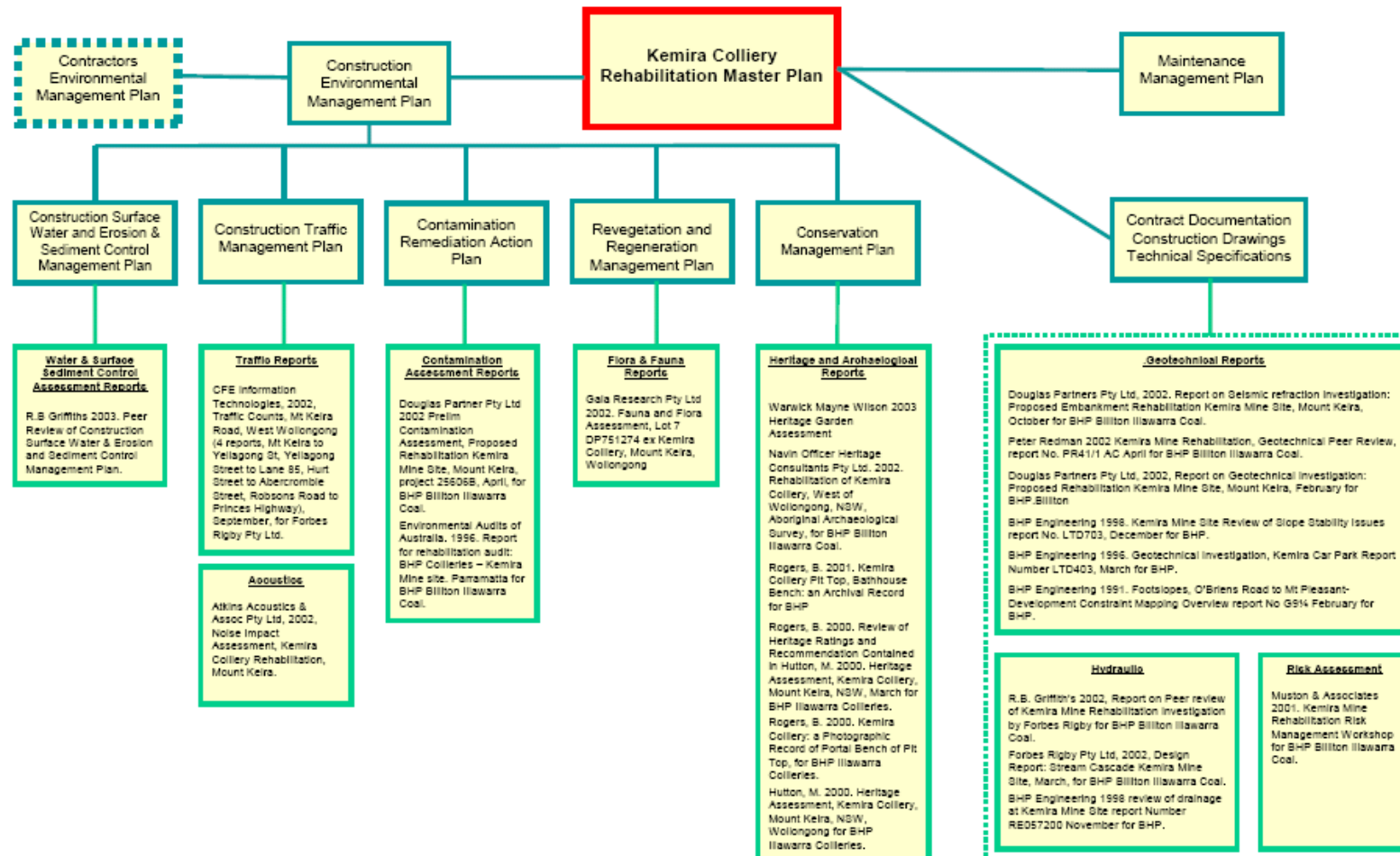


Fig 5 - Interrelationship of Documents for Kemira Rehabilitation

Stakeholder Consultation

Stakeholder consultation has been primarily undertaken via the following mechanisms:

Keiraville community meetings

Community involvement with the Rehabilitation of Kemira Colliery began in 1998 with Illawarra Collieries involvement in the "Cassian Street/Keiraville Community" meetings held regularly in the Cassian Street cul-de-sac, Keiraville. These meetings are organised and chaired by Wollongong City Council. These meetings have been well attended by local community members and provide a forum for the exchange of information including plans for the Kemira site and concerns of the community.

Government agencies

In order to gain exposure to the project from a range of expertise, BHP Billiton has sought advice from a variety of agencies prior to submitting the Rehabilitation Plan to DMR for approval.

In January 2002, BHP Billiton facilitated a workshop seeking involvement in the Rehabilitation Plan from a range of government agencies. The agencies involved were:

- Department of Mineral Resources;
- Department of Land and Water Conservation;
- Environment Protection Authority;
- National Parks and Wildlife Service;
- Roads and Traffic Authority; and
- Wollongong City Council.

Subsequent workshops were held in May 2002, September 2002 and November 2002 to which all the above agencies were invited.

In addition to the above, many meetings have been held with appropriate agencies regarding the detail of individual Management Plans. Several amendments have been made to the plans in accordance with agency's comments.

Broader Community Consultation

Community consultation has been undertaken to inform people about and seek their input on the rehabilitation project at the Kemira Colliery. Consultation has sought to identify community preferences and identify matters of concern relating to the options regarding rehabilitation of the site. Throughout the various phases of the project, a variety of community consultation methods were used.

Kemira Community Consultative Committee

During the community consultation in June 2002, a number of people suggested a Community Consultative Committee be formed to assist in identifying and addressing concerns about the project. As a result a Committee was established in September 2002 and meetings are held approximately monthly.

Some of the specific objectives of the committee are to:

- Help identify, pre-empt and resolve issues of concern to minimise the impact of potential problems involved with the Rehabilitation Works;
- Assist members understand the project, including potential benefits and concerns;
- Consider ways of appropriately communicating information about the project to the wider community, and assisting with flow of information;
- Provide member of the community with relevant information regarding the Rehabilitation Works; and
- Assist in ensuring a transparent consultation process.

The group has representation from the following groups:

- NSW Member for Keira; Wollongong City Councillors; Neighbourhood Committee Number 5; Mt Keira residents; Keiraville residents; Illawarra Escarpment Coalition; Mt Keira Demonstration School P & C; and representatives from BHP Billiton.

- An independent facilitator chairs the Committee. The committee has been an effective forum to exchange information and concerns regarding the project. The Committee has been involved in the formation and detail of the Management Plans. External agencies have also been involved in committee meetings as required.

This group will continue to meet regularly throughout the construction works. If required, the group may continue to meet post-construction.

EXAMPLE 2 - CORDEAUX COLLIERY

Site location and description

Cordeaux Colliery consists of facilities at various locations. The sites include:

1. Cordeaux Pit Top site (including Cordeaux No 1 and No 2 Shafts);
2. Corrimal No 2 Shaft site;
3. Cataract Weir Pump Facility;
4. Corrimal No 3 Shaft Site;
5. Corrimal No 3 Shaft Coal Bins;
6. Cordeaux Re-injection Borehole Field; and
7. Wilton Spray Irrigation Area.

The pit top is located adjacent to Picton Road, approximately 20 km north west of Wollongong. All sites (with the exception of the Wilton Spray Irrigation Area) are within the Sydney Catchment Area, which forms part of the water supply system for Sydney and the Illawarra.

History of Operations

Construction of the mine commenced in 1976 with the first coal being produced from the underground workings in 1980. In 1985, Cordeaux holed into Corrimal Colliery workings to officially merge the two collieries in January 1986. The Collieries Division of BHP Coal (*now BHP Billiton Illawarra Coal*) operated the mine continuously over the ensuing 21-year period until closure.

The economically minable Bulli Seam coal in the Cordeaux Colliery reserve area was depleted by the year 2001. The colliery ceased coal production on the 23 March 2001 and placed on care and maintenance on 14 April 2001.

The colliery was originally planned to be a two-seam operation at some time during its life. The Bulli Seam was the first to be worked with the Wongawilli Seam to be eventually mined and used as a blending component for coke making at the Port Kembla Steelworks. This plan was approved in 1995 and two sets of access drifts from the Bulli seam workings to the lower seam completed. The plan was subsequently put on hold due to market conditions and high mining costs.

Mining was undertaken using the longwall method, with up to two continuous miners preparing development roadways. The mine produced a high quality Bulli Seam product with an initial raw coal output of 31,300t in 1979 and a peak output of 2.97 Mt in 1994. The majority of coal produced at the mine (80%) went to BHP Port Kembla Steelworks and the remainder exported overseas. All coal from the mine was washed at the Steelworks.

The pit top covers approximately 8 ha of land on the Picton Road site and includes the majority of infrastructure required to support the mining operation. Men and materials entered the mine through a downcast shaft. Raw coal from the mining units was moved by conveyor through the workings and delivered to two 600t underground bins. Coal was then transferred from the bins to a bulk winder and brought to the surface via an up-cast shaft. On the surface, the bulk winder delivered the coal to a fully enclosed elevating conveyor that, discharged to two 1200t storage bins. Coal was bottom-loaded from the bins into trucks and transported on public roads to O'Brien's Drift.

The coal haulage route from Cordeaux to Port Kembla was along Mount Keira Road and Harry Graham Drive to O'Brien's Drift. The coal was conveyed through the drift to Kemira Valley Train Loading Facility then transported to the Steelworks along the BHP rail system to Port Kembla.

In addition to the two service shafts on the pit top, Cordeaux workings incorporate two remote ventilation shafts. Corrimal No 2 Shaft is used as a regulated intake while Corrimal No 3 upcast shaft provides additional ventilation for the workings via a mine fan mounted on the shaft. The locations of these shafts are shown on Figure 6.

Proposed and Future Operations.

The medium-term objective for Cordeaux Colliery is for the mine to remain on care and maintenance until longer-term options can be fully developed and implemented.

Under the care and maintenance program the pit top site shall remain virtually unchanged with all major buildings (excepting the Technical and External Services administration offices) and structures being retained. All non-fixed equipment and mine supplies on the pit top that are not considered necessary for the longer-term operations of the site will be progressively removed. The maintenance and upkeep of this infrastructure will continue until a decision on the ultimate future of the mine is made. During this period, the site will remain manned. Other than ongoing maintenance work the site will continue to be occupied by BHPBIC Technical & External Services personnel who are currently accommodated within Cordeaux mine's administration complex.

Recovery of underground mining equipment has ceased and ventilation seals at pit bottom have been completed and both Cordeaux ventilation shafts have been capped at the surface collar level. The steel capping structure is of a design that will prevent unauthorized entry and capable of being dismantled if a requirement to re-commission the shaft occurs. The mine ventilation fans have been turned off.

The longer-term objective for Cordeaux Colliery currently under consideration is to incorporate the workings into the future development of the new Dendrobium Mine. Preservation of the remaining Cordeaux coal reserve will ensure that a potential source of Wongawilli Seam (No 3 seam) coal is available to meet a 30-year supply contract to the Port Kembla Steelworks. The option would entail developing the remaining Cordeaux coal reserve for extraction through the Dendrobium workings.

The future development option for mining operations would require the re-commissioning of a significant portion of the existing pit top infrastructure and services to augment the support facilities operating for Dendrobium. The surface facilities that are likely to be incorporated in the development option include the up-cast ventilation shaft and fan, the man and materials winder and shaft and power, water and compressed air services. The bathhouse, workshop, bulk storage, yard storage and handling area, car park and sundry buildings may also be required for limited use.

Department of Mineral Resources Requirements

Rehabilitation of the Cordeaux Mine site is covered by the Mining Act 1992 and by lease conditions attached to the mine by the Department of Mineral Resources (DMR). The DMR encourages progressive rehabilitation of mine sites, and has a series of criteria to be met for rehabilitation works. Lease conditions require the preparation of an MOP, which incorporates a rehabilitation plan where appropriate.

The clauses in Consolidated Lease 768 pertaining to rehabilitation works include:

Consolidated Coal Lease No 768 Coal Mining Act, 1973 Part B Clause 70

"Upon completion of mining operations and before vacating the area, the registered holder shall restore the surface working areas other than the sealed access roads and shall establish vegetation to the satisfaction of Sydney Catchment Authority when so directed."

Consolidated Coal Lease No 768 Coal Mining Act, 1973 Part B Clause 82

"At the conclusion of operation and before the lease is terminated, the registered holder will be required to remove such installations and works as may be determined by Sydney Catchment Authority and to restore the whole area with vegetation of a type specified by Sydney Catchment Authority to the satisfaction of the Minister."



Conditions of Consolidated Coal Leases, 1985 Clause 25

“Upon completion of operations on the surface of the subject area or upon the expiry or sooner determination of this lease or any renewal thereof, the registered holder shall remove from such surface such buildings, machinery plant, equipment, constructions and works as may be directed by the Minister and such surface shall be rehabilitated and left in a clean, tidy and safe condition to the satisfaction of the Minister.”

Proposed Rehabilitation Works

Overviews of the proposed works planned for each of the Cordeaux Colliery sites are outlined below. Individual Rehabilitation Plans will be generated on a site-specific basis.

Cordeaux Colliery Pit Top Site

Since coal mining ceased at Cordeaux the surface infrastructure has remained in operation to support the care and maintenance status of the mine. To meet the medium and long term objectives for the mine, the majority of existing surface facilities will be retained as part of the future option to re-establish coal extraction operations in the reserve. Redundant facilities will be demolished.

All site roads, paved areas, service reticulation, stormwater drains and bathhouse effluent systems will be retained.

Corrimal No 2 Shaft site

The proposed works for the rehabilitation of Corrimal No 2 Shaft encompass the shaft itself, the surrounding structures, the access track from the fire road and the adjoining shaft material emplacement. The buildings are in excess of 20 years old and are generally secure steel clad structures with concrete floors in reasonable to good condition.

Remediation work will be conducted in stages and include:

1. Upgrading of the existing access track leading to the Corrimal No 2 Shaft to facilitate heavy vehicle access to the site;
2. Demolition and removal of mine debris;
3. Exposure of shaft and preparation for filling by import of approved fill;
4. Sealing of shaft with engineer approved concrete capping as required by the DMR;
5. Revegetation of all disturbed areas rendering it non-trafficable to motor vehicles but accessible for long-term maintenance requirements.

Corrimal No 3 Shaft Site

A company decision is still to be finalised as to whether the No.3 shaft is required as an additional ventilation source for the future Dendrobium workings.

If it is decided that the Corrimal No 3 Shaft will not be required, then the No 3 Shaft will be backfilled and sealed as required by the DMR. If the shaft is required then a temporary cap will be placed.

Corrimal No 3 Shaft - Coal Bins Site

The two 1200 tonne coal bin structures, service pipelines and paved areas on the site adjacent to Picton Road will be demolished and all building materials removed/recycled.

Cataract Weir Pump Facility

The water supply pump and pontoon were dismantled and removed in June 2003. It is proposed to remove the overland pipeline and the transmission line to the No 3 Shaft site. When the overland pipeline and transmission line have been removed the 1.25 km long, access track will be left as is, at the request of the SCA, as an access road to the waters of the Cataract Dam

Wilton Spray Irrigation Area

All mine water distribution pipe-work, sprays and ancillary equipment on the Wilton irrigation site will be removed. As the 14 km long supply pipeline from Cordeaux is buried along its entire route it will be left in place and not recovered.

The 40 ha spray irrigation site is on open grass pastureland. Regeneration work on the site will include removal of lines and soil remediation. Once the soil treatment is complete and normal pasture growth is achieved any temporary erosion protection measures will be removed.

Cordeaux Re-injection Borehole Field

This site has already been rehabilitated with works completed to the satisfaction of the Sydney Catchment Authority.

Final Land Use

The Corrimal No 2 Shaft site, Corrimal No 3 Coal Bins, Weir Pump facility, Cordeaux Re-injection Borehole Field and associated access tracks and service corridors are established on leasehold land owned by the Sydney Catchment Authority. Consequently the former mine sites will be rehabilitated, coal leases relinquished and the lands once again become solely part of the Sydney Catchment Authority area. There are no alternative land use options proposed.

When the Wilton Spray Irrigation area was in use the land was subject to the dual purpose of disposal of excess mine water and stock grazing. Since the irrigation system ceased operation, intermittent stock grazing has been the sole land use for the area. This land use will continue.

The Cordeaux Colliery Pit top and No 3 Shaft site will remain on care and maintenance as part of the mining lease until final requirements have been determined. At which stage mining operations will either continue (with rehabilitation works being completed at the end of the next phase of mining operations) or the sites will be rehabilitated with the mining lease being relinquished.

Closure and Rehabilitation Criteria

The rehabilitation objectives include maintaining BHP Billiton Illawarra Coal HSEC standards, meeting DMR requirements in relation to lease relinquishment and satisfying the requirements of the landowner and relevant government authorities. The standard of rehabilitation to be achieved is specified in the coal lease and is conditional on the lease being relinquished.

Ongoing Maintenance

As part of the rehabilitation process ongoing maintenance management plans will be developed for all sites, with maintenance being carried out until revegetation has been successfully established and the lease relinquished or in the case of Cordeaux Colliery Pit Top and Corrimal No 3 Shaft site until mining operations commence again.

Environmental Management Controls*Air and Noise*

Air and noise pollution will be managed in accordance with current guidelines.

Surface Water And Erosion And Sedimentation

Measures will be taken to control the risk of surface waters and erosion and sedimentation impacts on the sites to be rehabilitated. The measures will be aimed at controlling the velocity and concentration of surface runoff water from rainfall events and flows in natural watercourses.

To control sedimentation impacts, when the mineshafts are being backfilled and capped and the site infrastructure is being removed, silt fencing will be erected along the lower side of each work area.

The "Track Stabilisation & Erosion Control Manual" as produced by Sydney Water will be utilised as a guideline when developing control strategies.

The risk of erosion and/or sedimentation resulting from the rehabilitation works will be minimised by the development and application of site-specific Erosion & Sedimentation Control Plans.

Ground Water

Controls will be put in place to appropriately manage potential groundwater pollution

Contaminated Land

Rehabilitation planning for the shaft sites will include investigations to identify any areas of land that may have been contaminated by past mining activities. If any areas of contamination are discovered the material will be either treated in-situ or removed and legally disposed of elsewhere, depending on the degree and type of contamination.

Hazardous Materials

Rehabilitation planning will include investigations to identify any hazardous materials present from past mining activities or in buildings. Where required, the removal and disposal of oils from machinery gearboxes or transformers etc will be handled off site at appropriate disposal depots. If hazardous products, such as asbestos or stored chemicals, are found during the health, safety and environmental risk analysis of the buildings, appropriately qualified contractors will be engaged to remove and dispose of these products.

A prerequisite of demolition and disposal will be compliance with requirements of the OH&S Act and subject to health, safety and environmental risk analysis prior to the commencement of any works.

The opportunity to recycle any of the building materials or whole structures will be canvassed with prospective demolition contractors.

Flora and fauna

All necessary flora and fauna studies will be carried out for the mine sites to be rehabilitated. Weeds will be controlled when required. Weed management will be incorporated as required into the maintenance activities to be undertaken.

Landform and revegetation

Land restoration issues associated with landform design, surface shaping, surface preparation and regeneration shall be fully addressed. The plan will ensure the restored sites are stable, will resist surface water and erosion and sedimentation impacts with revegetation using local native flora species that will provide suitable habitat for use by fauna species occurring in the region. The finished landform will be contoured to blend with the surrounding area.

All necessary flora and fauna studies will be carried out with rehabilitation works to meet the requirements of any studies completed.

A full ongoing maintenance programme for the restored sites will be developed to ensure ongoing maintenance, revegetation monitoring (and any necessary re-establishment) until the revegetation has successfully established and the mine lease can be relinquished.

Blasting.

It is not anticipated blasting will be undertaken in association with the rehabilitation works

Visual, Stray Light.

The mine site will not change in its general overall appearance and will not present an adverse visual impact. Nighttime security lighting of the site will not be altered. This degree of illumination is not expected to any adverse stray light impacts.

Aboriginal Heritage

A number of Aboriginal sites occur above the previously mined areas at Cordeaux. These sites were identified by archaeological survey and managed during extraction as required by the National Parks and Wildlife Service and the Department of Mineral Resources.

All necessary aboriginal and archaeological studies will be carried out. Studies undertaken to date have not found any aboriginal heritage place or artifact that would be affected by the works. Refer Navin Officer Pty Ltd Aboriginal Archaeology study.

Natural Heritage

All heritage studies necessary will be carried out for each site to be rehabilitated.

Bushfire

Bush fire management plans have been developed with the Local Bush fire Service. Protection measures are implemented and a hazard reduction programme is in place. The programme involves periodic burning off that is usually undertaken by the local Bush Fire Brigade or the Sydney Catchment Authority, on an as needed basis.

An Emergency Procedure is in place for controlling fire in or adjoining the Cordeaux pit top facilities and provides details on emergency contacts. This procedure is contained within the Fire Control Emergency System.

The Cordeaux pit top site has been used as a strategic base for fighting large fires in the surrounding catchments and farming lands. The closed status of the mine will not affect its future use for fire fighting.

Public Safety

All of the Cordeaux pit top area is contained within a chain wire cyclone mesh fence. The main entrance from Picton Road has a gate, which is locked when the site is unmanned.

All perimeter fences are signed to warn against unauthorised entry and potentially dangerous areas.

Visitors to site must register upon entry.

The entrance from Picton Road to the Corrimall No 2 and 3 Shaft sites is secured with a locked gate and chain wire fencing.

The Corrimall Nos 2 and 3 mine Shafts will be permanently sealed and all associated infrastructure, such as unused buildings, demolished and removed, as part of the mine closure plan. This will ensure that the rehabilitated mine sites do not pose any future public risk.

If it is the decision to temporarily cap the No 3 Shaft, a suitably designed access proof cap will be installed. The existing security fence would be retained.

Risk Management Review

Risk assessments will be undertaken for each site as required. BHPBilliton will conduct an internal risk management review to determine risks and hazards and the likely consequences of each identified risk/hazard.

The primary objectives of the reviews are to identify the major safety and environmental risks that could occur during construction works, and to identify the actions required to minimise the potential occurrence and/or consequences of these risks.

Final Land Use

The Corrimall No 2 Shaft site, Corrimall No 3 Coal Bins, Weir Pump facility, Cordeaux Re-injection Borehole Field and associated access tracks and service corridors are established on leasehold land owned by the Sydney Catchment Authority. Consequently the former mine sites will be rehabilitated, coal leases relinquished and the lands once again become solely part of the Sydney Catchment Authority area. There are no alternative land use options proposed.

When the Wilton Spray Irrigation area was in use the land was subject to the dual purpose of disposal of excess mine water and stock grazing. Since the irrigation system ceased operation, intermittent stock grazing has been the sole land use for the area. This land use will continue.

The Cordeaux Colliery Pit top and No 3 Shaft site will remain on care and maintenance as part of the mining lease until final requirements have been determined. At which stage mining operations will either continue (with rehabilitation works being completed at the end of the next phase of mining operations) or the sites will be rehabilitated with the mining lease being relinquished.

Stakeholder and Community Consultation

The consultation process commenced in March 2003 with a formal presentation to government agencies of Cordeaux Colliery's Asset Preservation Plan outlining the proposed future rehabilitation of the Cordeaux sites. This presentation included the recently formed "interagency group" consisting of DMR, DSNR, EPA, NP&WS, SCA and WCC. Site inspections were held for all the sites for all interested parties.

This consultation process was continued with another presentation by BHPBIC updating progress to the interagency group in October 2003.

Future stakeholder and community involvement will include:

1. Agency review of site-specific rehabilitation plans,
2. Re-issue of the site specific plans incorporating where practical/possible comment from agencies,
3. On going presentations during the rehabilitation process (such as at the interagency meetings),
4. Site inspections as necessary for government agencies during the rehabilitation process,
5. Placement of posters/newsletters at the Appin shop front and in Wollongong City Council offices as appropriate, and
6. Provision of a 24-hour hotline for general queries from the community.

CONCLUSIONS

Planning for mine site rehabilitation is a long and detailed process that should be accounted for from the design concept stage of any new operation and be kept in mind during all stages of the mines life cycle.. The issues that need to be addressed in any rehabilitation process are as broad and need the same level of detailed investigations and planning as does the initiation of a project.

Experiences at BHPBilliton operations in the Illawarra show the benefits of planning for rehabilitation whilst the mine is still operating.

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"Kemira Rehabilitation Master Plan, March 2003"

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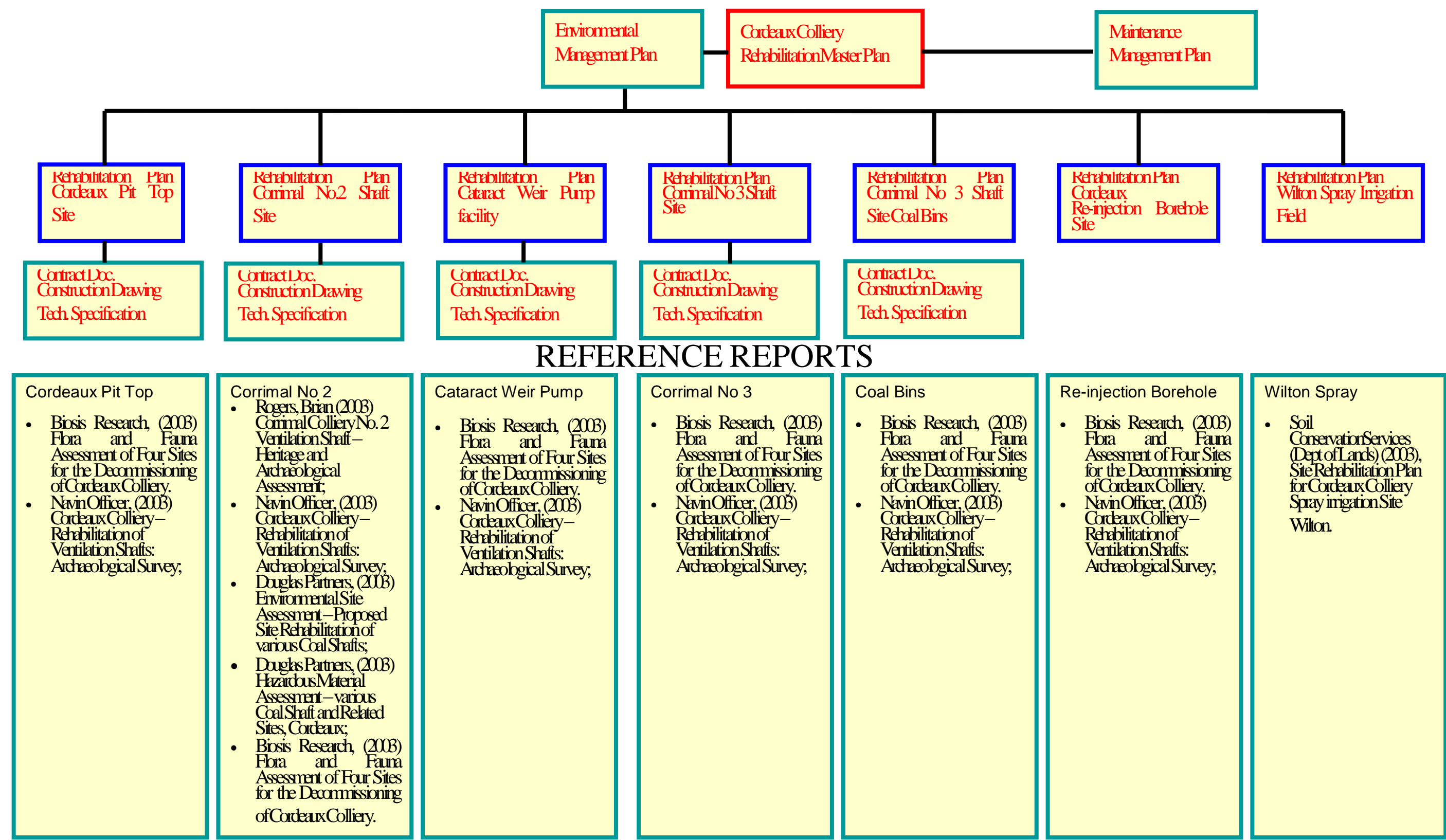
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DMR Policy "Guidelines to the Mining, Rehabilitation and Environmental Management Process", December 2002.

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FR Ref 103071 Doc Summary Rev 1 - 19 November 2003

Fig7 - Cordeaux Colliery Document Summary

ENVIRONMENTAL ASPECTS OF MINE PLANNING

Wendy Tyrrell ¹

ABSTRACT: Underground mining has occurred in the Illawarra region for more than 150 years. Over time, community expectations about the environmental effects of mining and other developments have changed dramatically. It is in this context that community, government, and environmental groups have raised concerns about the effects that mining can have on rivers and other natural features, residential property, and infrastructure, such as transmission lines, roads, and bridges. On natural features, these effects can include fracturing of the rock bedding in the river, water loss to the shallow sub-strata, gas release, rock falls, and vegetation dieback.

In the past, mine planning has considered potential effects on engineered structures, however, the same attention has not been paid to the effects of mining on natural features. Through a Stakeholder Involvement Programme conducted by Illawarra Coal, it was identified that government and community stakeholders were seeking a more sensitive approach be taken to mine planning, particularly in significant and high risks areas. As a result, mine planning processes are being reviewed to take into account the effects of mining on the surface features. The focus of this paper is the programme undertaken by BHP Billiton – Illawarra Coal to integrate environmental assessments into mine planning. This has involved internal workshops and projects, together with external consultation. The process that the company is utilising to fully incorporate environmental assessments into the mine planning process is described.

The purpose of this process is to identify surface features, determine their sensitivity, develop mitigation and remediation options, and potentially avoidance measures early in the mine planning process. Another key aspect of the process is to incorporate internal and external stakeholder feedback as part of the mine planning process.

In implementing this process, it is anticipated that it will provide a more secure outcome for the business. This is expected to occur through understanding and addressing environmental issues and incorporating stakeholder feedback early in the mine planning process, therefore minimizing the risk of costly changes to mine plans being required within short time frames.

The rigorous Integrated Mine Planning Process (IMPP) will give the business confidence that the mine plan submitted through the Subsidence Management Plan (SMP) application to government has the highest probability of approval and will provide the most sustainable outcome – for the environment, community, and Illawarra Coal.

INTRODUCTION

BHP Billiton - Illawarra Coal is a coal mining business operating in a sensitive locality about 1.5 hours drive to the south of Sydney, Australia are shown in Figure 1.

The Illawarra Coal business produced approximately 7 million tonnes of premium-quality coking coal during the financial year to June 2003. The coking coal is utilised in the production of steel at the Port Kembla Steelworks in New South Wales, Whyalla Steelworks in South Australia, and by overseas customers.

Mining has occurred in the Illawarra region for more than 150 years. Over time, community expectations about the environmental effects of mining and other developments have changed dramatically.

In this sensitive area, the operations must be sustainable within the context of urban pressure and sensitive ecological environments.

¹ Manager Environment and Sustainable Development, BHP Billiton – Illawarra Coal

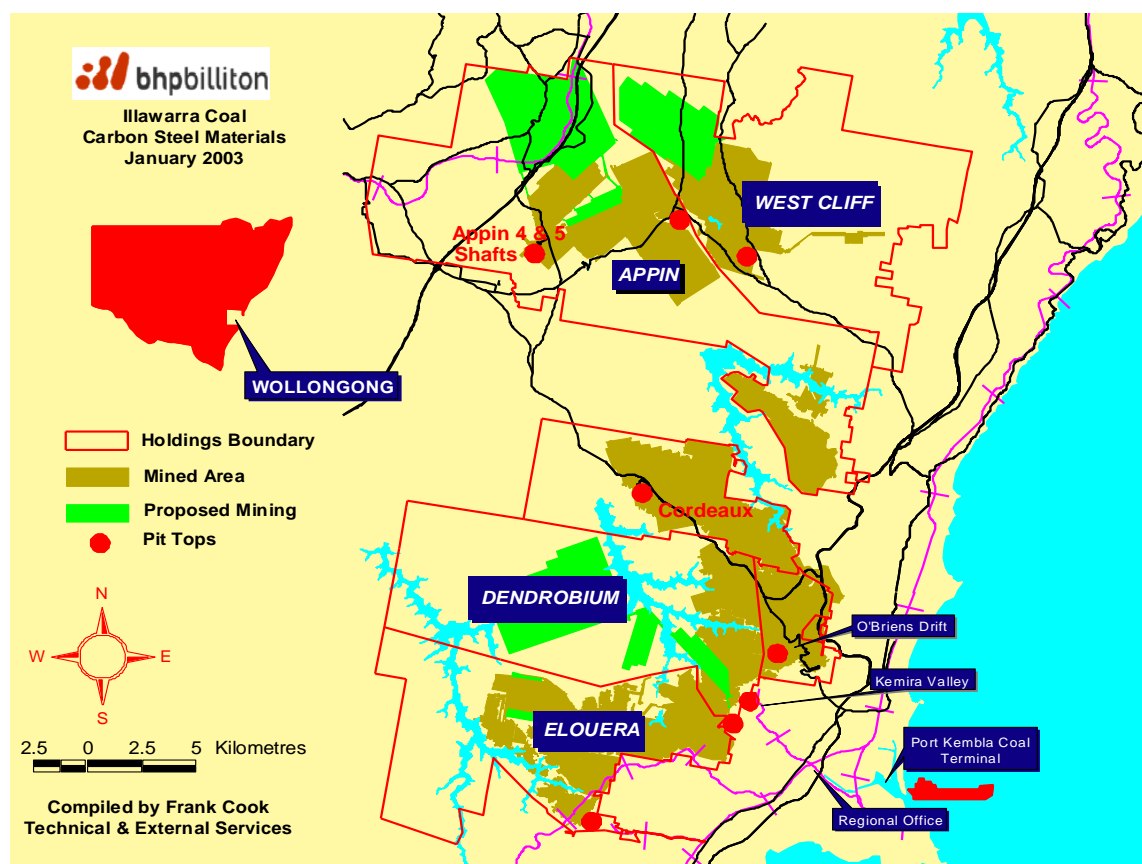


Fig 1 – BHP Billiton Illawarra coal operations including lease boundaries, current and proposed mining areas, and previously mined areas

As with many businesses, Illawarra Coal has recognised that future success is contingent upon achieving sustainable outcomes, including sustainability in environmental and community terms. These requirements represent one of the greatest challenges that lie ahead for the success of the Illawarra Coal business.

In 2002, Illawarra Coal commenced an extensive stakeholder consultation programme to identify stakeholder issues related to its underground mining operations. The program was facilitated by Coakes Consulting, who are social management specialists. It included individual interviews with over 100 stakeholders, including state and local government agencies, local residents, environmental and community organisations, indigenous groups and local businesses. In addition, a telephone survey of 1400 households, randomly selected from across the Wollondilly and Wollongong Local Government Areas has also been undertaken to assess community attitudes towards mining.

A key environmental issue facing the business is that mining under rivers and other natural features can cause fracturing of stream-beds, redirection of water to shallow sub-strata, water quality impacts and methane gas emissions from the strata to the atmosphere.

One of the key stakeholder themes identified in the consultation program was the need for 'sensitive mine planning', particularly in relation to mining under rivers and other natural features in sensitive areas.

INTEGRATING ENVIRONMENTAL ASSESSMENTS INTO MINE PLANNING

In the past, mine planning has considered potential effects on engineered structures, however, the same attention has not been paid to the effects of mining on natural features. Through the Stakeholder Involvement Programme conducted by Illawarra Coal, it was identified that government and community stakeholders were seeking a more sensitive approach be taken to mine planning in significant and high risks areas. As a result, mine planning processes are being reviewed to take into account the effects of mining on surface features.

The development and implementation of a new IMP was identified as a key strategy to address stakeholder concerns such as mining under rivers, and has been developed as a planning process to identify and manage subsidence effects on natural and constructed surface features.

The IMPP provides a systematic and ongoing approach to incorporating environmental assessments as part of mine planning. It is an iterative process enabling key events and business challenges to be identified and dealt with in a timely and systematic manner. A primary input into the process is the BHP Billiton Charter and Health Safety Environment and Community (HSEC) Policies. The business principles developed in these documents cascade through the Carbon Steel Materials planning strategy and into the Illawarra Coal business plan.

The Illawarra Coal planning cycle shown in Figure 2 supports and optimises the business plan. There are a number of inputs into the process, which is iterative and normally spans a period of 12 months or more.

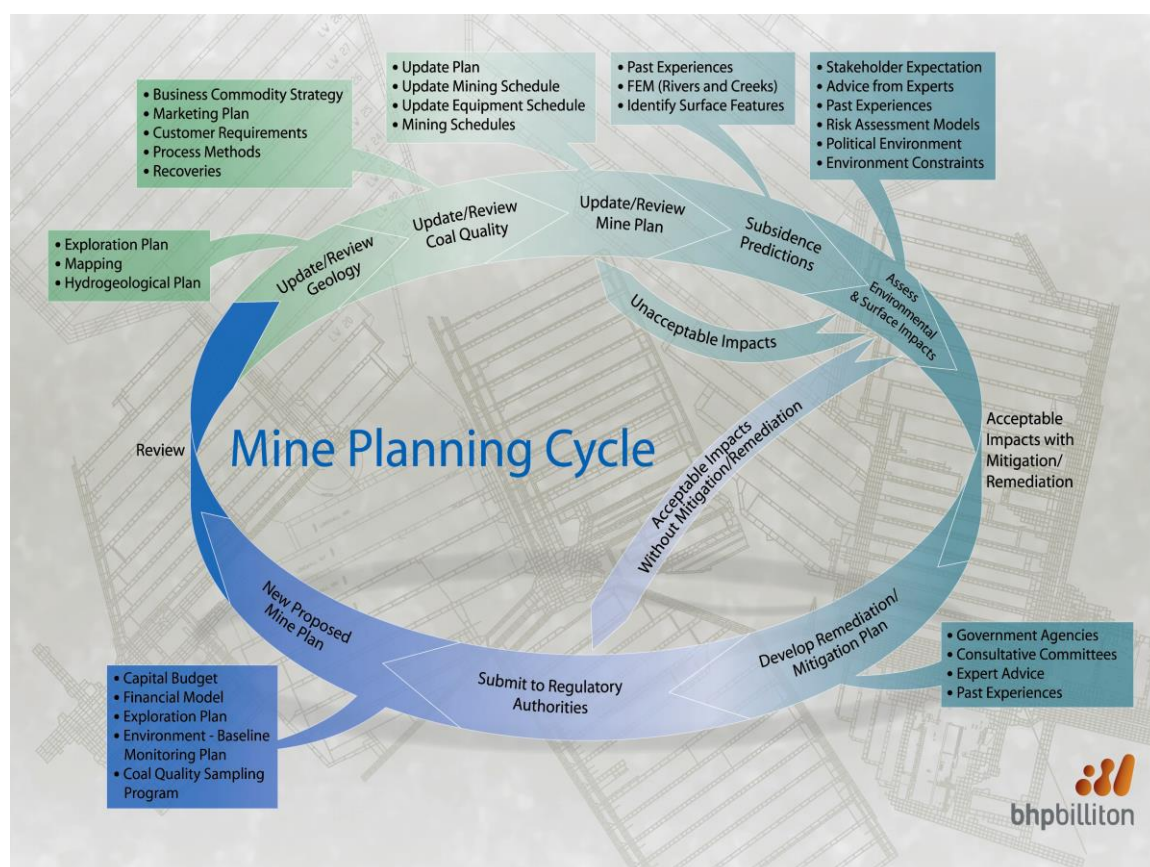


Fig 2 - Illawarra Coal Mine Planning Cycle

Impacts on natural features are addressed in the planning cycle through its assessment of environmental and surface effects. The key elements of the planning cycle include:

- continuing stakeholder consultation and participation;
- comprehensive baseline environmental assessment;
- consideration of environmental impacts and mitigation measures during the assessment of alternative mine plan options;
- consideration of monitoring results from past mining activities; and
- monitoring and stakeholder reporting programs.

In order to build an approach with the ownership of both internal and external stakeholders, the development of the IMPP process has involved both internal and external consultation. The process followed for the development of the IMPP is illustrated in Figure 3.

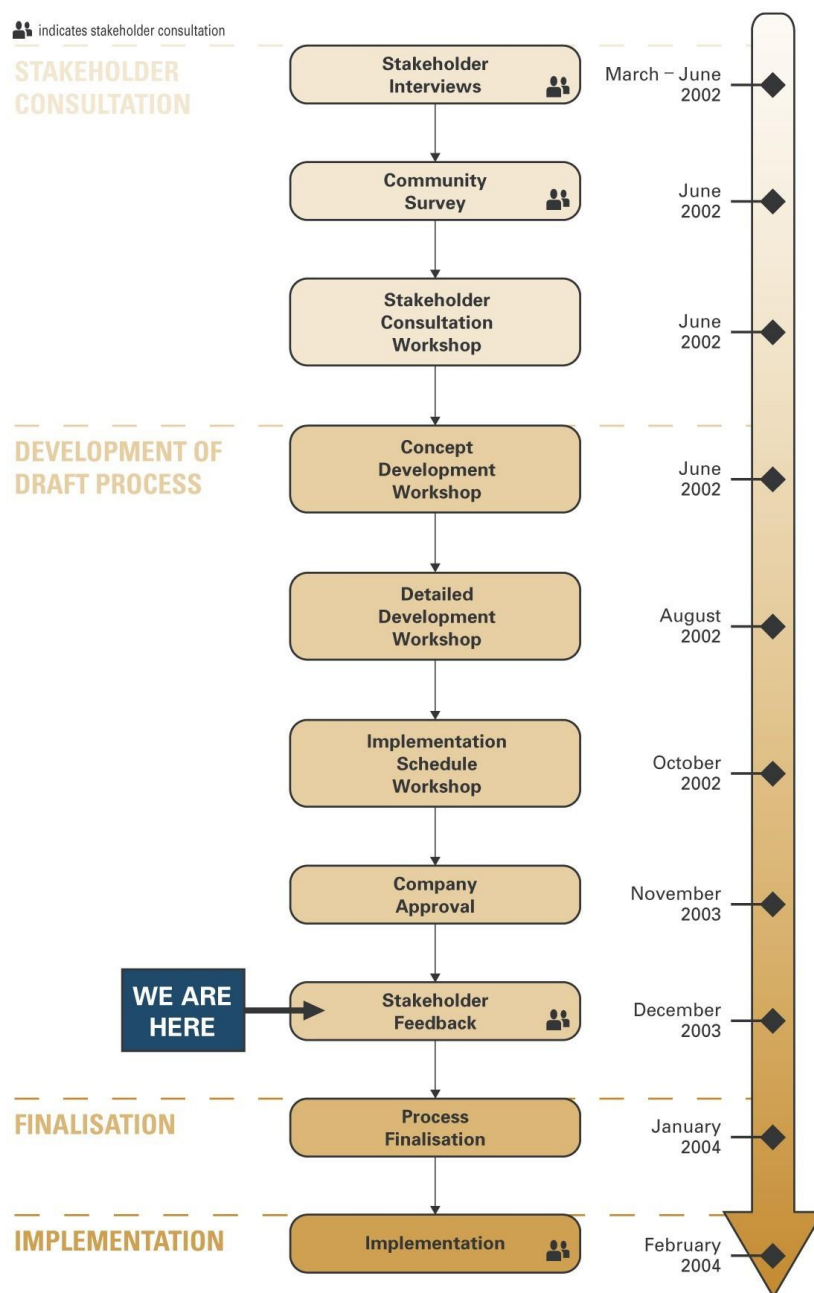


Fig 3 - Process for development of IMPP

The external community and government consultation phase conducted in 2002 was followed by a series of company workshops and meetings. From this, a draft, integrated mine planning process was developed with key planning and operations personnel. Development of the process then involved company approval and internal communication to the broader planning and operations staff within Illawarra Coal. Consultation is now underway with external stakeholders, including government, environmental groups, and community groups. The IMPP will then be finalised taking into account stakeholder feedback, and then implemented at Illawarra Coal mining operations.

The Illawarra Coal mine planning process is being developed to be consistent with the Department of Mineral Resources (DMR) SMP process. The SMP will replace the environmental assessment and approval components of the current approval process. The new DMR process will be implemented from March 2004.

TIMING OF MINE PLANNING

Mining of coal by longwall methods involves considerable expenditure and lead time (up to 5 years) for the development of access roadways before longwall coal extraction can commence. The cost of longwall development is high and needs to be done well in advance of mining to ensure continuity of extraction. This continuity is critical to the success of a longwall mining business. Without continuity, lost production, cost-impacts, loss of jobs and closure of mines would result. This reinforces the need for suitable long-term planning to ensure that timing requirements can be met. The IMPP process considers these factors.

Overview of the IMPP

The IMPP involves a holistic approach to the development of mine plans for the total minable resource area, rather than simply a plan for a set of mining panels associated with the next longwall mining approval. This approach ensures a greater level of awareness of issues, an ability to plan and implement mitigating strategies, and minimises business risk to the company associated with potential changes to mine plans.

The IMPP is designed to integrate stakeholder engagement and environmental impact assessment into the mine planning process. This will enable future mine plans to be developed on the balanced consideration of all relevant factors including stakeholder expectations, environmental impact, geology, resource utilisation, operational constraints and economic feasibility.

For the purpose of sensitivity assessment, surface features are grouped into three categories being natural features, infrastructure and private properties. The natural features that tend to be most sensitive to mining induced subsidence include rivers, creeks, wetlands or swamps, and cliff lines.

STEPS IN THE PROCESS

The process of developing the IMPP involves five steps which are illustrated in Figure 4 and are discussed in detail.

Step 1 - Preliminary Sensitivity Assessment

Step 1 is the initial assessment of options for mine planning. It includes a review of geological information, mine layout, development requirements and access to the coal resource.

It also includes the preliminary assessment of the sensitivity of the surface features to underground mining. Illawarra Coal has a substantial database of information in relation to surface features, subsidence impacts, mitigation measures, and stakeholder expectations. The preliminary sensitivity assessment is designed to provide an initial focus for mine planning based on existing information. This enables the consideration of alternative mine planning options and/or mitigation measures at an early stage.

Step 2 - Preliminary Mine Planning Assessment

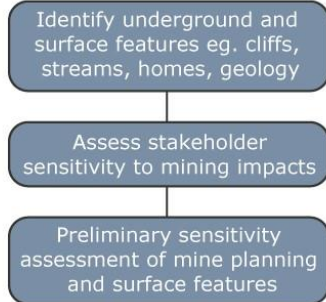
This step involves the preliminary evaluation of alternative mine plans to determine the preferred mine plan/s. The alternative mine plans are developed utilising proven mitigation or remediation of subsidence impacts on sensitive features and/or avoidance of sensitive surface features identified in Step 1.

The evaluation of alternatives is conducted by a multi-disciplinary mine planning team and involves a balanced consideration of economic, environmental and social issues. The multi-disciplinary team includes representatives from a number of internal groups including exploration, mine planning, mine infrastructure, environment, and community. The preferred mine plan or plans provide an initial focus for the detailed sensitivity assessment of surface features conducted in Step 3.

 indicates stakeholder consultation

STEP 1

PRELIMINARY SENSITIVITY ASSESSMENT OF MINING AREAS



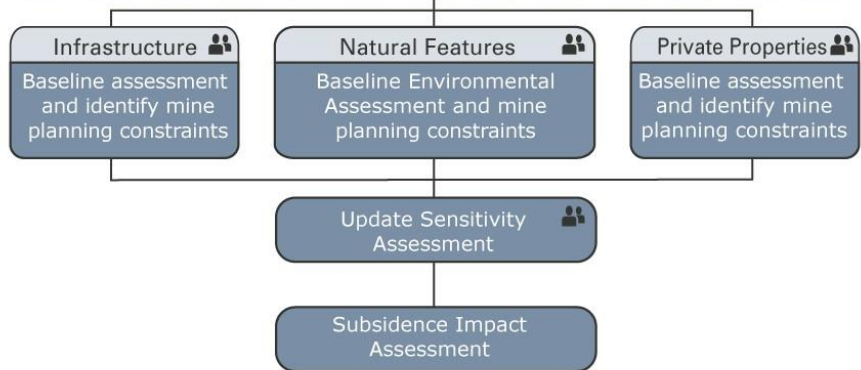
STEP 2

PRELIMINARY MINE PLANNING ASSESSMENT



STEP 3

DETAILED SENSITIVITY ASSESSMENT



STEP 4

DETAILED MINE PLANNING



STEP 5

MINING APPROVAL APPLICATION

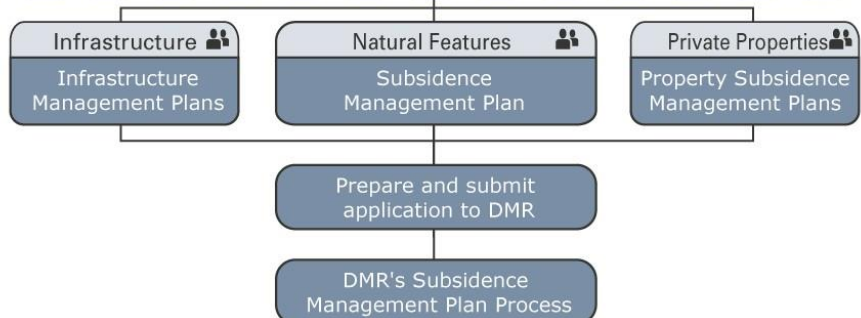


Fig 4 - Steps of the Integrated Mine Planning Process

Step 3 - Detailed Sensitivity Assessment

Step 3 involves a detailed sensitivity assessment of surface features, and mining constraints. It includes the collection of baseline data on surface features, subsidence impacts and mitigation measures, and seeks to identify mine planning constraints.

At this stage, the baseline assessment is undertaken in consultation with relevant stakeholders, leading to the revision of the sensitivity assessment from Step 1. These stakeholders include government, environmental groups, and community representatives. A subsidence impact assessment will be undertaken for natural features that will potentially be undermined, and mitigation options will be identified for significant features. Ongoing monitoring and assessment requirements will also be identified during this step.

Step 4 - Detailed Mine Planning

This step involves a detailed re-evaluation of alternative mine plans based on the results of Step 3. This review is conducted by the multidisciplinary mine planning team, in a similar process to Step 2, and involves a balanced consideration of economic, environmental and social issues. Step 4 results in the selection of a final preferred mine plan and development of preferred mitigation measures.

Step 5 - Preparation of the Mining Approval Application

This step involves the preparation of an SMP to support the subsequent longwall mining approval application. The SMP will include the impact assessments and proposed mitigation measures for natural features prepared in consultation with relevant government and community stakeholders. Step 5 will be repeated for each longwall mining application required over the life of the mining operation.

The DMR's processing of the SMP involves additional community consultations including public advertisements and access to the SMP.

HOW THE PROCESSING IS BEING IMPLEMENTED

Illawarra Coal has developed centralised mine planning teams incorporating expertise from the relevant mine, and representatives from exploration, environment, community, and long term mine planning areas to implement the integrated mine planning process. The objective of these teams is to ensure consistency of approach with the IMPP across the company's operations and to manage the development of the mine plan and the approval requirements. The teams are currently implementing the process for the development of new mining areas at both Appin and West Cliff mines.

CONCLUSION

The future of mining in the Illawarra region will be dependent on adequately addressing sustainability issues. This paper has shown BHP Billiton-Illawarra Coal's approach to incorporating environmental aspects into mine planning.

This has involved internal workshops and projects, together with external consultation. The paper describes the process that the company is utilising to fully incorporate environmental assessments into the mine planning process.

The purpose of this process is to identify sensitive surface features, mitigation and remediation options, and potential avoidance measures early in the mine planning process. Another key aspect of the process is to incorporate internal and external stakeholder feedback as part of the mine planning process.

The IMPP now being utilised within Illawarra Coal business is the process that has been developed to incorporate environmental assessments into mine planning.

It is anticipated this will lead to more secure outcome for the business. This is expected to occur through understanding and addressing environmental issues and incorporating stakeholder feedback early in the mine planning process, therefore minimizing the risk of changes to mine plans being required within short time

frames. The thorough and rigorous process will give the business confidence that the mine plan submitted through the SMP application to government has the highest probability of approval. We are confident that this process will provide the most sustainable outcome – for the environment, community, and the business.

ACKNOWLEDGEMENTS

Managers, exploration staff, mine planners and other representatives from the Illawarra Coal business are acknowledged for their contribution and willingness to embrace the significant change that is occurring to fully incorporate environmental assessments as part of our mine planning processes.

There has been considerable input from Coakes Consulting into the development of this process, including facilitation of the stakeholder involvement programme, facilitation of workshops, contribution of ideas towards modifying our mine planning processes, and documentation, much of which has been utilised in the preparation of this paper.

The role of internal managers and the expertise that Coakes Consulting have brought have both been crucial to the success of the IMPP within Illawarra Coal.

EXPLORATION FOR RESULTS MOURA COAL MINE

John Hoelle¹

ABSTRACT: Moura Coal Mine operates 3 draglines, 2 excavators, and 2 highwall mining systems producing 6.5 million tons per annum for 3 products from 7 different seams and operates a commercial seam gas recovery operation - with active mining spread over 40 kilometers of strike length and dips up to 20-degrees. To minimize the impact of geological-geotechnical problems, the exploration program has to be thorough and comprehensive since the data may be used for several different types of operations.

For open cut mining, strength data is obtained on overburden and floor strata for low wall stability design. Strength information of the overburden strata and fracture data are used in the design of pre-split and overburden shots and highwall stability analysis. Successful highwall mining requires geotechnical data for analysis for roof, pillar and floor stability analysis. Once mining commences the feedback on the response of strata to mining is hazy; adequate information is required to lessen the possibility of unpleasant surprises. The impact of seam gas content on highwall mining production is analyzed using routine seam gas sampling. Samples of coal seams are obtained routinely for seam gas data for the seam gas operation and highwall mining.

INTRODUCTION

The Moura Coal Mine is located on the east flank of the Bowen Basin. At present, all mining is open cut mining with highwall mining being conducted after economical limits of the pits are reached. The active mining is 40 kilometres long with reserves covering 150-km of strike length. The width of active mining is up to 3 kilometres. Mining is complicated by the range in geologic conditions in strata types and the steep dips (up to 20 degrees).

The exploration programs attempt to obtain data to satisfy the design requirements of open cut, highwall mining and commercial seam gas recovery operations. At times, the requirements are mutually exclusive and choices have to be made. The exploration programs at Moura Coal Mine are similar to other mines in the type of data that is obtained. Table 1 lists the information that is routinely obtained from a core hole.

Two examples of typical exploration dilemmas will be presented – one in which a choice of what data to collect had to be made and one in which a large quantity of data had to be obtained in order to successfully start a new pit.

RESERVE WITH HIGH QUANTITIES OF SEAM GAS

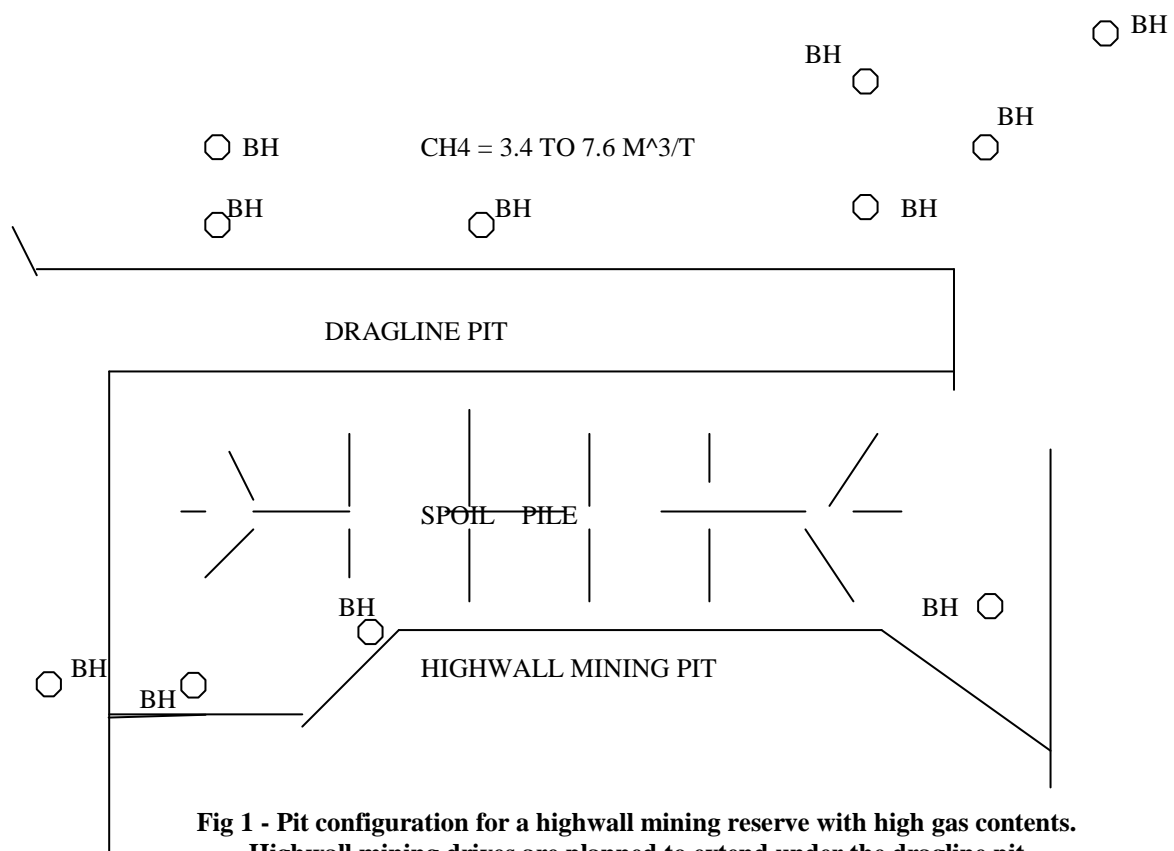
An area located down dip from an open cut and highwall mining reserve had previously been identified as a potential seam gas recovery reserve based upon gas desorption tests obtained from a number of boreholes. The high levels of methane gas (up to 7.6 cubic metres per tonne) down dip also indicated that these gas quantities may impact on the proposed highwall mining. Coal quality samples from this same area indicated that the coal could be a semi-soft coking coal. A proposal was made to re-open an open-cut previously mined over 25 years ago for an additional cut on the highwall and then conduct highwall mining.

Additional boreholes were required to delineate both the gas quantities as well as the coal qualities in the area of the proposed mining. However, the area between the “known” reserve down dip and the proposed open cut pit was an active dragline pit in an overlying seam and unavailable for exploration. The area under consideration is shown in Figure 1.

¹ Mining Engineer Moura Coal Mine

Table 1 - Information usually obtained from boreholes in an exploration program

- Geologic Data
 - Coal thickness
 - Samples for coal quality
 - Geological features
 - Strata type
 - Structure
 - Dip
- Geophysical electric logging
 - Density
 - Natural gamma
 - Calliper
 - Sonic
 - Dipmeter
- Geotechnical Data
 - Sample for laboratory testing
 - Field logging
 - Estimated field strength
 - Point Load Tests (diametric and axial)
 - Rock Quality Designation (RQD)
 - Fracture Frequency Index (FFI)
 - Joint Roughness Coefficient (JRC)
- Seam gas quantity



**Fig 1 - Pit configuration for a highwall mining reserve with high gas contents.
Highwall mining drives are planned to extend under the dragline pit**

This active pit left a limited area for 4 boreholes and these boreholes would have to be placed within 50 to 70 metres of the highwall. The coal quality characteristics were critical to the viability of the proposed pits. Although the quantity of the seam gas was required, historically at Moura, seam gas samples obtained from coal seams with less than 50 metres of overburden usually showed low gas contents. Coal samples obtained for seam gas content cannot be used for testing for coking coal properties since the time delay and the oxidation of the seam during the desorption process significantly alter the properties of the coal. Samples that have been tested for seam gas are only tested for raw coal quality parameters.

Therefore the decision was made to obtain samples for coal quality. The experienced field geologists were to determine if copious quantities of gas were desorbed from the core whilst logging. The reserve was determined to be of a coking coal quality.

There are limits to mining in methane, even though the steep dip system uses inert gas forced into the drive to reduce the quantity of oxygen and prevent the occurrence of an explosive atmosphere. Prior to mining, it was estimated that methane levels that would be encountered would be in the range from 4 to 6% and that these levels would be encountered approximately 250-metres into a drive. In practice, high levels of methane were encountered approximately 40-metres into the drives and the levels have range up to 11%, a level that has to some extent adversely affected production.

DESIGN OF A BOXCUT IN ADVERSE CONDITIONS

Moura Coal Mine is on the edge of the Bowen Basin and the dragline pits start on the crop line with a box cut. Box cuts are notorious for potential stability problems so that the strength characteristics of the various strata (immediate floor, coal, immediate roof, weathered coal, weathered roof strata and soil) are all required to properly design a stable pit.

In starting a new pit, one important issue that has to be addressed is the exact starting point of the initial cut. Too far up-dip and weak strata is encountered and unsaleable soot is recovered. Too far down-dip and saleable coal is sterilised at low overburden ratios. The limit of oxidation (Lox) line is invariably not a straight line; whilst a straight line is a requirement for the dragline.

The parameters for this box cut consist of:

- A thick coal seam (about 5 to 5.5-metres thick);
- Steep dipping (14-degrees, which is moderate for Moura);
- Weak roof strata in the box cut area;
- Moderately strong floor;
- A wet seam;
- Faulting that cuts across the crop creating offsets in the crop.

Previous studies have recommended one of two design parameters for the boxcut depending on the clay content of the roof above the coal. If there are no clay seams in the roof strata, then a 20-metre berm should be left between the crest of the low wall and the toe of the spoil. If clay seams were present, then the toe of the spoil should be moved further from the projected crop of the coal seam, creating an approximate 70-metre wide berm as indicated in Figure 2. Other alternatives have also been suggested, such as a false box-cut or removal of the wedge of strata above the coal.

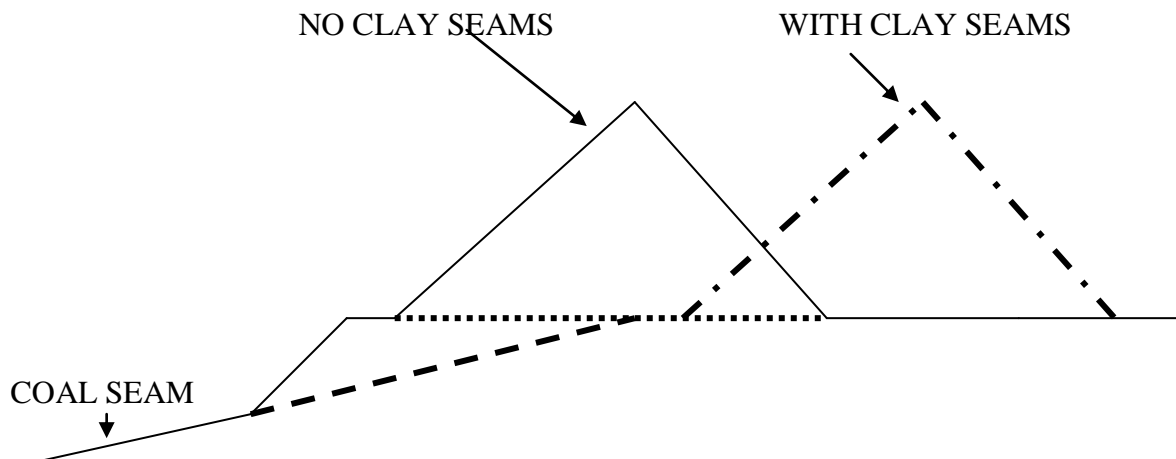


Fig 2 - Configuration comparing the two different spoil pile locations depending on the strength of the strata overlying the coals seam.

The exploration program required answers to the following questions:

- Cut-off limits for LOX line
- Minimum thickness of coal seam in which to start mining
- Coal quality
- Good definition of the coal location horizontally and vertically
- Location of the several faults
- Strength of the in-situ material above the coal
- Coal strength
- Partings within the coal
- Fracture orientations

A large number of chip holes (91) were drilled in rows across the anticipated LOX line. Whilst the main reason was to define the location of the coal and to determine the LOX line, sonic geophysics was used to obtain strength characteristics as well as an indication of the degree of weathering of the coal. The proposed pit is slightly over 2000 metres long and the main objective of this exploration program was to determine the limits of the pit.

A smaller number of cored boreholes were drilled in order to obtain samples of the strata above the coal seam for geotechnical testing and to obtain field geotechnical information.

The exploration drilling determined that clay seams were present in the area of the proposed box cut. In addition, the strata above the coal was highly fractured with numerous high angle fractures and shear zones that approached gravel in consistency. Due to the weathered and fractured nature of the samples, lab testing was difficult. Only 5 of 9 samples obtained during the exploration process were suitable for testing. Direct shear tests were conducted by a soils laboratory. These direct shear tests produced the following strength values.

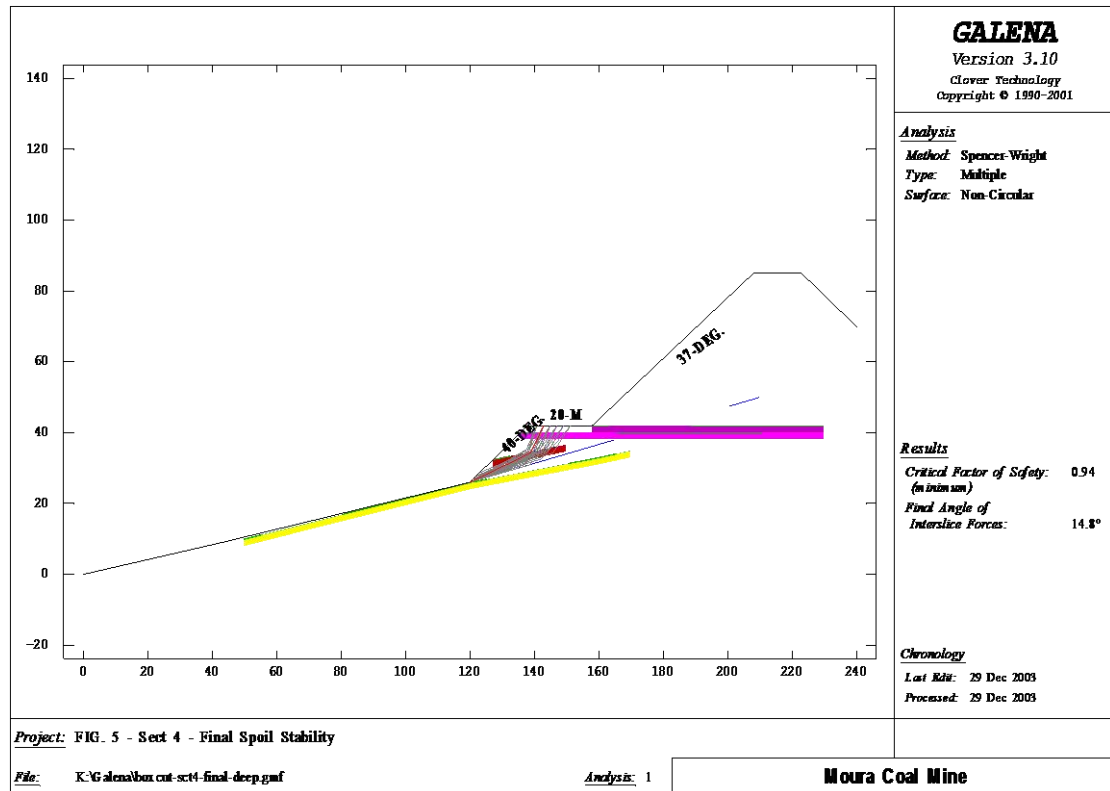
Cohesion of 20 to 30 KPa

Internal friction angles of 17 to 28 degrees

Based upon the lab test results as well as estimates of the strata strength obtained during the logging process, stability analyses were conducted using the software slope stability program Galena. The final design produced a boxcut with a 40-degree low wall slope, a 20-metre wide berm and a spoil pile with nominal 37-degree slopes. Shallow failures in the low wall were anticipated; however, these projected slips did not affect the stability of the spoil (which had a Stability Factor = 1.20). Refer to Figure 3.

The first pit and over half of the second pit has been completed and the anticipated movement in the low wall did not occur. There were just 3 areas where very limited shallow slips occurred, none of which encroached on

the berm. The success of the box cut was created by the quality and quantity of the basic geological and geotechnical data obtained during the exploration program.



**Fig 3 - Example of slope stability analysis indicating shallow failure in low wall.
Analysis of spoil stability after removal of “failed” low wall produced a stable spoil pile
with a Stability Factor of 1.20**

CONCLUSION

Any exploration program is always less than is desired. The object is to strive to obtain as much information as possible from each core hole and borehole. At times, this is not always possible. The two examples presented indicate some of the problems created in making choices.

GETTING USEFUL GAS AND STRESS MEASUREMENTS OUT OF EXPLORATION DRILLING

Ian Gray ¹

INTRODUCTION

This paper focuses on two areas, gas and stress. It discusses the measurement of relevant parameters for mine design and aims to assist operating mines keep abreast of changing conditions to avoid problems. An understanding of the factors which affect the reservoir capacity of a coal seam is essential as an aid to better mine planning.

DESCRIPTION OF A COAL SEAM RESERVOIR

Coal seams are frequently gas reservoirs and may pose significant problems for mining. It is important to establish the behaviour of this reservoir as part of the exploration process.

The gas in coal seams is stored in the seams by a process of sorption, which involves aspects of surface bonding and chemisorption. The gas stored in pore space is usually far less than that stored in the coal itself. The coal matrix changes dimension depending on the level of moisture and gas in the coal. This feature is of extreme importance.

Most coal seams exist initially in a state where all the pore space is filled with water, although exceptions where gas caps exist can be found. In these latter instances, fluid production may start with gas. In the more usual case, gas production will not occur until the water pressure has been lowered to less than the sorption pressure (akin to a bubble point in an oil reservoir).

Once the pressure in the cleats in the coal has been lowered, desorption of gas from the coal may occur. The process of gas movement in the coal solid is considered to be one of diffusion. This occurs from the solid to the fracture space where the pressure is lowered and bubbles form in the fractures.

The flow within the fractures in the coal follows Darcy's law of flow down a potential gradient (pressure and gravitational components). The presence of gas and water within the fractures leads to a two-phase flow regime, in which water impedes the movement of gas and vice versa. Coal seams frequently display several forms of fracturing. These can be divided into microfractures, cleats, major joints and faults. All are important in the behaviour of coal as a reservoir. Flow from coal is a process of diffusive flow into microfractures to the cleats and from cleats to any major joints or faults that may exist. The fractures all have direction and contribute to a directional permeability dependent on their orientation. The cleated coal may be considered to represent, in oilfield terms, the matrix permeability, whilst the major fractures represent fracture permeability.

The permeability of coal is generally directly related to the effective stress to which the coal is subjected. Effective stress is the total stress field (a tensor) minus the fluid pressure. Permeability is thus a tensor dependent on stress and fracture orientation.

The effect of stress on coal permeability may be described in non-directional terms by Equation 1 below:

$$k = k_i 10^{\left(\frac{\sigma_i - \sigma}{\alpha}\right)} \quad (1)$$

¹ Principal, Sibra Pty Ltd

where k is the permeability at effective stress σ ,
 k_i is the permeability at initial effective stress σ_i ,
 α is the effective stress change required to produce a ten-fold reduction in permeability,
 σ_i is the initial effective stress, and
 σ is the effective stress.

In coals, α may range from a lower value of about 2 MPa up to high values. The higher the value of α , the less permeability is influenced by changing effective stress. The softer the coal, the lower α tends to be. If α is of similar or lower value than the expected drop in reservoir pressure, then stress related permeability changes may be expected to be extremely important.

As reservoir pressure drops, the effective stress within the coal can be expected to increase and with it an associated permeability reduction. This behaviour does occur, particularly in the short term. In the longer term, an opposing effect may occur due to shrinkage. If the coal shrinks as it gives up gas and then dries out, the coal carries less stress and transference of stress to the surrounding rocks occurs. The condition may occur where no stress exists between cleats, in which case the permeability may increase sharply.

The process of fluid movement in coal may be interrupted by a producing borehole or by an absence or blockage of any of the levels of fracturing. Gas production may be thought of as having several contributing steps.

If the diffusion coefficient is low or the spacing between fractures too great then, despite an apparently high permeability, the gas release may be limited by the diffusive escape of gas from the solid coal. Similarly, if the matrix permeability is too low then gas production will be impeded by this. The presence of major fractures may lead to permeability that is an order of magnitude different to that of the matrix. However, high fracture permeability is of little benefit if the matrix is blocked and diffusion is the only mechanism by which gas may reach the fractures.

Faults often act as boundaries between areas of different reservoir characteristics.

DETERMINATION OF RESERVOIR PROPERTIES

Determining reservoir properties is extremely important to the viability of a mine. Normal mining exploration should include the correct procedures to ensure that the correct reservoir parameters are obtained. In addition to measuring gas content, an endeavour should be made to determine diffusion coefficients and permeabilities. The plural is used here deliberately, as coal usually exhibits at least two diffusion coefficients and permeabilities to match the varying fracture types.

Gas Content and Diffusion Coefficient

Coring is a good start to assessing the gas production characteristic of a coal seam gas reservoir. It enables a sample of the coal to be obtained and provides a hole in which to conduct reservoir tests. This hole may then be used as an entry to the reservoir for monitoring.

The coal core can be taken and placed in a desorption vessel so that the gas released can be measured in volume and type. The use of strain gauges to measure the change in dimension of the coal is recommended. If the coal core is of regular cylindrical form it is theoretically possible to arrive at diffusion coefficients. This is a result of fitting diffusion equations to the short and long term release of gas from the core. Unfortunately very few coal cores are neat cylinders, more typically they are fractured and of variable composition.

Despite this, the same equations that are used to describe the diffusion from a cylinder also fit the rates of desorption extremely well. The real issue is then of examining the core carefully and making an estimate of what sort of characteristic dimension applies to the coal lumps.

Gas is invariably lost on core recovery and it is usual to estimate the amount by measuring the cumulative gas release with time as soon as the core can be placed within a canister. A plot of the square root of time since removal of the core and cumulative gas release usually plots as a straight line as could be expected from the equations relating to short-term diffusion. This can be extrapolated back to zero time to arrive at a lost gas estimate. The slope of the curve may be directly related to the initial diffusion coefficient.

The fitting of the diffusion equation to the longer term desorption characteristic usually leads to a good fit, but invariably the associated diffusion coefficient is significantly lower than the one arrived at by examining the initial gas desorption characteristic.

The process of diffusion is therefore open to question. Why two diffusion coefficients? An explanation may be found in the fact that initially significant pressure of gas in the micro-cleats could be enlarging these and expediting the release of gas. Alternatively, there could simply be two or possibly more diffusion behaviours. If the long-term diffusion coefficient can be estimated, however roughly, then for that number to be of value, it must be associated with an open fracture spacing. From these two numbers the rate of diffusion can be calculated.

It is therefore extremely important to have some idea of the cleat spacing in the coal. Unless the cleats are extremely closely spaced, and their separation can be measured within core, the estimation of cleat spacing is unfortunately quite difficult.

Permeability Measurement

Measuring a coal reservoir's permeability is frequently challenging because of (i) stress-dependent permeability displayed by coal; (ii) shrinkage which affects stress; and (iii) two-phase effects. Additionally, the fracture permeability is frequently significantly greater than the matrix permeability. What, then, is the best way to proceed given these factors?

The first and most simple test for permeability is to get a piece of core and to look at it. If no fractures exist, and particularly if the coal has a waxy feel to it, then it is likely to be tight on the scale of core. If fractures are visible, but filled, then a similar comment applies. If the core contains cleats that are free of infill, then the prospects are much better for gas production. This does not necessarily mean that it will be permeable, because cleated coal can be very tight under high stress conditions. This applies particularly if the coal is soft.

The next stage of assessing the core for permeability is to pick it up and suck through it. This may seem amusing, but it is remarkably effective in allowing one to gain some feeling for the nature of the fractures and their ability to carry gas. An inability to suck air through the core at zero stress implies a much bigger problem for passing gas under stress.

On a more formal note, it is most important in assessing the permeability of coal to realise that any measurement must be put in the context of the state of development of the reservoir and that a permeability measurement made at a specific time may be quite different from one that exists later. It is equally important to be able to predict whether the coal seams' permeability will increase or decrease. Self-sealing coals do exist, as do instances of permeability enhancement of two orders of magnitude.

Given the complications associated with stress permeability relationships, it is generally not wise to introduce two-phase effects (flow of gas and water) during initial reservoir testing. Picking the stress related permeability effects is quite complex enough. The use of a drill stem test (DST), which may occasionally produce a minor amount of gas, followed by an injection fall-off test is advocated.

A procedure sometimes used is to core a seam and to pull back the core rods to above the seam. This is followed by running a twin packer DST tool through the drill string and displace water out of the string using compressed air. Packers are then set, one below and one in the string, to release compressed air. The bottom valve can then be opened by lowering the string and inducing flow into the rods. The gross flow rate of gas and water is measurable on a surface gas flowmeter. The volume of water in-flowing may be measured by means of a head increase in the rods above the DST tool. The bottom valve is then closed and pressure builds up and is measured by a bottom hole pressure transducer. During the build-up period, the rods are filled with water. An injection test at a constant rate is then performed and the hole is then shut in while pressure approaches equilibrium.

The exact mechanism of the test may be varied if DST tool in which the packers are inflated off the rig pump is used. Essentially the test remains the same. A trace of bottom hole pressure from a DST and injection fall-off test is presented in Figure 1. The tool used is shown in Figures 2a and 2b.

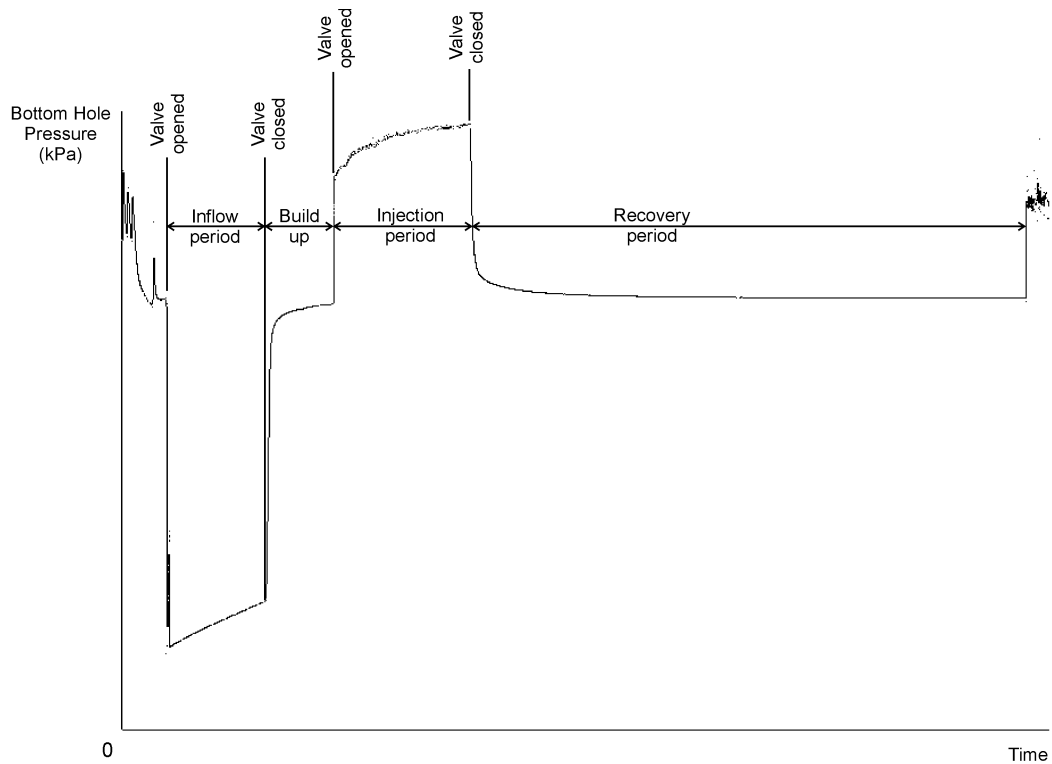


Fig 1 - DST and injection fall-off trace.



Fig 2a - A DST tool being lifted on a wire-line.



Fig 2b - Stress measurement tool about to be lowered into HQ drill rods.

The DST flow period may last from several hours to a few minutes, depending on formation characteristics. The same applies to the recovery period. The DST test gives an indication of the injection flow to follow. The injection test is usually conducted at a single rate, though in some instances it is useful to vary this to see what the short-term local response is to rate changes.

The analysis of DST or injection fall-off tests in coal may be simple or complex, depending on the seam. If the coal seam is permeable, the permeability not particularly stress-dependent or the sorption pressure low, then a normal Horner-type analysis is usually sufficient for both the build-up and fall-off behaviours. If, however, the seam deviates from these characteristics, then the assessment of seam parameters becomes significantly more complex. The analysis essentially becomes one where a simulator is required that incorporates all the complex characteristics of coal seam reservoir behaviour.

Not all assessments of permeability need be so complex that they require a simulator to find a solution. Some characteristics may be found by basic calculation. One of the first comparisons to look at is the inflow rate during a DST test and the injection rate. It is important if the flow rates are significantly different for similar variations from reservoir pressure. If the DST inflow rate is significantly higher than the injection rate then in all probability well-bore damage (plugging of the fractures surrounding the borehole) has occurred. This will be revealed in the nature of the pressure change after well closure, with instant pressure changes indicating well-bore damage.

When a DST inflow rate is significantly lower than the injection rate there is a good indication that stress-related permeability effects are important. The reduction in fluid pressure around the well-bore during a DST test leads to increased effective stress and in a coal with stress-dependent permeability, a reduction in permeability. As the pressure drop extends from the well-bore, this effect spreads and cannot be lumped into well-bore effects represented by a single numerical value representing skin. The opposite effect occurs on injection with effective stress being reduced progressively from the well-bore outwards.

Curvilinear Horner build-up or draw-down plots are one of the characteristics of stress related permeability effects. Unfortunately, there are other causes of curved plots such as well-bore storage caused by packer movement and fractures. Numerical solutions for these effects can be found by the use of a simulator in which the primary unknowns are the in-situ permeability at reservoir pressure and the value of α (the effective stress change required to produce a ten-fold reduction in permeability).

Many well tests show more than one linear portion. This may be due to a matrix permeability that is different from the fracture permeability. Extending a test interval will help to confirm the real reservoir permeability. Alternatively, an interference test may be considered. Interference tests involve pressure monitoring remote from what is usually an injection well. Their advantages are three fold:

1. Because they are remote they do not suffer from near well-bore effects associated with skin or local effective stress changes around the well-bore;
2. If more than three observation points are used they permit the measurement of directional permeability; and
3. In a single-phase situation they permit the estimation of reservoir storage parameters.

On the down side interference tests are usually expensive to conduct.

A happy alternative that can more normally be accommodated is to fit exploration core holes with pressure transducers and to use these to monitor the reservoir during drainage. Such transducers are far more effective than compliance cores for gas content because they provide a continuous record of pressure (and hence gas content) rather than a single expensive sample.

Stress Measurement

An economic initial approach to finding directional permeability is to measure stress. Normally the direction of maximum permeability is perpendicular to the direction of minimum principal stress. So too, frequently, is the direction of the principal cleat. Thus, if the magnitude and orientation of the stress field can be determined, then the likely directions of principal permeability can also be determined.

Coal is frequently a cleated, weak material that makes the determination of stress impossible by overcoring, and virtually impossible by either hydrofracture or borehole breakout techniques. The rocks above and below the coal seam are, however, much more amenable to having stress measurements carried out on them. If no borehole breakage in these rocks occurs, then overcoring is undoubtedly preferable to hydrofracture techniques where the value of stress is sought. Where only direction is sought, multiple hydrofracture measurements may yield an adequate measurement of stress direction. Where borehole wall failure has occurred, scanners may be used to determine the orientation of the breakout.

The measurements used to determine stress by either hydrofracture or by assessment of borehole breakout are subject to more interpretation error than measurements accomplished by the overcore process. In addition, the mathematics to interpret the measurements is significantly less precise than those used to derive stress levels from overcore measurements. Stress is a tensor and the use of scalar measurements (pressure) to determine components of the tensor has limitations.

Converting rock stresses to coal stresses may be undertaken theoretically based on knowledge of the Young's modulus and Poisson's ratio of the coal. The limitations to this approach are the difficulty in measuring these parameters on many coals and the question of how creep and shrinkage have affected coal stress. An estimate of the magnitude of principal stresses in a coal seam is, however, useful for estimating directional permeability.

Monitoring the coal seam reservoir during drainage is extremely important. This monitoring should consist of three elements. The first two involve the measurement of gas flow and water flow from boreholes. Knowledge of one without the other is incomplete. Monitoring of water production is the key to knowing whether the reservoir is being effectively drained of water so that gas can be produced. Many potentially good reservoirs are simply drowned by an inability to keep out water that may come from the edges of the production field or from surrounding strata directly or via faults. The measurement of gas production is obviously important from a mining viewpoint. It is also a direct indicator of problem areas - a borehole not producing gas is a liability. The measurement of gas water ratio is also a good indicator of the reservoir state.

Finally, the measurement of reservoir pressure is of great value in assessing what is happening in a coal seam methane reservoir. The measurement of reservoir pressure can provide an indication as to whether the water pressure has dropped below sorption pressure and thus whether gas is free to move. Pressure also provides, through the sorption isotherm, a basis for measurement of gas in place and therefore is a key to any material balance calculation for a reservoir under production. It is considered that the only sensible use of reservoir simulators in monitoring reservoir behaviour is to force the simulator with known flows and to match pressures. Pressure measurement is essential to this.

Rock stress measurement is an important component of mine design. Much has been written on the need to orient mine layouts to minimise the effects of high stress. Just as important, though, are the effects of low stress. If the mine roof is jointed and stresses are low then inadequate horizontal stress may lead to roof collapse because inadequate friction is generated along the joints. The measurement of stress can therefore be as important in low stress environments as it is in high stress conditions. Another benefit of accurate stress measurement is in the location of structures. Stress values usually reflect the types of structures that may exist. For example, normal faults are frequently reflected by the existence of low horizontal stress across the faults. Changes in direction and magnitude of stress are normally associated with faulting.

Accurate, verifiable, stress measurement can be undertaken using overcoring. Siga can undertake this from the surface as part of exploration drilling using the HQ coring system. This process is shown as steps 1 to 8 in Figure 3. The raw traces taken during the overcore are shown in Figure 4 and the processed stress interpretation is shown in Figure 5. The quality of individual traces is readily assessable from data such as that shown in Figure 4. The stress solution shown in Figure 5 is in terms of mean effective stress and deviatoric stress. The quality of the fit of the experimental data to the theoretical best solution is presented as an RMS error expressed as a percentage.

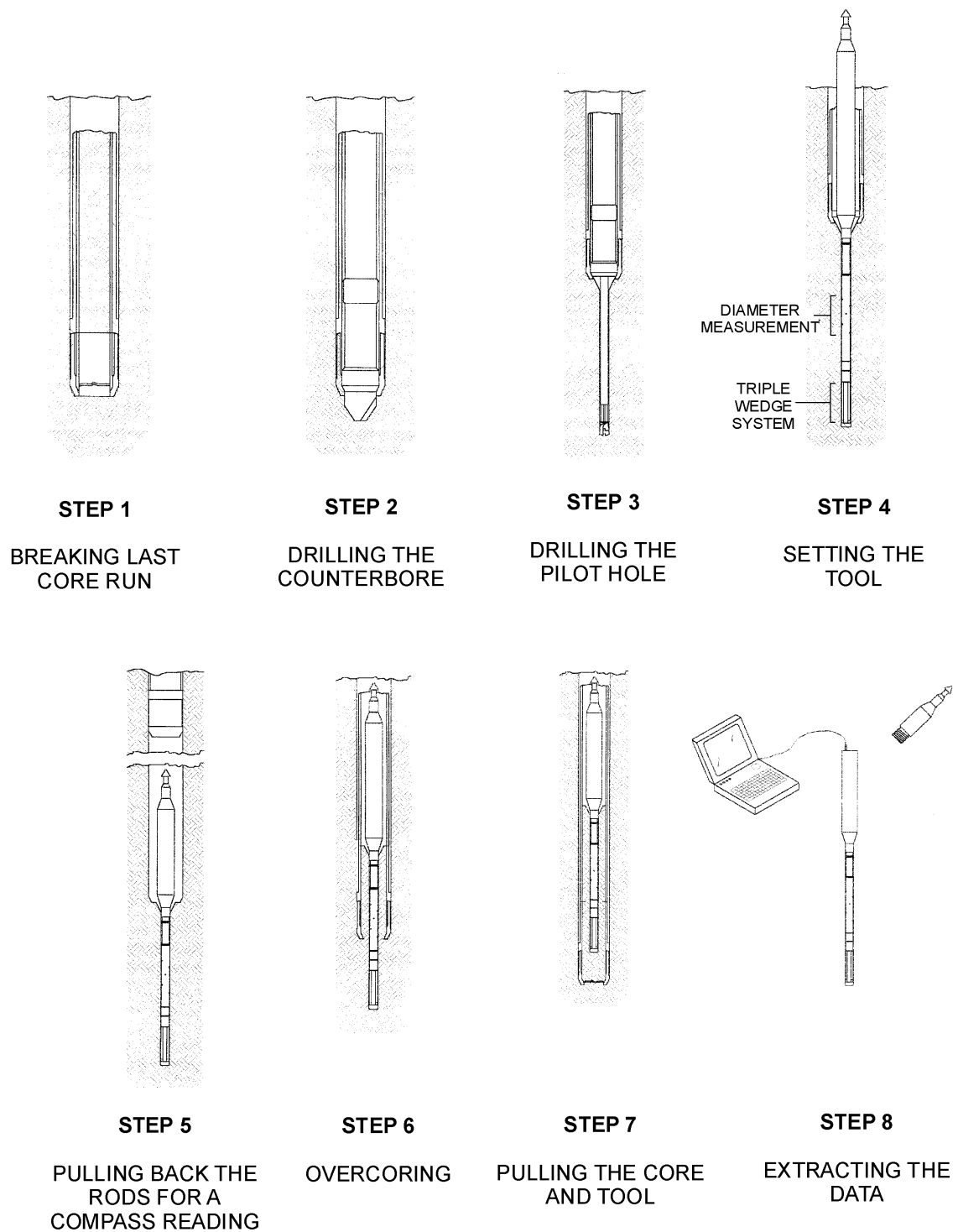


Fig 3 - Overcore steps 1 to 8

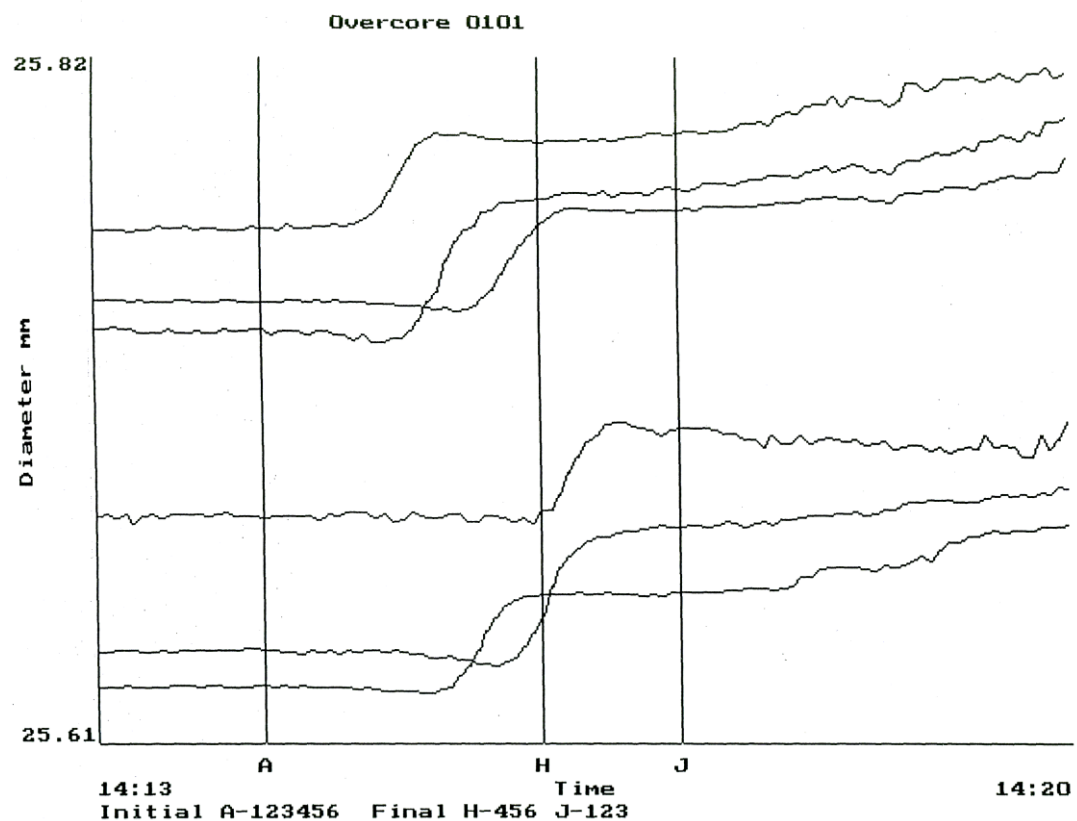


Fig 4 - Raw overcore diameter change traces.

Sigra Stress Measurement
<http://www.sigra.com.au/>
 EQ17

D:\SDATA\STR0101.007

DATE: 22/ 9/2002

TIME: 15:38

Mag Field (nT): 54152

Modulus (MPa): 12590

Poisson: 0.223

Depth (m): 251.8

STR MEAN (MPa): 8.053

STR DEV (MPa): 0.883

ANGLE (degrees): 13.44

ERROR (%): 5.7103

CASE : 1

Pin deflections (mm)

0.05 0.025 0.0 -0.025

X considered

x c - + l n b h 4 5 6

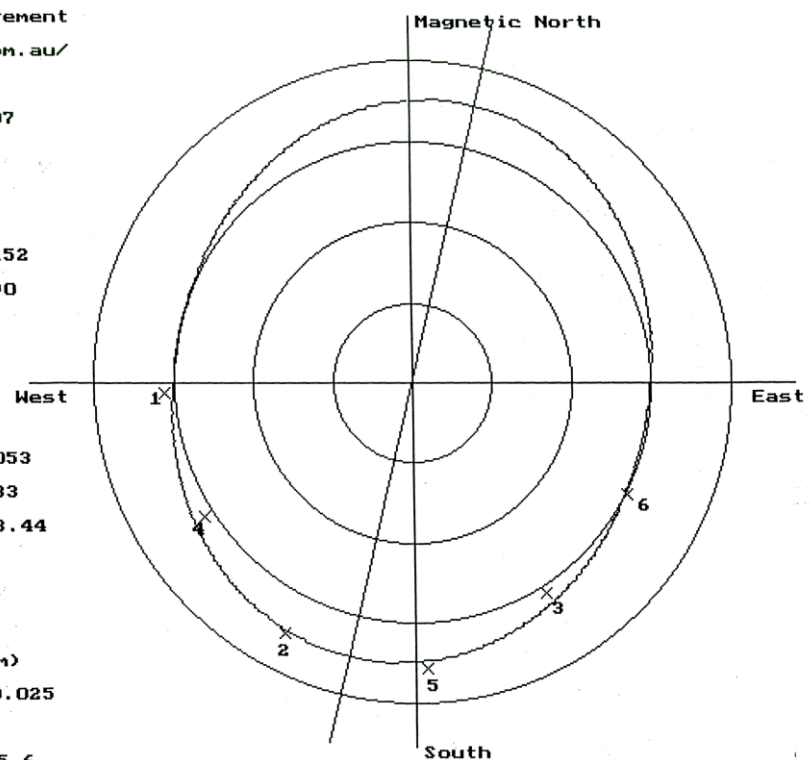


Fig 5 - Graphical presentation of overcore solution

Experience shows that the state of horizontal strain through coal measured through rocks tends to be far more uniform or uniformly varying than the stresses. The stresses tend to reflect the stiffness of the rocks. For this reason, it is preferable to measure stress then subtract the horizontal component due to self-weight, assuming a state of zero horizontal strain. Values of stress that reflect some other strain imposed on the rock mass can then be found and are referred to as tectonic strain. This calculated tectonic strain is a more useful indicator of the stress regime in the ground than are stress values alone. If depth, the Young's modulus and Poisson's ratio of the rock are all known then the stress for the rock can be calculated readily given the tectonic strain values.

The rock stress alone is not in itself of vital importance, what is important is the ratio of stress to strength of the rock. It needs to be remembered that both stress and strength vary with direction and that mining will change the stress direction. Frequently sedimentary strata are much weaker in shear along and tensile strength across the bedding planes. Failures frequently occur along these planes.

To assist mining operations understand their changing stresses, Sigra has now undertaken several stress change measurements in boreholes drilled from the surface. These have involved grouting triaxial and axial stress change cells in boreholes along with fluid pressure transducers. The use of post-initial set expansion agents to the grout permits such devices to be pre-loaded into the borehole so that unloading states¹ can be detected. The measurement of virgin stress and stress change then permits the calculation of stress during the mining cycle. This can be invaluable in assessing difficult to understand roof support problems.

The incorporation of fluid pressure transducers into such instrumentation arrays is important, as rock failure is a function of effective stress. Effective stress is the total stress minus the fluid pressure and hence fluid pressure may be a very important factor in failure. Fluid pressure increases due to strata compression have been detected up to 80 m ahead of a longwall.

CONCLUSION

The accurate measurement of coal seam reservoir and stress parameters is invaluable to mine design and successful operation. This paper details the methods developed by Sigra Pty Ltd for this purpose. The techniques described are based upon wire-line coring at HQ size, as is commonly practised in Australian coal exploration drilling.

¹ Unloading state- If stress reduces the state is one of unloading

A REVIEW OF METHODS TO DETERMINE PANEL AND PILLAR DIMENSIONS THAT LIMIT SUBSIDENCE TO A SPECIFIED IMPACT

Ross Seedsman ¹

ABSTRACT: Coal mine subsidence can be separated into subsidence that develops above the pillars and the sag that develops in beams above extraction panels. Such a separation allows the use of engineering analytical methods to predict vertical subsidence. The subsidence above pillars can be considered to result from compression of the coal pillars and the immediately adjacent roof and floor strata. Estimation of the coal compression needs to consider the difference between yield and failure of coal pillars. Elastic solutions to the settlement of rigid footings may be used to estimate roof and floor compression. Stability and centreline sag of rock beams above extraction panels can be analysed using voussoir beam concepts. Appropriate factors of safety to use in designs needs to be developed and in the meantime a conservative application of methods is required. So that the development of analytical methods is not restricted, there is a need to redesign the databases being used to report mine subsidence.

INTRODUCTION

Historically, the general approach to planning a longwall coal mine has been to determine a layout for maximum reserve recovery and optimum return on investment, and then to assess the maximum possible subsidence impact. Two notable exceptions to this approach have been Gretley and South Bulli/Bellambi West where subsidence constraints were major determinants of the layouts (limitation of impact on urban areas and stored waters respectively). In the last decade, there has been a change in community expectations with regards to the joint use of the surface, with the result that the community are less tolerant of subsidence impacts. Modern mine planners are faced with increasingly stringent subsidence constraints to which they must design. In addition, the subsidence design methods are required to be more transparent so that external reviews by "expert panels" are possible.

Designing productive and reserve-efficient mine layouts to pre-set subsidence constraints is a new challenge. Some of the constraints that are being applied are reasonably well-based, such as the safe, serviceable, and repairable criteria (SSR) for domestic dwellings. There is a wide range of sophistication in the approach to subsidence constraints being adopted by owners of public utilities ranging from reference to Australian Standards to the untenable extreme of zero impact. In environmental cases, some constraints are implied but not quantified.

If and when subsidence constraints are applied, it is in the context of reducing impacts – i.e. reducing vertical subsidence, tilts, or strains. Mining options that are available include not extracting the coal (placing areas of concern outside the impact zone), reducing the width of the extraction panel, or increasing the size of the pillars. To date, empirical² methods have been used to predict subsidence outcomes. Reflecting the variability of subsidence phenomena, these methods offer worst-case outcomes for specific mining geometries. In fact, the methods are not absolutely worst case, as the design lines are often drawn to include "most" points. When the empirical relationships are used in the new environment of subsidence constraints, a range of more productive and reserve efficient layouts can be excluded. Much of the reported variability relates to a failure to account for geological variables in the empirical prediction method. Implicit in the new Subsidence Management Plan (SMP) process is the requirement for the applicant to assess the impact of different geological conditions on their subsidence predictions.

¹ Seedsman Geotechnics Pty Ltd

² empirical: - based on measurements, observation, or experience, rather than theory.

It is important to review how well mine layout decisions can be supported by analytical¹ engineering assessments. From a geotechnical perspective, subsidence design requires consideration of not only stability (stresses/strengths ~ factor of safety), but also acceptable deformations. The application of analytical methods requires validation/calibration to prior subsidence events. As will be seen, the ability to calibrate/validate is limited to some extent because the structure of the subsidence databases has been distorted by the use of the empirical design methods in their construction. Key subsidence parameters are not stored, only the reduced parameters used by the empirical design method.

DEFINING THE SUBSIDENCE CONSTRAINTS

Vertical subsidence constraints are applicable to environmental issues such as flooding and interaction with shallow groundwater tables. In most cases involving surface improvements, the subsidence constraints relate to either maximum ground strains or ground tilts. This introduces an immediate challenge as the analytical methods that are available for subsidence prediction relate to the determination of vertical movements. If tilt and strain constraints are applied, there is a need to relate them back to vertical subsidence.

Traditionally, tilts and strains have been predicted through the use of the constants K1, K2 and K3 that are applied to the ratio of maximum vertical subsidence (S_{\max}) to depth of cover (H):

$$\text{Parameter (mm/m)} = Kx \cdot S_{\max}/H,$$

Where Kx = K1, K2, or K3 for tensile strain, compressive strain, and tilt respectively.

Recalling that new mine layouts may be required to have low panel widths (W), there is a major concern about how well strains and tilts can be predicted. The K constants have been determined empirically by relating them to the panel width/depth ratio. There is a large scatter in the data for K1 (Figure 1) and the design line has been drawn through *most* of the points. Plots for K2 and K3 show similar trends. In Figure 1, there is no basis for drawing the design line to origin, as there are no data points for values W/H ratio less than 0.25. Furthermore, it should be noted that the wider spread of K1 values for W/H ratio less than 0.6 may be related simply to the increased error in the strain measurements as the detection limits are reached. There is no reason why the K value needs to be zero at low W/H values, as the relationship that defines K has vertical subsidence as a parameter – and this obviously tends to zero at low W/H ratios. It is possible that the K values are independent of panel geometry.

When data are available from mine layouts with known similar geologies, they should be used in preference to the compilations such as Figure 1. In the absence of local profile data, appropriate K values for design in the Southern and Newcastle coalfields at low W/H ratios are 1.0 and 2.5 for tensile and compressive strain respectively and 4.0 for tilt, even though values of 0.5, 1.0, and 3.0 may be applicable across the full range of W/H ratios.

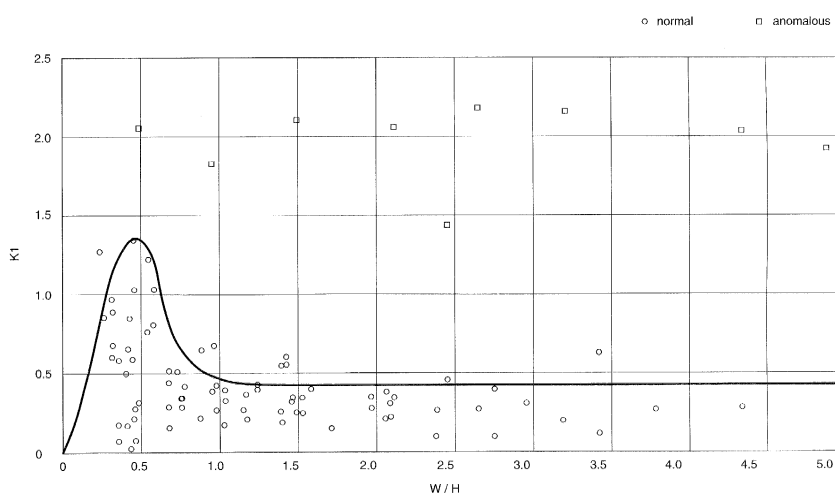


Fig 1 - Variation in K1 as a function of panel width/depth ratio for the Southern Coalfield (Holla and Barclay, 2000)

¹ analytical: – output is a function of a physical law applied to one or more input variables

DEFINING THE SUBSIDENCE IMPACT ZONE

The option of not extracting coal has been adopted in the past as a way of controlling subsidence impacts. The angle of draw concept is the standard way by which this control is implemented. Empirical data are available on the angle of draw as a function of panel width/depth (Figure 2), and it can be seen that there is a very large range even within a single coalfield. Note that the variation in Figure 2 represents about 530m of un-mined coal when applied to the case of mining at 400m depth of cover. The degree of variation in the angle of draw and its impact on reserve recovery highlights the critical need to develop an understanding of what geological parameters are at play. Once again, subsidence data from layouts under known similar geologies is the preferred way of evaluating the appropriate angle of draw for mine layout design. Without such data, and until a better understanding of the variations is achieved, a risk management approach will probably lead to the routine use of values of 26.5° or 35° for the Southern Coalfield. For Newcastle, lower angles of draw have been measured.

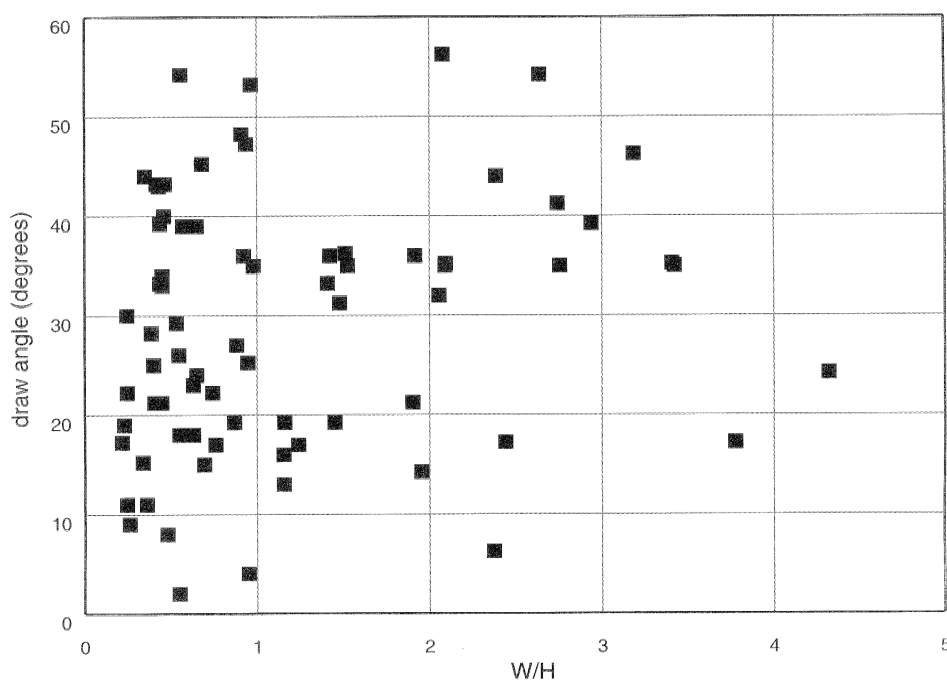


Fig 2 - Variation of angle of draw with panel width/depth ratio for Southern Coalfield (Holla and Barclay, 2000)

PILLAR DEFORMATIONS

Surface subsidence develops above coal pillars as well as above the extraction panels (Figure 3). In deep mines, pillar subsidence represents the majority of the surface subsidence. Pillar subsidence is the result of compression of the coal in the pillar and the compression of the roof and floor. It follows that if the applied stresses and the deformation properties are known, it should be possible to determine the pillar subsidence.

Pillar design methods such as ALTS (Colwell Geotechnical Services, 1998) can be used to estimate the vertical stresses that are applied to chain pillars. For deep mines, the ALTS data base suggests that the tailgate loading angle is less than for shallower mines: in fact a value of 10° is recommended in lieu of the standard 21° . At Bellambi West, vertical stresses were measured in tailgate pillars and the monitoring continued during the extraction of subsequent walls. The low 10° tailgate loading angle was found and significantly, when subsequent walls were extracted, the loading angle increased to a value similar to the standard 21° . The increase in the loading angle may be related to a change from cantilevering over unmined coal to dead weight loading over multiple panels (Figure 4). Interaction between pillars and solid coal with different stiffness may also play a role. It is assessed that simple pillar loading models can be used to estimate vertical stresses on chain pillars in multi-panel layouts.

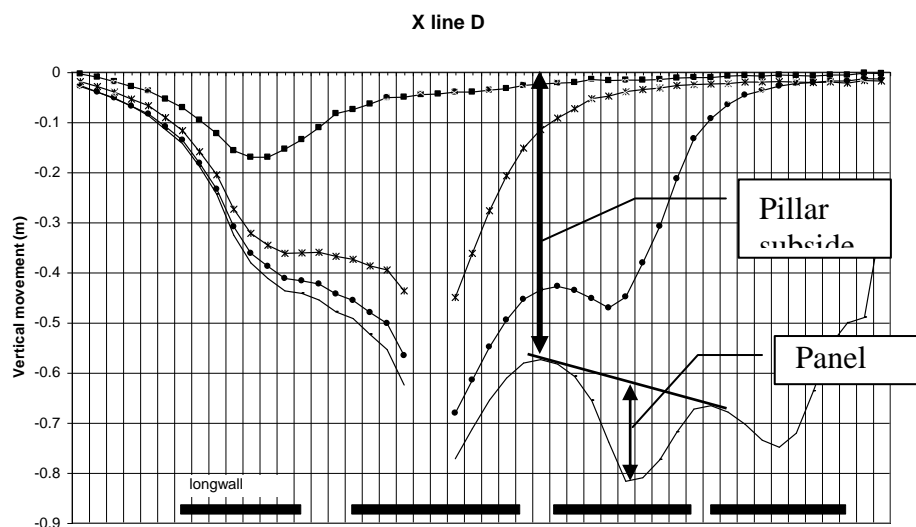


Fig 3 - Typical subsidence pattern above deep longwall panels in the Southern Coalfield

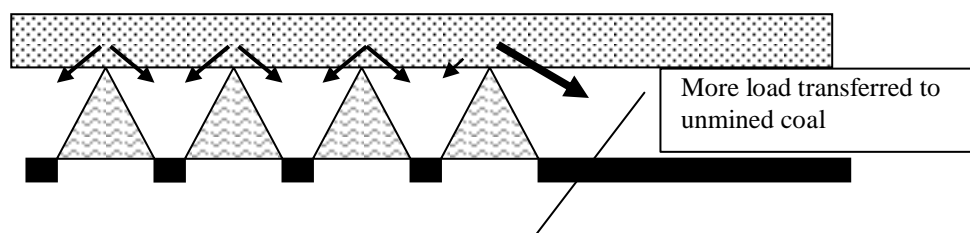


Fig 4 - Schematic showing greater load transfer onto un-mined coal

As a reasonable approximation, the compression of the roof and floor stone can be calculated using elastic theory for the settlement of a rigid circular footing (Poulos and Davis, 1976). For the simple case of no layering in the roof and floor, the deformation of the roof or floor (d_r) can be estimated by the following equation:

$$d_r = 0.7 \times \frac{\text{Vertical Stress Change}}{\text{Elastic Modulus}} \times \text{Pillar Width}$$

The deformation of a coal pillars (d_c) can be estimated by:

$$d_c = \frac{\text{Vertical Stress Change}}{\text{Coal Modulus}} \times \text{Pillar Width}$$

For typical coal measure strata, a modulus of 15 GPa can be assumed for stone and 1.5 GPa for coal. This leads to about 70mm of pillar compression for non-yielding pillars at 250m depth and about 200mm for non-yielding pillars at 500m depth. Lower modulus materials will give higher deformations – for example in some of the shallower Queensland mines, and also in the Newcastle area with the Awaba Tuff in the floor. In some cases, time dependent consolidation of roof and floor strata may need to be considered.

Holla and Barclay (2000) and Ditton (2003) provide data that show that subsidence above chain pillars can be as large as 50% of the height of the pillar (Figure 5). The crossline data in Figure 3 indicate pillar subsidence of in excess of 25%. When roof and floor compression effects are removed, the deformations are still up to 40% of the coal thickness. Using typical pillar stress changes that can be obtained from ALTS, the indicated elastic modulus of the coal is in the order of 200 MPa, well less than the 1.5 GPa that is typical for the Young's Modulus of coal. Clearly, the coal is not behaving elastically.

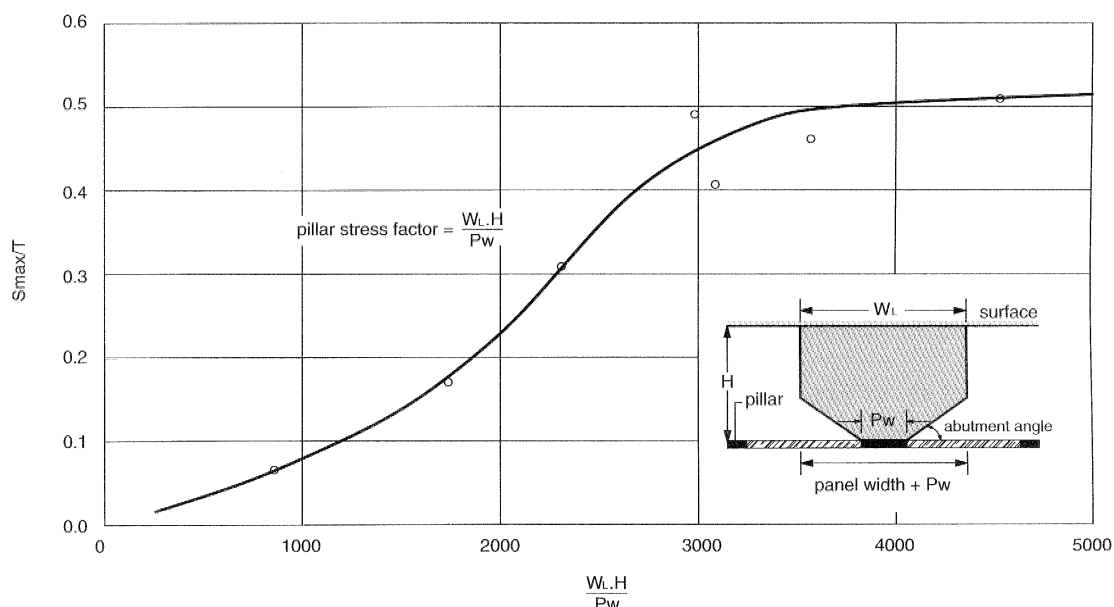


Fig 5 - Subsidence above chain pillars in the Southern Coalfield

Seedsman (2001) discusses a model for the deformation behaviour of coal pillars based on the assumption that the linear strength equation for coal represents a yield criterion and that the squat equation represents an ultimate strength or failure criterion (Figure 6a). A simple application of the model is shown in Figure 6b and 6c, using some data from subsidence above coal pillars in the Southern Coalfield. Figure 6b shows the stress/strain plot for the pillar and Figure 6c shows the derived secant modulus as a function of the pillar stress. The secant modulus is equal to the Young's Modulus up to the point of yield, and then decreases rapidly to be 200 MPa at 45 MPa applied stress.

The model may provide the ability to design yielding chain pillars for subsidence control. Where pillar subsidence needs to be reduced, an approach could be to design pillars with the following characteristics - ultimate stability as given by squat pillar equation greater than say 2.0, and a yield factor as given by linear equation greater than 1.2. A study of pillar subsidence values is required to determine the shape of the secant modulus/pillar stress relationship for yielding pillars with different width/height ratios.

Compression of the coal and roof/floor also develops over the unmined coal at the goaf edge in response to induced stresses. The published subsidence databases do not provide values for actual goaf edge subsidence so it is not possible to investigate this compression in detail. Referring back to the ALTS methodology, the loads are about half those carried by the chain pillars – the distances over which they are applied are poorly known so the stresses are difficult to estimate.

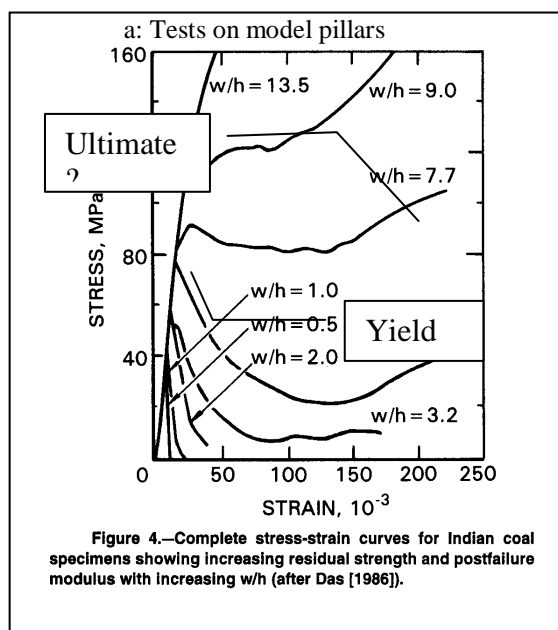
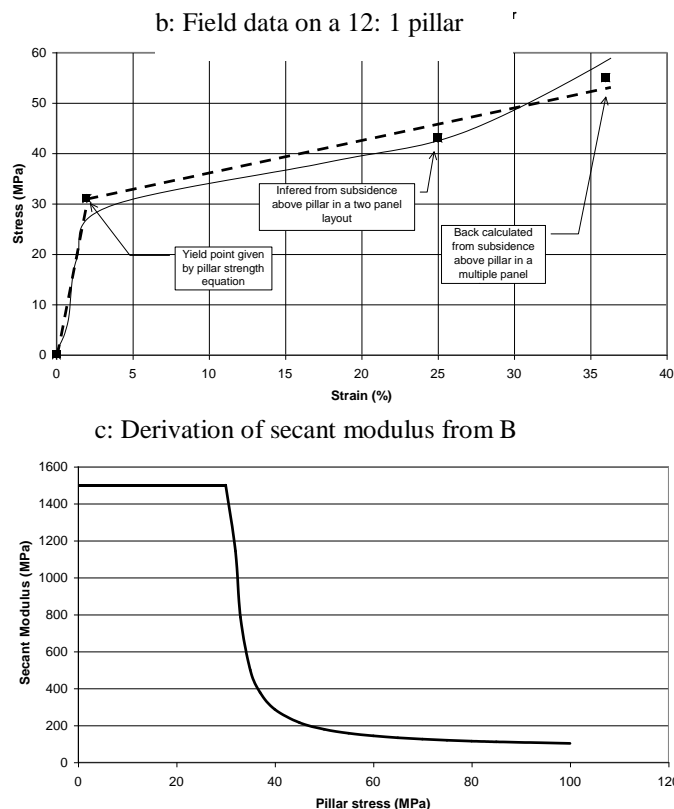


Figure 4.—Complete stress-strain curves for Indian coal specimens showing increasing residual strength and postfailure modulus with increasing w/h (after Das [1986]).

Fig 6 - Model for post-yield deformation of coal pillars



PANEL DEFORMATIONS

The deformations above extraction panels are referred to as panel sag (Figure 2). Panel sag can be considered to be equal to the subsidence that develops above isolated panels, for which there are data from many Australian coalfields. The empirical approach has been to plot normalized subsidence against panel width/depth ratio and to produce a curve that encloses “most” observed values under it. The difference between the Newcastle data (Figure 7) and Southern Coalfield data (Holla and Barclay 2000) has been related to the presence of massive conglomerates and the difference in mining depth. Creech (1995) provided additional Newcastle data for longwall panels without thick conglomerates such that, if Holla was to analyse Newcastle data today, he may have drawn a line offset to the left by about 0.2 units of W/H. This would represent about a 450m reduction in panel widths.

Traditional empirical design methods do not give the ability to respond to the presence of spanning units. As shown in Figure 7, the area under the curve is a legitimate target for design, especially given the fact that such outcomes have been and continue to be achieved. Ditton (2003) has addressed this for the case of spanning conglomerates in Newcastle by defining a subsidence reduction potential in terms of panel width, depth, and the location of unit within the overburden – his method is empirical and as of November 2003 his data base has not been published.

Linear arch or voussoir beam analogues provide an opportunity to analyse panel sag. Sofianos and Kapensis (1998) provide an analytical method that can be implemented readily via a simple spreadsheet. The geotechnical model allows consideration of panel width, depth of cover, thickness of spanning unit, location of spanning unit, density of strata, and the strength and modulus of the spanning unit (Figure 8).

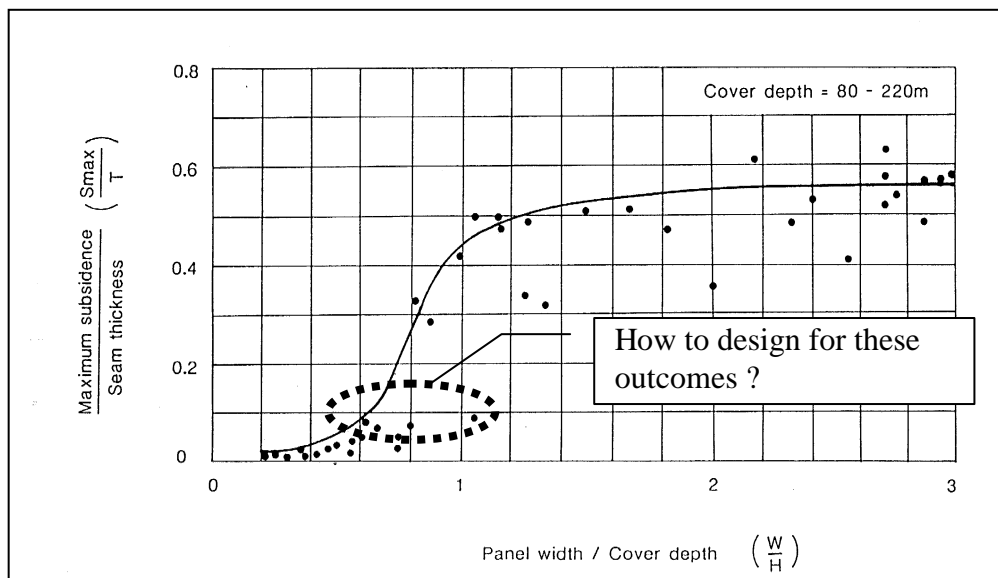


Fig 7- Nomogram to predict maximum expected maximum subsidence in the Newcastle coalfield (Holla 1987)

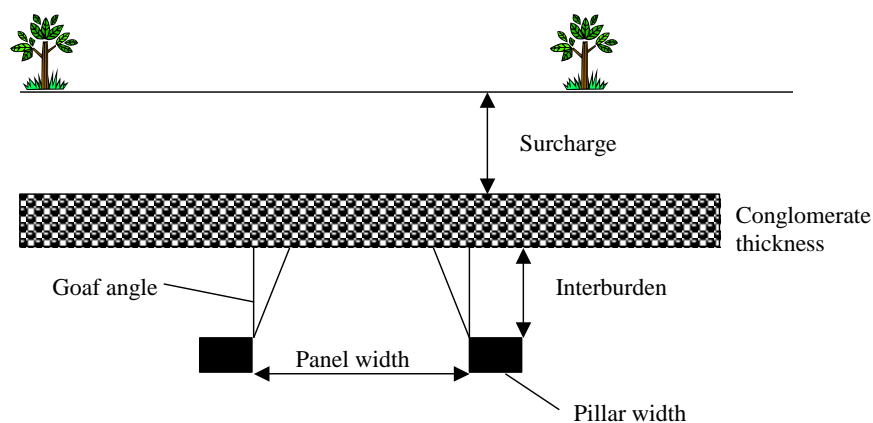


Fig 8 - Geotechnical model for voussoir beam analysis

Figure 9 shows the centreline deflection of a voussoir beam as a function of span and thickness. The relationship between stability and deflection for a 40m thick beam is shown in Figure 10, where it can be seen that deflection increases more rapidly as the onset of compressive failure is approached. The maximum deflection of a beam immediately prior to failure is shown to be equal to about 0.7% of the span and for a factor of safety of 2.5 the deflection/span ratio is 0.15%. Long-term stable spans in civil engineering structures (10m-20m spans), have recorded deflections of 0.1 % of the span (Pells, 2003).

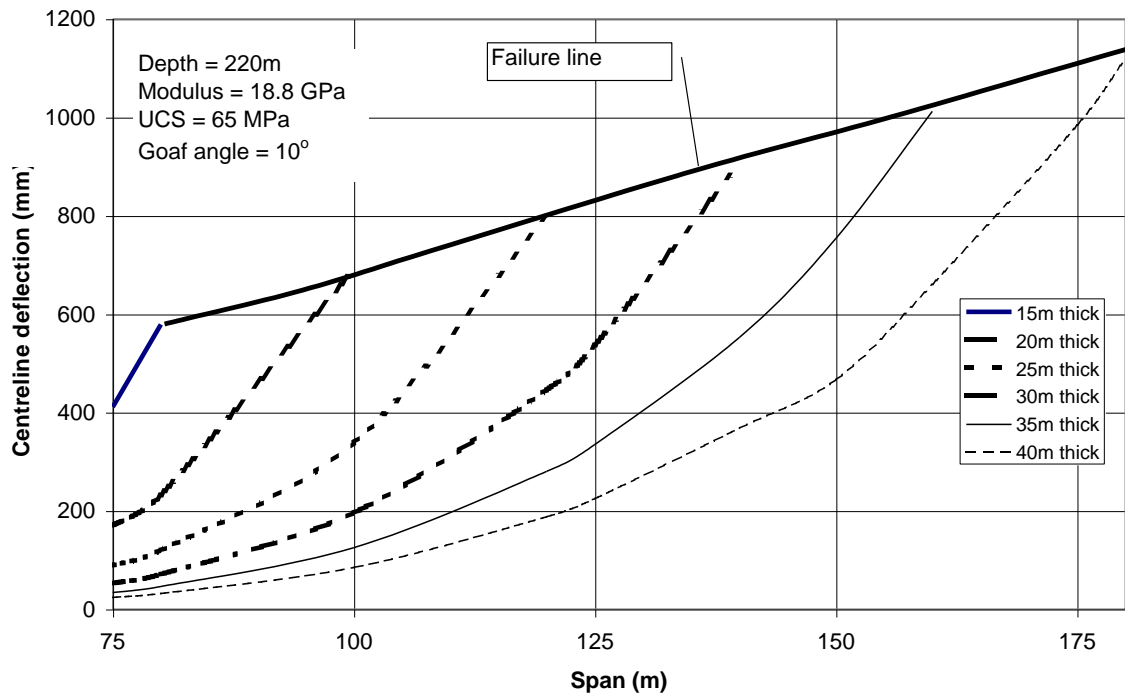


Fig 9 - Centreline deflection of a voussoir beam as a function of panel width and beam thickness

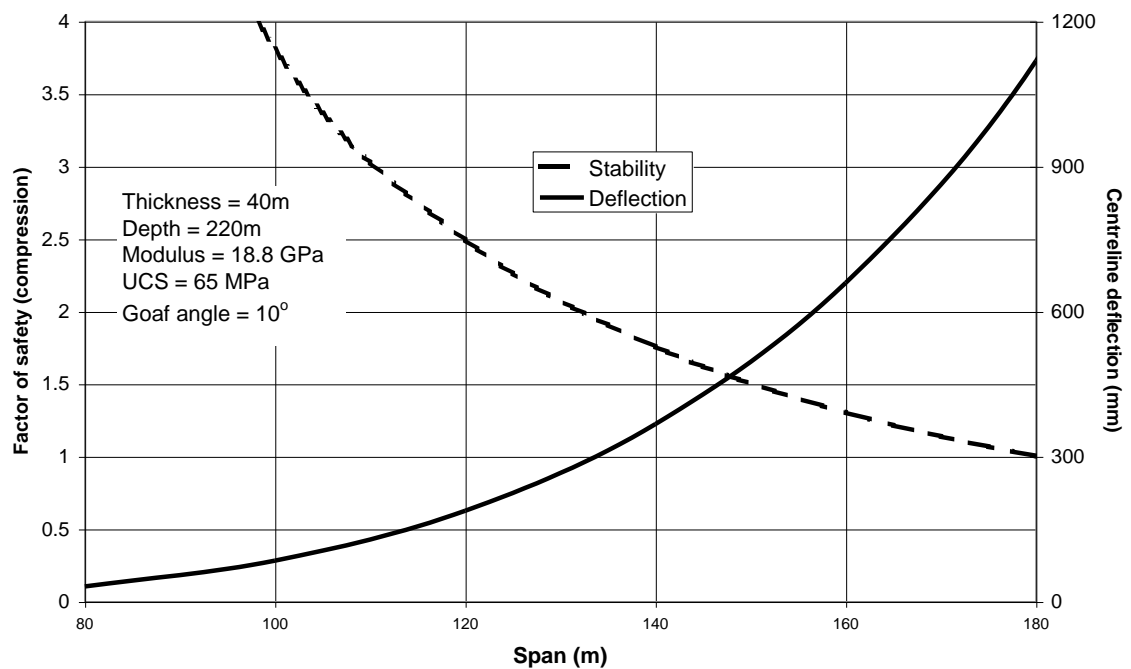


Fig 10- Variation in stability and deflection as a function of span

In a recent application of the voussoir beam model to sub-critical extraction in the Newcastle coalfield, the data of Creech (1995) was used to provide a calibration for the goaf angle shown in Figure 8. It was found to range between 4° and 20° , with an average of 10° . This was assessed to be a reasonable value, since the alignment of the longwall panels tended to be parallel to the regional jointing: an alignment that would tend to provoke relatively steep caving. The face loading data of Strata Engineering (1997) was used to validate the model in terms of the prediction of beam failure.

The model has also been applied to longwall subsidence profiles from the Southern Coalfield such as Figure 3. The analyses suggest that spanning units of say 10m thick may be located towards the base of the Bulgo Sandstone. Microseismic studies suggest that there is no mining induced rock breakage at this level (CSIRO, 1999).

The key input parameter for the model is the thickness of the spanning unit. Knowledge of the depositional environment of the overlying strata can assist in assessing if such units may be present. Spanning units have been found in high-energy environments such as braided rivers (conglomerates), reworked beach barriers (coarse grained sandstones), and biogenically reworked deposits (marine sandstones). Basalts and dolerites may also form spanning units. Drill core is needed to make a definitive assessment. Once calibrated to core, geophysical logs such as neutron, gamma, or sonic logs can be used.

SUMMARY AND RECOMMENDATIONS

The new subsidence requirements present a challenge to mine planners. To maximize extraction while minimizing impacts, the empirical methods of the past need to be complemented by methods that address the geotechnical environment. There is a need to challenge the prediction methods that have evolved over the last 20 - 30 years.

There are alternative analytical methods that may be applied, but the necessary confidence in the predictions will take many years to develop. The methods allow variations in geotechnical conditions to be considered. They need ongoing development and validation. In the meantime, it is important that subsidence databases are not distorted by applying empirical prediction methods in their design.

To advance the use of analytical methods, the following are required:

1. Routine use of methods to predict and back-analyse subsidence events so as to provide confirmation of the input parameters. Lower-bound values for input parameters should be used for calculation of stability indexes and prediction of deflections. High values for stability indexes (ie factors of safety in excess of 2.0) should be used. Calibration data from similar layouts/geologies should be exploited in preference to the existing subsidence databases.
2. Better data bases and data base design— mine owners should allow the Department of Mineral Resources to release subsidence data after a period of years. The minimum data set is panel geometry, pillar heights, extraction height, S_{max} , maximum tilts, maximum strains, goaf edge subsidence, and the distance from the goaf edge to 20mm subsidence. The databases should have absolute values- not ratios. Ideally the full profile data should be released.
3. There needs to be a better method to relate strains and tilts to vertical subsidence. A study based on profile functions may provide such an improved method.
4. A review of pillar subsidence on an industry basis (NSW and Queensland) should be conducted to assess if the secant modulus of yielding pillars can be used as a design tool.

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CASE STUDIES FROM SIMULATING MINE FIRES IN COAL MINES AND THEIR EFFECTS ON MINE VENTILATION SYSTEMS

Stewart Gillies ¹ and Hsin Wei Wu ²

ABSTRACT: The structure of a comprehensive research project into mine fires applying the Ventgraph mine fire simulation software, preplanning of escape scenarios and general interaction with rescue responses is outlined. The project has ACARP funding and also relies on substantial mining company site support. This is essential and allows the approach to be introduced in the most creditable way. The outcome of the completed project will be that the Australian mining industry is in an improved position in their understanding of mine fires and the use of modern advances to preplan actions to be taken in the event of mine fires and the handling of possible emergency incidents. The essential work program of the project is described and work already undertaken at individual mines discussed as examples. The effort is built around the introduction of fire simulation computer software to the Australian mining industry and the consequent modelling of fire scenarios in selected different mine layouts.

Application of the simulation software package to the changing mine layouts necessitates experience to achieve realistic outcomes. Most large Australian mines of size currently use a ventilation network simulation program. Under the project a small subroutine has been written to transfer the input data from the existing mine ventilation network simulation program to Ventgraph. This has been tested successfully. To understand fire simulation correctly the mine ventilation system must be understood correctly first. The results of the project to date are discussed.

INTRODUCTION

Mine fires remain among the most serious hazards in underground mining. The threat that fire presents depends upon the nature and amount of flammable material, the ventilation system arrangement, the duration of the fire, the extent of the spread of combustion products, the ignition location and, very importantly, the time of occurrence. The response to fire by mining personnel will depend upon all of these factors.

There is a lack of knowledge about fire/heat dynamics, some unproven technology in the field of gas sensors and no general agreement on appropriate alarm response systems and measures to be taken in the event of a significant incident. There is a need to couple the detection system with the response system. A research project has been undertaken focused on the application of mine fire and ventilation software packages for contaminate tracing and fire modelling in coal mines and validation of fire modelling software against real mine incidents to reduce the effects of fire incidents and possible consequent health and safety hazards.

With the increasing complexity of technological and managerial development of mines, the effects of mine fires must be better understood. Task Group No 4 Report arising out of the Moura No 2 coalmine disaster of August 1994 in Australia made a number of recommendations including that, *“the capability to model ventilation and the mine environment following an incident should be available at mines”*.

Following these recommendations, sub-committees were formed in 1997 to further progress various findings with Sub-Committee 5 – Incident Management given certain tasks including the question, *“that there is a need for a wider appreciation of current knowledge and improved capability of ventilation management at mines for both routine as well as emergency conditions; guidelines for modelling should include.....”*

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- *Computer modeling of post incident mine ventilation and atmosphere to be a required element in mine safety management plans,*
- *Models interface with standard mine planning packages and be kept up-to-date, and*
- *Development of learned ventilation and fire control responses occur for different incident scenarios and locations, pre-determined for each mine and with plans prepared and personnel trained in appropriate action plans”.*

The primary objective of the study has been to implement a program of research into this complex area utilizing the recently upgraded Polish mine fire simulation software, “Ventgraph”. There is a need to understand the theory behind the simulation program and to allow use by those already familiar with the main existing mine ventilation analysis computer program “Ventsim” currently popular within the Australian industry. “Ventsim” was not designed to handle fire simulation or in fact compressible flow effects in mine networks but can be used as an aid to incident and emergency management.

When a fire occurs outbye the working section, the immediate safe evacuation of miners should always be the first action during the rescue operation. Usually, the intake entries are dedicated as the primary escapeways from the working section. In many cases, the dedicated escapeways are contaminated with fire by-products from abutting entries (eg, belt entry) due to interconnection or leakage through stoppings. It is important to keep these escapeways unobstructed and free from contamination.

It is difficult to predict the pressure imbalance and leakage created by a mine fire due to the complex interrelationships between the mine ventilation system and a mine fire situation. Depending on the rate and direction of dip of the entries (dip or rise), reversal or recirculation of the airflow can occur because of convection currents (buoyancy effect) and constrictions (throttling effect) caused by the fire. This reversal jeopardizes the functioning of the ventilation system. Stability of the ventilation system is critical for maintaining escapeways free from contamination and therefore available for travel.

Simulation software has the great advantage that underground mine fire scenarios can be analysed and visualized. The “Ventgraph” program is described by Trutwin, Dziurzynski and Tracz (1992). The software provides a dynamic representation of the fire's progress (in real-time) and utilizes a colour-graphic visualization of the spread of combustion products, oxygen and temperature throughout the ventilation system. During the simulation session the user can interact with the ventilation system (eg, hang brattice or check curtains, breach stoppings, introduce inert gases such as those generated by a GAG unit and change fan characteristics). These changes can be simulated quickly allowing for the testing of various fire-control and suppression strategies. Validation studies on “Ventgraph” have been performed using data gathered from a real mine fire as undertaken by Wala, et al (1995).

The simulation program has recently been put in a “Windows” format for ease of use and interrelation with other software and with some ability for transferring mine ventilation planning data from programs such as “Ventsim”.

The outcome of the project will be that the Australian mining industry is in an improved position in their understanding of mine fires and the use of modern advances to pre-plan actions to be taken in the advent of mine fires and the handling of possible emergency incidents. The goals of the project are built around the introduction of a modern fire simulation computer program “Ventgraph” and the consequent modelling of fire scenarios in a number of different mine layouts.

The project is undertaking simulations of the effects of common fire causes and fire progress rates. It is undertaking further validation of the simulation model through back-modelling of past fires where gas and other relevant data records are available. It is undertaking some simulation testing exercises incorporating the GAG and other inertisation methods. The simulation of safe escape scenarios before or during a fire as part of emergency evacuation is being examined. Some reference is being made to work undertaken by appropriate bodies such as mines' rescue organizations during pre-planning as part of mine rescue and recovery strategies. Preparation of education (teaching) and training materials on the theory of mine fires (interactions between fire and ventilation and conversely, between ventilation and fire and so on), controlling and combating mine fires, development of evacuation management plans (escape ways) is occurring as part of a technology transfer thrust.

An overview on the findings and results of the project to date and an examination of the effects of fires on mine ventilation systems using numerical fire simulation software such as “Ventgraph” is presented. Various case studies based on the modelling of fire scenarios in a number of different mine layouts are discussed.

EFFECTS OF FIRES ON MINE VENTILATION

An open fire causes a sharp increase in the temperature of the air. The resulting expansion of the air produces a number of distinct effects. First the expansion attempts to take place in both directions along the airway. The tendency to expand against the prevailing direction produces a reduction in the airflow. Secondly, the expansion in volume increases air velocity downwind from the fire causing additional pressure loss, this is known as the choke or throttle effect. Finally, the decreased density results in the heated air becoming more buoyant and causes local effects as well as changes in the magnitudes of natural ventilating energy.

The Choke or Throttling Effect

This effect results from an increase in volume of air as it passes through the fire. This increase in volume is due to gas expansion as well as the addition of combustion products such as fire gases and evaporated water. As a result the velocity of air downwind from the fire is increasing and additional pressure loss following the square law.

The choke effect is analogous to increasing the resistance of the airway. For the purposes of ventilation network analyses based on a standard value of air density, the raised value of this "pseudo resistance", R_t , can be estimated in terms of the air temperature as follows (McPherson, 1993).

$$R_t \propto T^2$$

The value of R_t increases with the square of the absolute temperature (T). However, it should be recalled that this somewhat artificial device is required only to represent the choke effect in an incompressible flow analysis.

Litton et al (1987) have produced an estimate of the increased resistance in terms of the carbon dioxide evolved from a fire. Because the temperature of fumes falls off rapidly downwind of the fire, the choke effect is generally limited to the vicinity of the fire. Thus, the magnitude of the flow constriction (choke or throttle) is likely to be small when compared to the ventilation pressure in the mine ventilation system.

According to the studies undertaken in USA and Japan, reduction of flow and velocity of air due to the choke effect can be in the range of 10 to 25% of the flow before the fire started. Even at this level it can create secondary hazards like accumulation of contaminants, inadequate oxygen content and smoke roll-back.

Buoyancy (Natural Draft) Effects

Local or Roll Back effect

The most immediate effect of heat on the ventilating air stream is a very local one. The reduced density causes the mixture of hot air and products of combustion to rise and flow preferentially along the roof of the airway. The pronounced buoyancy effect causes smoke and hot gases to form a layer along the roof and, in a level or descent airway, will back up against the direction of airflow as shown in Figure 1.

This phenomenon of roll-back creates considerable difficulties for firefighters upstream from the fire, particularly if the conflagration has become fuel-rich. The roll-back is visually obvious because of the smoke. However, it is likely to contain hidden but high concentrations of carbon monoxide and other toxic or explosive gases. Furthermore, the temperatures of the roll-back may initiate roof fires of any combustible material above the heads of firefighters. The most critical danger is that tidal flames or a local explosion may occur throughout the roll-back, engulfing firefighters in burning gases.

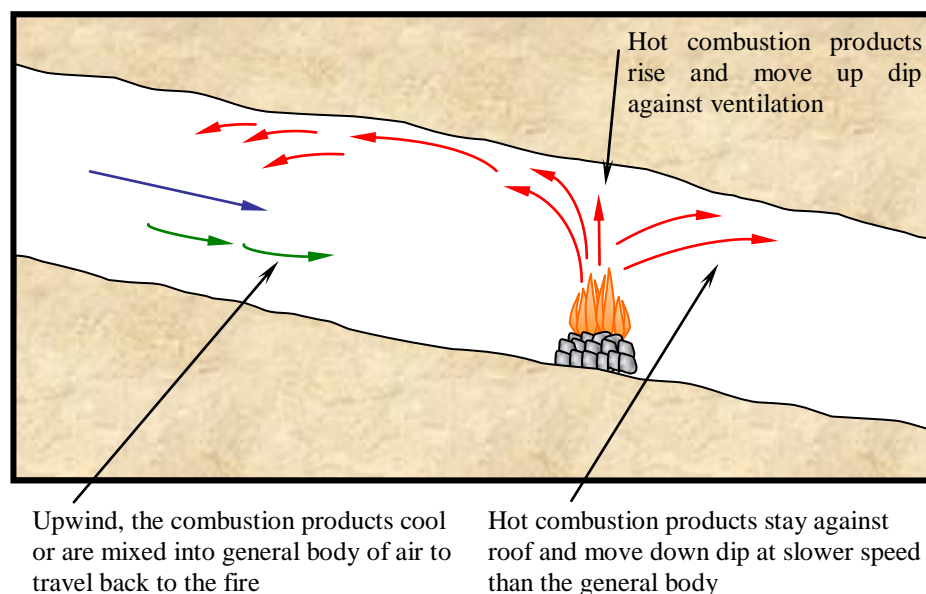


Fig 1 - Buoyancy effect of fires in descental airway (after Bofinger, 1998)

Smoke will roll-back against low airflows, except in a rising airway. How fast and far smoke rolls back depends on how much lower the air velocity is from the minimum air velocity listed in the following table as suggested by Mitchell (1990).

Table 1 - Minimum air velocity for roll-back occurrences (after Mitchell, 1990)

Height of Entry (m)	Minimum Air Velocity, (m/s)		
	Dip 0%	Dip 10%	Dip 20%
1.2	1.00	1.20	1.50
1.8	1.25	1.45	1.80
2.4	1.40	1.68	2.10
3.0	1.60	1.88	2.35

One method of reducing roll-back is to increase the airflow in the airway. This, however, will increase the rate of propagation of the fire. A second method is to advance with hurdle cloths covering the lower 60 to 80 per cent of the airway. The increased air velocity at roof level will help to control the rollback and allow firefighters to approach closer to the fire. However, this technique may also cause the roll-back gases to mix with the air and produce an explosive mixture on the forward side of the hurdle cloth. Furthermore, the added resistance of the hurdle cloth might reduce the total airflow to the extent that a fuel-rich situation is promoted. The behaviour of open fires is very sensitive to modifications to the airflow. Hence, any such changes should be made slowly, in small increments, and the effects observed carefully.

A third method of combating roll-back is to direct fog sprays towards the roof. In addition to wetting roof material and cooling smoke laden air, the air induction effects of the sprays will assist in promoting airflow in the correct direction at roof level. However, it should be noted that sprays would cool the rock surface and possible cause spalling.

Whole mine Natural Ventilation Pressure effects

A more widespread effect of reductions in air density is the influence felt in shafts or inclined airways. The conversion of heat into mechanical energy in the ventilation system is called the buoyancy (natural draft, natural ventilating pressure or chimney) effect. Such a conversion under steady state conditions requires cyclic processes that are provided by every loop of the ventilation network.

The effect is most pronounced when the fire itself is in a shaft or inclined airway and promoting airflow if the ventilation is ascensional and opposing the flow in descental airways. Indeed, in the latter case, the airflow may be reversed and can result in uncontrolled recirculation of toxic atmospheres.

Natural ventilating pressure always exist in a mine and its magnitude mostly depends on the mine's depth and difference between the temperatures of the air in downcast and upcast shafts. In the case of fire, this effect is magnified due to high temperatures leading to unpredictable changes in the airflow distribution.

If the air temperatures can be estimated for paths downstream of the fire then it is possible to determine the modified natural ventilating pressures. Those temperatures vary with respect to size and intensity of the fire, distance from the fire, time, leakage of cool air into the airways affected and heat transfer characteristics between the air and the surrounding strata.

At any given time, air temperatures tend to fall exponentially with respect to distance downstream from a fire. Climatic simulation models may also be employed to track the time transient behavior of air temperatures downstream from a fire, however; in that case, two matters should be checked. One is that the limits of application of the program may be exceeded for the high temperatures that are involved. Secondly, the transient heat flux between the air and strata will be much quicker than for normal climatic variations. Hence, the virgin rock temperature (VRT) in the simulation input should be replaced by a "surrounding rock temperature" (SRT), this being an estimate of the mean temperature of the immediate envelope of rock around the airway before the fire occurs.

Having determined air temperatures in all paths downstream from the fire, the revised natural ventilation pressures for the mine can be determined. These may then be utilized in network analysis exercises to predict the changes in flow and direction that will be caused by a fire of given output. A number of fire simulation packages have been developed to allow numerical modelling of mine fires (e.g. Greuer, 1984; Greuer, 1988; Dziurzynski et al, 1988; Deliac et al, 1985; Stefanov et al, 1984).

Case Studies of Australian Longwall Development Panels

One of the major goals of the study is to examine the effects of fires on mine ventilation systems. To demonstrate how the choke or throttling and buoyancy effects influence mine ventilation systems, fire scenarios were simulated for several case studies based on a typical Australian two entries longwall development panel with various panel configurations.

Panel configurations varied in the case studies including development panel length of 1.5 km or 3 km with panel dipping angles of plus and minus 5 degrees and 10 degrees. To generate a uniform ventilation airflow of 22.8 m³/s through the panel at the working face, a differential pressure equal to 70 Pa was introduced across the stopping at the first cut-through between intake and return entries for the 1.5 km panel length cases studied and a differential pressure of 235 Pa was used for the 3 km panel cases studied.

Figure 2 shows typical two-entry longwall development panel ventilation systems with various panel lengths of 1.5 and 3 km. The fresh air reaches the face through one entry and exhausts from the face through the other, which is the belt entry. The fresh air entry is isolated from the return entry by a series of short life stoppings.

Table 2 shows a summary of the simulation results with diesel fires set in the middle of the development panel for each case study at either intake or return airways. The fire has a 5m fire zone length, a fire intensity of 10 and a time constant of 120 seconds. This is assumed that the fire is spreading (building-up) with a given time constant to reach the given visible fire area and with an intensity of fire i.e., the amount of fuel burning during the fire in the scale of 1 to 10. There were two cases under which the face airflow almost reversed.

When the diesel fire is set in the middle of the return airway at nodal points 24-25 in the 1.5 km longwall development panel mining on a 10% incline upward. The buoyancy effect of the fire acted against the fan pressure but was not enough to overtake the fan pressure to reverse the airflow. As the airflow reduced, the oxygen supplied to the fire is decreased. This caused a significant reduction of the magnitude of the fire thus the buoyancy effect of the fire working against the fan pressure is then reduced. All this means is that now there is more air available to the fire so the fire starts to grow again. This can be observed in the fire simulation output graphic and is illustrated in Figure 3 showing the heat production of the fire during simulation. It can be noted from the figure that the fire causes a sharp reduction in the airflow entering the development panel. In the case of a mine with high seam methane levels this will lead to higher gas levels in the mine air.

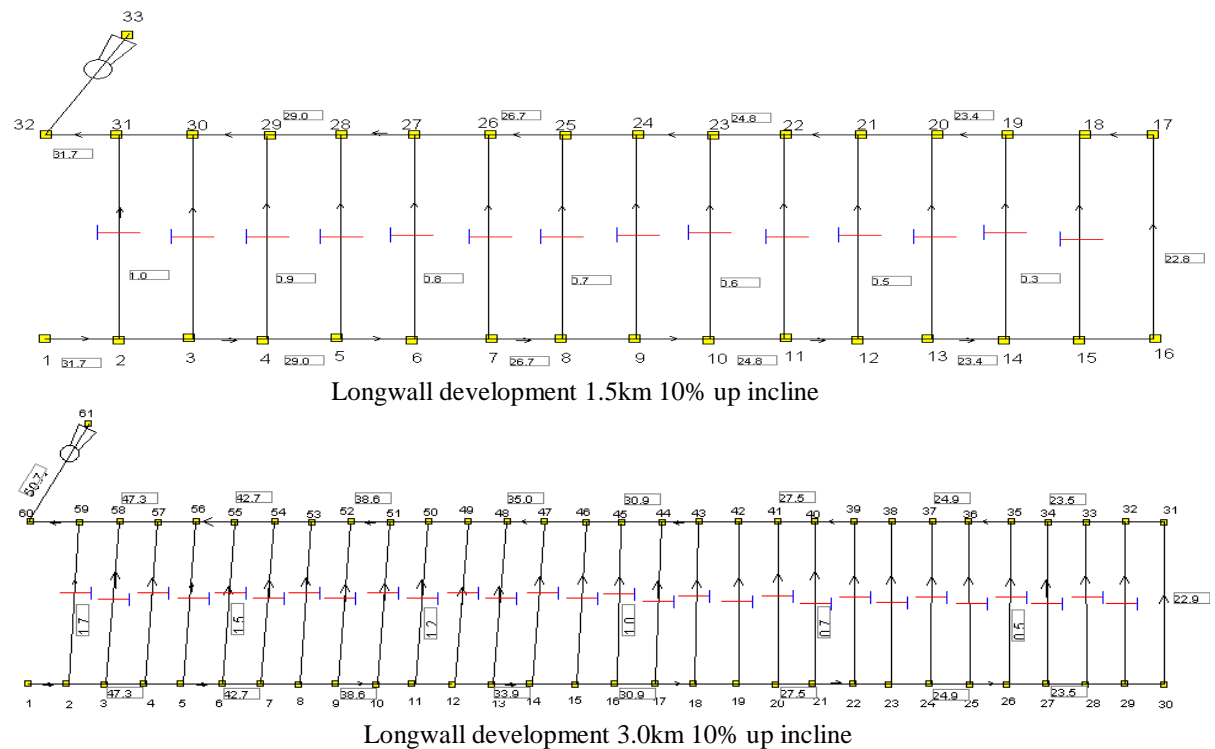


Fig 2 - Two-Entry Longwall Development Panels

Table 2 - Summary of simulation results for 5m-fire zone

Panel Length	Panel Inclination		Face Q m ³ /s		Air Reversal
			Initial	Final	
Fire at return airways (branch 24-25 or 45-46)					
1.5km	5%	Up	22.8	16.1	No
1.5km	5%	Down	22.8	25.7	No
1.5km	10%	Up	22.8	1.3	Almost
1.5km	10%	Down	22.8	30.2	No
3.0km	5%	Up	22.8	21.3	No
3.0km	5%	Down	22.8	24.4	No
3.0km	10%	Up	22.8	20.1	No
3.0km	10%	Down	22.8	24.5	No
Fire at intake airways (branch 8-9 or 14-15)					
1.5km	5%	Up	22.8	26.7	No
1.5km	5%	Down	22.8	16.2	No
1.5km	10%	Up	22.8	29.3	No
1.5km	10%	Down	22.8	1-1.3	Almost
3.0km	5%	Up	22.8	24.3	No
3.0km	5%	Down	22.8	21.4	No
3.0km	10%	Up	22.8	25.7	No
3.0km	10%	Down	22.8	19.4	No

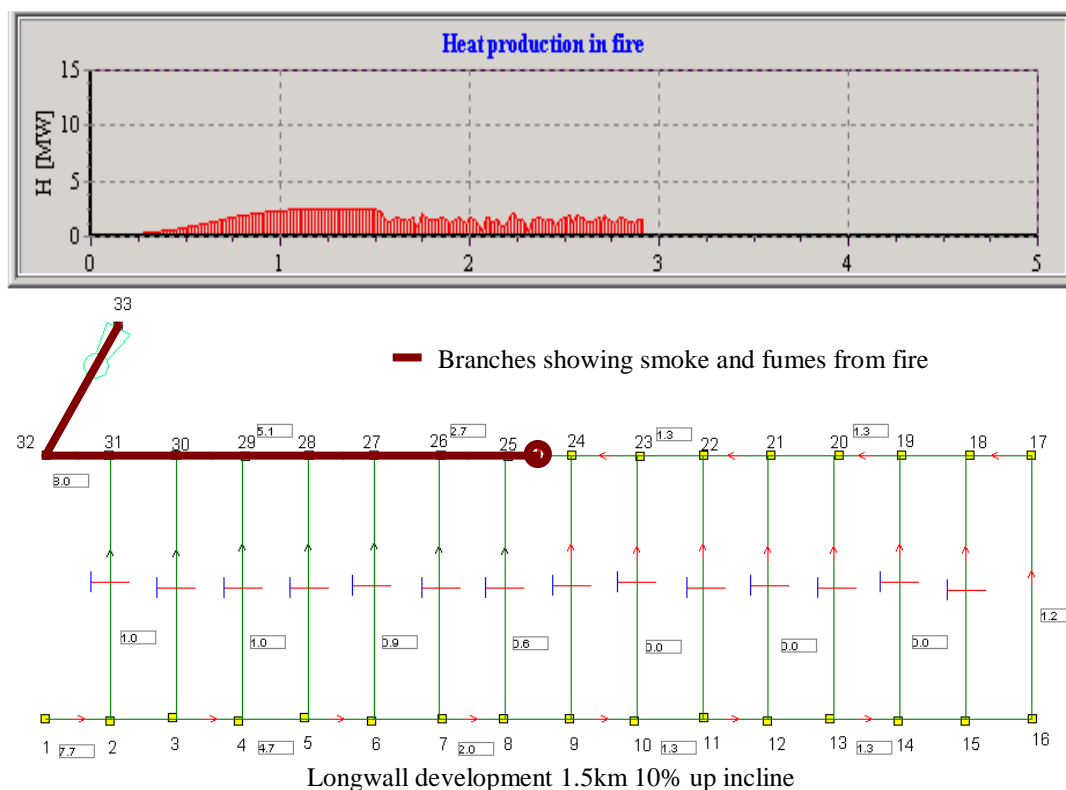


Fig 3 - Smoke progressions and heat production during simulation for 5m long fire zone in return roadway.

When the fire was in the intake airway at nodal points 8-9 in the 1.5 km longwall development panel mining at a 10% decline down. Again, the buoyancy effect of the fire acting against the fan pressure was not enough to overtake the fan pressure to reverse the airflow. The airflow at the faces in both cases was drastically reduced to approx. 1-1.3 m³/s.

The simulations were re-run for these cases with a 10m fire zone length instead of 5m and a summary of results is shown in Table 3. Airflow reversals at the face were observed in the 10% incline case with fire in return airway and also with 10% decline with fire in the intake airway. Figure 4 shows how the smoke progressed and the amount of heat produced during different stages of the fire simulated in the 10% incline case with fire in return air. It is noted that a reduction in heat production from the fire was observed before the airflow reversal occurred. As the products of fire are forced past or over the fire at the moment when airflow reversal occurs, the amount of oxygen available to the fire is drastically decreased.

Table 3 - Summary of simulation results for 10m-fire zone

Panel Length	Panel Inclination		Face Q m ³ /s		Air Reversal
			Initial	Final	
Fire at return airways (branch 24-25 or 45-46)					
1.5km	5%	Up	22.8	4.1	No
1.5km	5%	Down	22.8	29.0	No
1.5km	10%	Up	22.8	16.8*	Yes
1.5km	10%	Down	22.8	33.4	No
3.0km	5%	Up	22.8	20.9	No
3.0km	5%	Down	22.8	24.6	No
3.0km	10%	Up	22.8	18.4	No
3.0km	10%	Down	22.8	26.2	No
Fire at intake airways (branch 8-9 or 14-15)					
1.5km	5%	Up	22.8	28.1	No
1.5km	5%	Down	22.8	3.5	No
1.5km	10%	Up	22.8	36.6	No
1.5km	10%	Down	22.8	1.5-4.5*	Yes
3.0km	5%	Up	22.8	24.9	No
3.0km	5%	Down	22.8	20.6	No
3.0km	10%	Up	22.8	27.4	No
3.0km	10%	Down	22.8	16.0	No

* Air flows in opposite direction to initial flow direction.

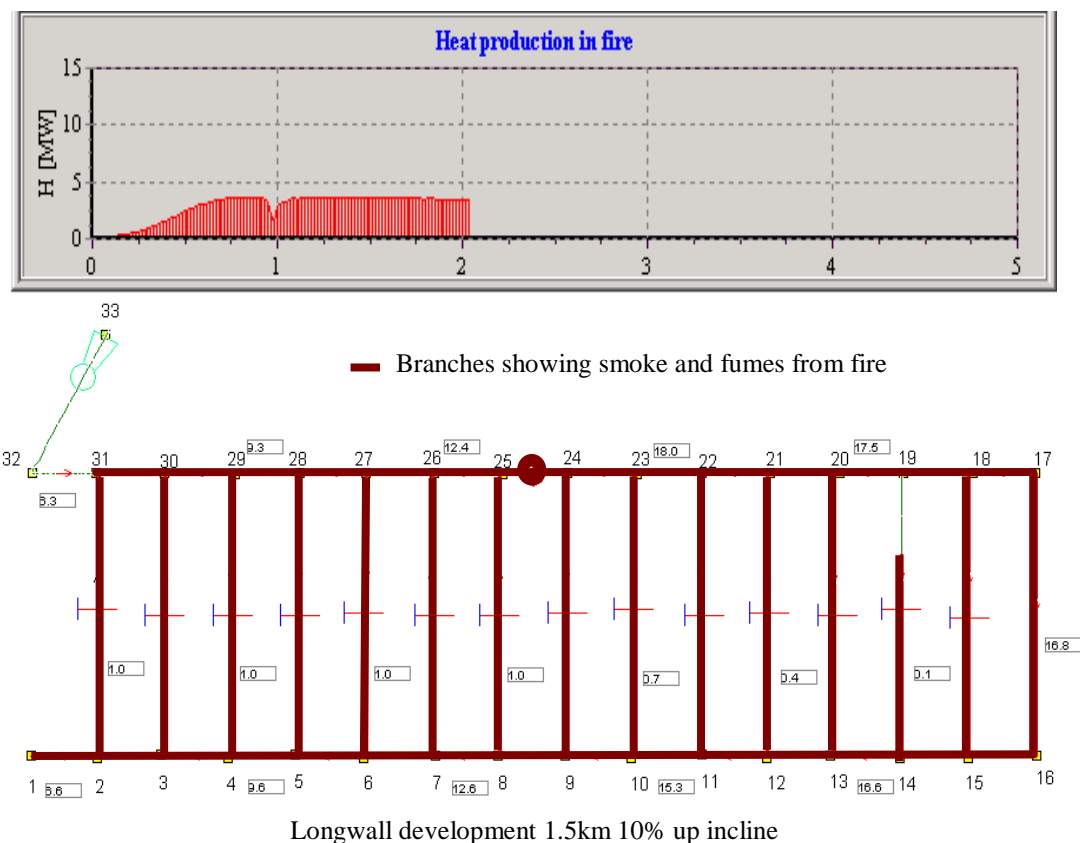


Fig 4 - Smoke reversal and heat reduction observed during simulation for 10m long fire zone in return roadway

MODELING OF FIRES

One of the immediate questions about fire simulation is the magnitude of the fire and how the fire can be modeled in the simulator. In the simulator, various fires can be modeled through combinations of the types of fuels, the length of fire zone (surface area) and intensity. It is necessary to investigate the effects of the length of fire zone and fire intensity on the mine ventilation system. Validation exercises were undertaken to establish how location, type and intensity of fires could be established and modeled.

As for the intensity of diesel fuel fire according to Dziurzynski ¹, some empirical work had been undertaken to establish the amount of heat produced by various types of fuels. For example, 20 litres of diesel fuel was burned in a pool of a surface of 1.72 m². This amount of diesel fuel burned for 17 minutes in an air velocity of 1.5 m/s. With a calorific value of 34 MJ per litre for diesel fuel, a total of 680 MJ of energy was produced. With a burning time of 17 minutes, this means that the heat production averaged at 0.67MW (680MJ/1,020 seconds).

It is understood that with a larger surface area the fire would burn faster and with a larger amount of the fuel available, the fire would burn longer. For example, if 1,000 litres of diesel is available for the same surface areas, the fire will last 850 minutes with a constant burn rate of 0.02 l/s and heat output of 0.67 MW instead of 17 minutes for 20 litres fuel with the same burn rate and heat output. Similarly, if the fuel is spread over a larger surface area say 10m² then it will burn at a fast rate providing that there is an abundant oxygen supply. In this case, the fire will only last for 146 minutes but with a much faster burning rate (0.114 l/s) and a higher heat output (3.88 MW) from the fire. Table 4 shows a summary of calculated heat productions for various combinations of surface areas and amount of fuels available.

According to Yung ², approximately 1 m³/s of air is required for every 1 MW of heat produced from a fire, if the fire is not to be air/oxygen starved. Based on this, it is possible to calculate the minimum amount of air required to sustain a fire without oxygen deficiency. The minimum air quantity is also calculated and included in Table 4.

Table 4 - Prediction of heat outputs from various fires

Fuel (l)	Area (m ²)	Burn time (min)	Rate (s)	Rate (l/s)	Heat Output (MW)	Heat Output (MW/hr)	Q m ³ /s
20	1.72	17	1020	0.020	0.67	0.19	0.67
1000	1.72	850	51000	0.020	0.67	9.44	0.67
1000	10	146	8772	0.114	3.88	9.44	3.88
458	25	27	1607	0.285	9.69	4.33	9.69
818	50	24	1435	0.570	19.38	7.73	19.38
1020	100	15	877	1.140	38.76	9.63	38.76

Several simulation exercises were carried out to examine the relationship between the length of fire zone and the heat production predicted by the simulator. For each of the exercises, a diesel fire was used with an intensity of nominal value 10. Three lengths of fire zone were tested namely, 25m, 50m and 100m. The fires were set in a horizontal airway with about 25 m³/s of air initially. The fires were simulated for about 160 minutes to let the fires become stable. Areas under the heat production charts were measured and average heat outputs for the first one-hour period and the whole period were obtained. Based on heat production predicted by the simulator, it is possible to calculate the amount of fuel burned during the exercises, these are shown in Table 5.

The values for amount of fuel burned and averaged heat production in the fire for the first hour as shown in Table 5, compare well with the values calculated in Table 4. Therefore, it should be possible to model the magnitude of various fires by varying the length of fire zone.

¹ Personal Communication D. Dziurzynski 2003

² Personal Communication Yung D. 2003 CSIRO North Ryde

Table 5 - Prediction of heat production and fuel burnt

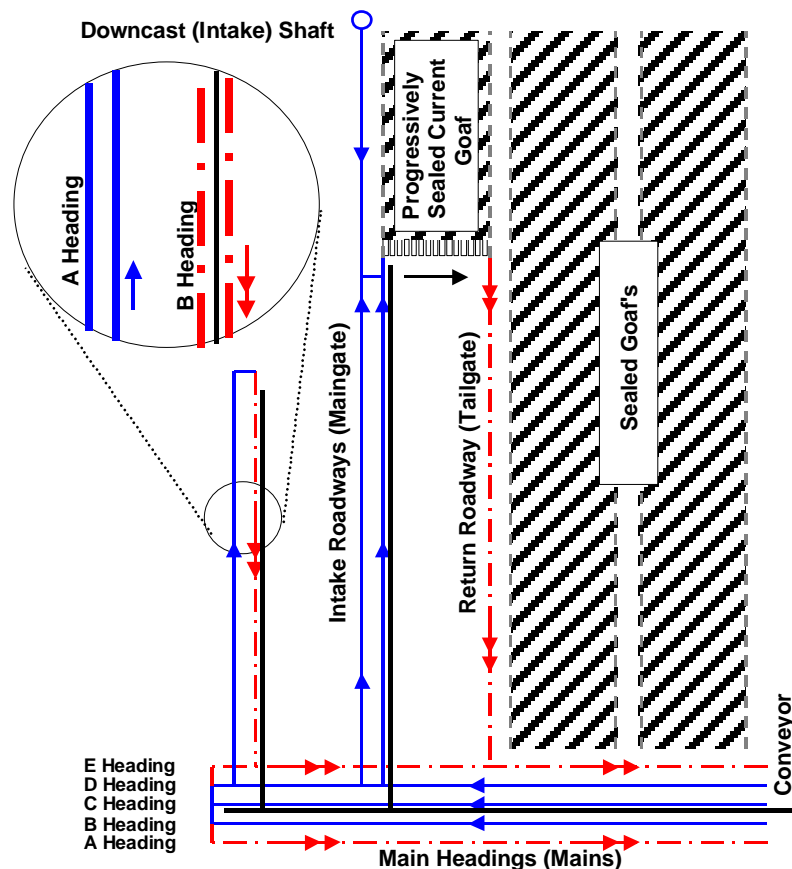
Length (m)	Average Heat Output		Fuel Burned (l)
	Whole (MW)	1st hr (MW)	
25	5.26	4.32	458
50	8.86	7.73	818
100	8.97	9.64	1020

LONGWALL VENTILATION CASE STUDIES

Typical Aspects of Australian Longwall Mining

An example of the layout of an Australian longwall mine is shown in Figure 5. In terms of ventilation nomenclature intake roadways are shown as solid, single arrow roadways whereas return roadways are shown as dashed, double arrow roadways. In this case a raisebore exists behind the current goaf and is shown as a circle with an intake roadway connecting to the longwall face roadway (Mayes and Gillies, 2002). Australian longwalls at present normally use twin headings for development and have typically between five and seven Mains roadways. In longwall development, A Heading (as shown in Figure 5) is an intake roadway with B Heading the return roadway through which the panel conveyor runs.

In the Mains, B, C, and D Headings are typically intake with flanking return roadways, A and E Headings. When all longwalls are being extracted on one side of the Mains only, D and E Headings may be used as return roadways with A, B and C Headings as intake roadways. The conveyor runs in the intake headings, typically in C Heading. In Queensland this roadway is segregated from either one or both of the other intake roadways.

**Fig 5 - Typical Layout of Australian Longwall Mining (After Mayes and Gillies, 2002)**

Mine Ventilation Circuits and Diagonal Connections

Ventilation systems in an operating mine generally consists of an arrangement of multiple, interconnected airway. In most of the cases, airways are interconnected by cutthroughs. In this case cutthrough can be termed a "diagonal" airway and the mine ventilation network is much more complicated than the usual parallel and/or series network.

The airflow through the diagonal connection can be unstable, and its direction depends on the resistances of the branches interconnected with it. Diagonal connections almost always exist in a mine ventilation network; however, good planning can decrease the total number of these branches. Usually it is not possible to eliminate all of them, but it is important to recognize the instability problems associated with their existence and to pay attention to their effect on the mine ventilation network.

The stability of the airflow through the diagonal connection is related to the resistance of the airways interconnected with the diagonal branch which is well known as the basic Wheatstone Bridge balance conditions for electrical circuits as shown in Figure 6. These relationships are as follows:

- a) No flow in the diagonal branch

$$\frac{R_1}{R_2} = \frac{R_3}{R_4}$$

- b) Flow from A to B in the diagonal branch

$$\frac{R_1}{R_2} < \frac{R_3}{R_4}$$

- c) Flow from B to A in the diagonal branch

$$\frac{R_1}{R_2} > \frac{R_3}{R_4}$$

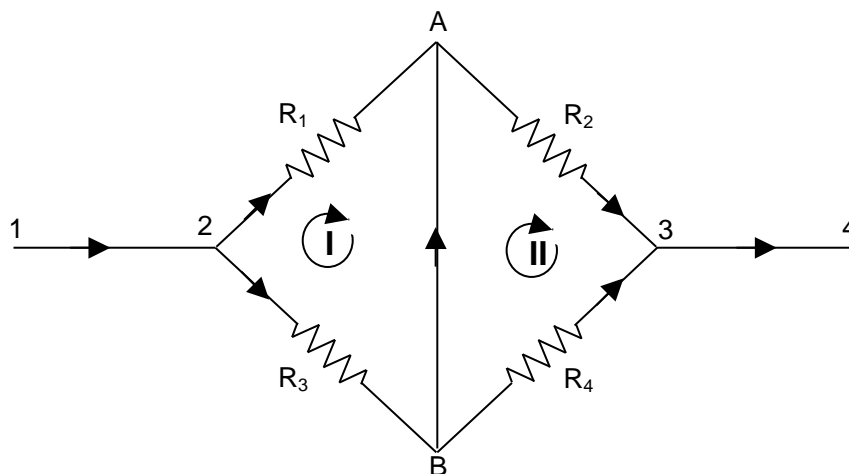


Fig 6 - Ventilation network circuit - Wheatstone bridge

A typical situation where a diagonal connection exists in Australian coal mines is shown in the following diagram. The diagonal connection is established when the next longwall block face road holes through to the existing longwall panel.

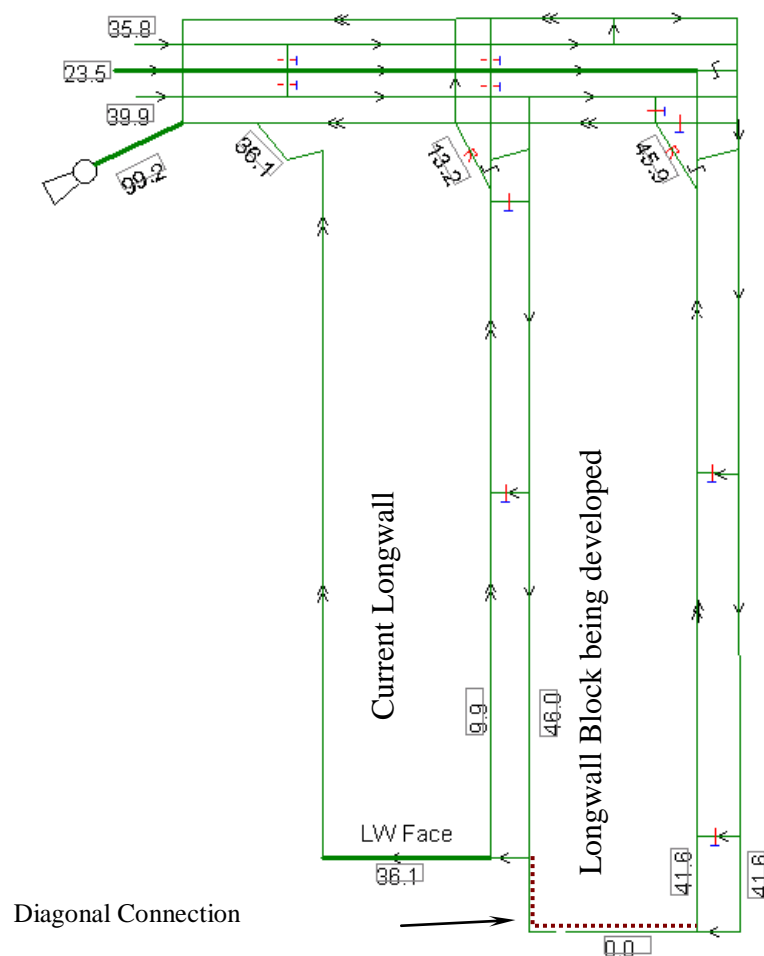


Fig 7 - A typical diagonal connection in Australian coal mine

Case studies were undertaken to simulate the effects of fires on the simplified ventilation circuits before and after the next longwall face road holed through and created an airflow connection to the current longwall circuit. Figure 8 shows how the smoke progresses compared to Figure 7 if a diesel fire with a 30m length of fire zone, a fire intensity of 10 and a time constant of 120 seconds is started 50m outby of the current longwall face. In this case, the fire affects only the current longwall face.

Once the next longwall block face road holes through, an airflow ventilation connection is established. In this case, $2.2 \text{ m}^3/\text{s}$ of air flows from next longwall block to the current longwall panel when the same fire is set at the same location for this case. It was found the fire will affect both current longwall and the next longwall development blocks, after the diagonal connection is established, as fire causes the reversal of airflow through the diagonal connection as shown in Figure 9 when compared to Figure 7. The fire here generates a net buoyancy effect on the airflow in the current longwall maingate. This causes an increased air pressure, which causes air to flow in reverse into the next longwall faceline.

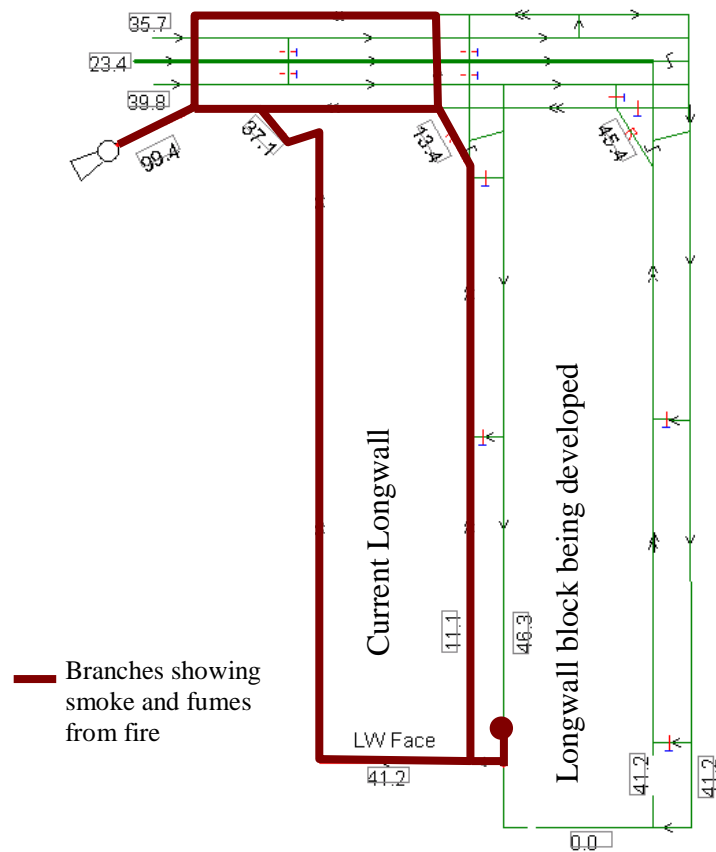


Fig 8 - Smoke progression without diagonal connection

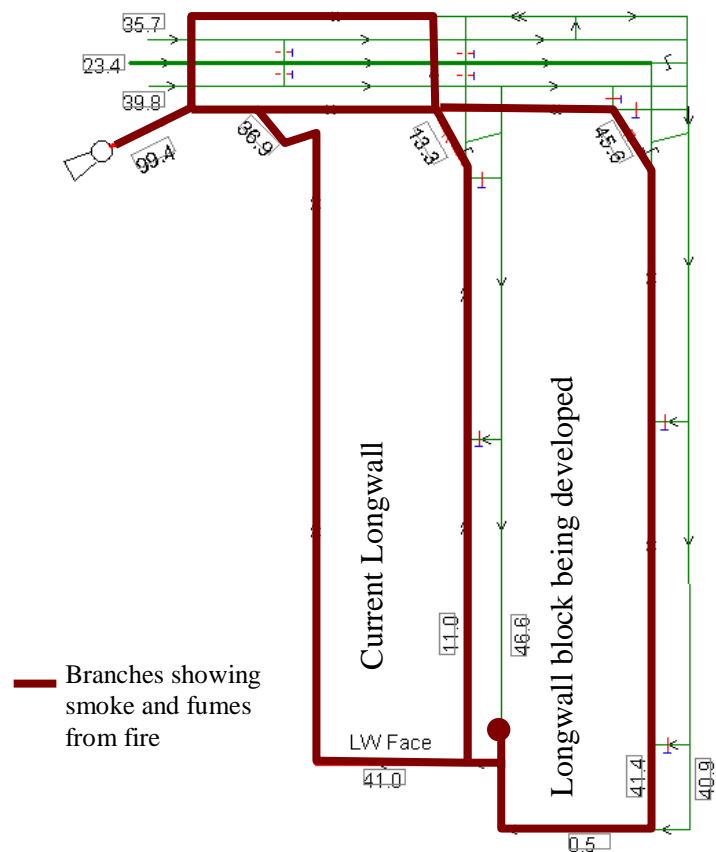


Fig 9 - Smoke progression with diagonal connection

There are two possible situations when diagonal branches can adversely affect the ventilation system (Wala, 1999):

1. When a diagonal connection already exists and a flow reversal or stagnation occurs due to the resistance changes around it (this is illustrated in the above case where the fire causes a net buoyancy change).
2. When a diagonal branch is added that can cause airflow redistributions in the surrounding branches.

It is important to identify the presence of existing or potential diagonal connections within the mine ventilation network as the airflow through the diagonal connections could reverse or stop, due to changes in the adjoining branches within the ventilation network. In some case, contaminated air may attempt to flow into the working areas. However, the existing diagonal branches could be useful during fire fighting or mine rescue operations, where, by alteration of the flow direction in a diagonal branch it is possible to deliver fresh air to areas where it is needed, or to redirect the contaminated air away from working areas where adverse efforts would otherwise result, if the contaminated air was present.

CONCLUSIONS

A comprehensive research project into mine fires funded by the Australian Coal Association Research Program with substantial mining company site support has been carried out. The project involved applying the "Ventgraph" mine fire simulation software, preplanning of escape scenarios and general interaction with rescue responses. Mine site testing is essential and allows the approach to be introduced in the most credible way. Outcomes to date from the project have been detailed.

The project is intended to assist the Australian mining industry to attain an improved position in their understanding of mine fires and the use of modern advances to preplan for mine fires and possible emergency incidents. Examples of work undertaken so far at individual Australian coal mines has been discussed. The effort is built around the introduction of fire simulation computer software to the Australian mining industry and the consequent modelling of fire scenarios in selected different mine layouts.

Application of the simulation software package to changing mine layouts requires experience to achieve realistic outcomes. Most Australian large size mines currently use a ventilation network simulation program. Under the project a small subroutine has been written to transfer the input data from the existing mine ventilation network simulation program to "Ventgraph". This has been tested successfully. To understand fire simulation behaviour on the mine ventilation system, it is necessary to understand the possible effects of mine fires on various mine ventilation systems. Case studies demonstrating the possible effects of fires on some typical Australian coal mine ventilation circuits with diagonal connections have been discussed. It is important to identify and understand these potential effects on the mine ventilation network as the airflow through the diagonal connections could reverse or stop due to the changes in the adjoining branches within the ventilation network. Mining companies need to identify the existing and potential diagonal connections in their ventilation system and analyse how these connections will affect their ventilation system especially in the case of fires. Training is necessary to equip mine ventilation personnel how to identify and minimize diagonal connections in their ventilation system.

ACKNOWLEDGEMENT

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DESIGN AND TESTING OF A RATED FLEXIBLE MEMBRANE STOPPING

Verne Mutton¹, Naj Aziz² and Robert Hawker³

ABSTRACT: The introduction of flexible explosion rated stoppings has become an item of interest to many supply, contract and mining companies in the last few years. These ventilation devices are characterized by the fixing of various fabrics with direct bolt attachments to the mine roadway perimeter. They are often rated through design calculations rather than practical assessment of the system for load carrying capacity. These design calculations are often based on the tensile capacity of the fabrics and assume full distribution of load over the entire cloth surface which is impossible to achieve in practice. The potential limitations that may exist with flexible stopping systems through Minova Australia's experience in the development of its Flexi-Stop ventilation stopping is described.

INTRODUCTION

During the course of underground coal mining, it sometimes becomes necessary to install stoppings to separate the air paths in the mine. In order to fulfill this function efficiently, the stopping must have minimal leakage over its intended life. Underground mining imposes a variety of conditions on the stopping such as fluctuations in ventilation pressure, changes in the boundary conditions due to movement of strata, impoundment of water and changes in atmospheric humidity. Stoppings have been constructed of either cementitious based products including cement based shotcrete, Gypsum plaster, ash bricks and stoppings consisting of props, wooden battens and plasterboard. Stoppings can be damaged by strata convergence and are often difficult to repair in order to minimize leakage. Severe damage often necessitates complete replacement which is difficult when personnel inbye require ventilation air. Many of these stoppings are required to possess an explosion rating. Legislation introduced in Queensland in 1993 required that stoppings be able to withstand pressures of 14 kPa (2 psi) and 35 kPa (5 psi) under the guidelines of the Queensland Department of Mines and Energy's "Approved Standard for Ventilation Control Devices". Some mines in New South Wales have adopted the same standards for stopping installation.

Minimization of leakage through the coal ribs is also an important issue that sometimes requires treatment with strata injection and effective bolt support. Strata support in cut-throughs has not been as rigorously treated as gateroad support for economic reasons and this has often affected the long-term performance of stoppings in terms of structural integrity and air leakage. However, rib support has become part of the stopping design system in recent years. Minimization of strain softening in the roadway will help in the performance of any stopping design.

The flexible stopping concept allows for strata convergence and has many advantages over a conventional rigid stopping. Depending on the design, it can be installed temporarily and can be unbolted and hung to the roof if renewed access for men and materials becomes necessary. The Flexi-Stop design can be installed where high ventilation pressure differentials exist. It is possible to transport many stopping kits in a conventional materials pod whereas other rigid designs require large material weights and volumes with a greater component of manual handling. Depending on the design, these stoppings offer productivity gains to the contractor or mine installing them with potentially up to three being installed per shift.

In 2002, Minova Australia embarked on a test program of flexible stopping designs that was based on evaluation of the response to both static and dynamic pressure testing. Initially static testing of various cloths

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was undertaken at the School of Civil, Mining and Environmental Engineering, University of Wollongong (UOW). This research program evaluated the static load response of a stopping design. Results from this testing gave confidence to undertake dynamic pressure evaluation at the National Institute of Occupational Safety and Health's testing facility at Lake Lynn Experimental Mine (LLEM), PA during November, 2003. These stoppings were designed to resist an explosion pressure of 14 kPa (2 psi) and 35 kPa (5 psi) and were evaluated in a range of pressures from 19.6 to 42.7 kPa (2.8 to 6.1 psi).

These research programs focus on the ability of particular stopping designs to maintain their structural integrity whilst being subject to a specific methane or methane and coal dust explosion. A series of controlled explosions of successively increasing magnitude provided data that can be used to optimize future seal designs in terms of strength and the economics related to material usage and installation times.

The installation methods, static load results and the explosion test results associated with the Flexi-Stop flexible stoppings are presented in this report. Measurements of stopping response to both static and dynamic load are summarized in tabular format.

STATIC LOAD TEST PROGRAM

Static load testing of the stopping system was conducted in three separate programs. This testing series documented the static load results achieved on four fabric types referred to in this paper as Fabric A, Fabric B, Fabric C and Fabric D. The aim of the test series was to achieve a static load equivalent to 14 kPa (2psi).

Each static load test was conducted by fixing the fabrics inside a purpose built steel frame manufactured from "I" beams. The internal dimensions of this steel frame were 4.0m x 2.7m with the steel frame designed for a maximum static load capacity of 19.6 kPa (2.8 psi), which is equivalent to 21.5t.

The frame was located in a horizontal position above the lab floor and supported using a combination of concrete blocks and steel props 1m from floor level.

The fabric loading was carried out using a 70t capacity hydraulic jack mounted at the centre of the load frame. The ram was powered by a Rexroth 630 ATO pressure pack with the applied load monitored by an Interface Model 24/HL load cell. Figure 1 details the frame assembly and load testing position.



The first test program involved the static load testing of Fabric A and Fabric B. Fabric A was a white PVC non-woven fabric of thickness 0.45mm and tensile strengths documented as 40 kN/m. Fabric B was a green canvas type fabric of thickness 0.65mm with no documented strength characteristics. Both fabrics were fixed to the steel frame by wrapping the periphery of the fabrics around 150mm wide wire mesh (aperture size 50mm x 75mm) then bolting to the steel frame using 150mm square steel plates and 20mm diameter bolts. In addition to this, plywood and G clamps were used to further distribute load of the cloth around the frame perimeter. Figure 1. shows Fabric A loaded into the test frame.

Fig 1 - View of the testing frame with Fabric A attached.

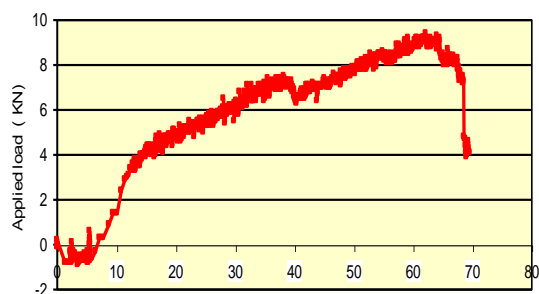


Fig 2 - Hydraulic jack load vs ram displacement

Loading of Fabric A commenced with the placement of 1112.10 kg of weights. It was evident during the placement of these weights that the fabric was unevenly loaded. This uneven tension was further evident during loading via the hydraulic ram, which eventually led to failure of the fabric at a total load of 2032 kg, which was well short of the desired load needed to achieve 14 kPa (2psi). Figure 2 shows the Hydraulic jack load vs the ram displacement. Fabric B also exhibited uneven loading with failure occurring during placement of the weights at 661kg.

Both Fabrics exhibited failure at the anchor plate zones located in the central positions on the long span section. This failure was due to point loading of the fabrics at these points and characterized by initial puncturing of the fabric then tear propagation of the fabric around the plate zones. Figure 3. shows the failure of Fabric A around plate zones.



Fig 3 - Fabric A failure at plate zones

It was evident from the first test program two issues needed to be addressed for a 2psi load rating to be achieved. The first of these was the effect of uneven load distribution concentrated around the plate zone. This effect was hoped to be overcome through either a more efficient load transfer system or higher tensile strength fabric. The second consideration was tear propagation of the fabric once puncturing of the cloth had occurred.

As a consequence of the results achieved during the first test program Fabric C was selected for the second test program. Fabric C was a composite fabric consisting of a high tensile strength geogrid backed by a non woven fibrous mat. The tensile strength of Fabric C was 26.5 kN/m. No tear strength data was available but it was thought the composite nature of the fabric would offer improved tear propagation resistance.

Fabric C was again fixed in the same method as Fabrics A and B however a thin sprayable liner (TSL) Tekflex was used to seal the fabric around the perimeter. It was anticipated that the TSL would be later used as the sealant for Fabric C thus the reason for its inclusion in the load test.

Uneven loading was again seen during placement of the weights at a value of 1050kg. This weight was actually increased during the loading test due to the addition of spacers placed between the hydraulic jack and the loading frame. Spacers were required during the ram loading due the vertical displacement of the fabric exceeding the jack pistons stroke length. The final weight of 1089.5kg was added before applied ram load.

Several stages of failure were evident during the test program including tearing of the TSL, snapping of the geogrid around the plate zones and tearing of the fibrous mat around the plate zones. A total load of 7900kg was achieved during the test with a vertical displacement of 300mm. Figure 4. shows the three failure events seen during the test. Figure 5 shows the hydraulic jack load vs the ram displacement for the test program 2.

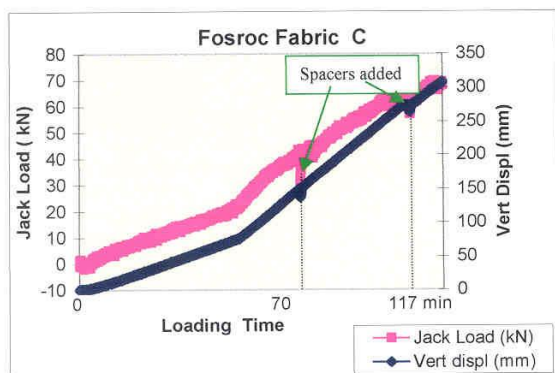


Fig 4 – Jack load and FFC sagging versus loading time



Fig 5 – 280mm lateral width of the torn felt around the central plate @ 68 kN load (total applied load – 0.795t)

Although a significant improvement in load had been achieved the result was still well short of the required 15.5T needed for 2psi. To overcome point loading at the plate zones, across the short span, a different fixing system was required and a higher strength single component fabric.

Test program 3 involved the testing of both these items with Fabric D selected for the test work. Fabric D was a PVC coated ultrahigh tensile strength woven fabric with weight of 2kg/m, thickness of 1.7mm and tensile strength of 200kN/m. Initial in-house testing by fixing Fabric D in a steel slot structure indicated that even distribution of load could be more effectively achieved. Figure 6 and Figure 7 detail the test work and slot method used to achieve loads of 7000kg/m. Based on this result it was thought that over the 4m length of the loading frame a load near 28t could be achieved and would give greater confidence to undertake further testing at UOW.



Fig 6 - In house testing of fixing system and Fabric D



Fig 7 - Slot fixing system

The final test program involved fixing fabric D using the slot system along the 4m length of the testing frame and bolting the cloth using square steel plates with rubber washers along the 2.7m lengths. This test program was completed in two sections due to early failure of the slot fixing system in the first section. Adjustments were made to the steel grade of the slot and the method used to hold the fabric in the slot. Section 2 involved loading of fabric D with the same method as for the previous fabrics tested with the addition of sand to further distribute load. After placement of the initial 2226kg of weight it was evident the load was more evenly distributed along the 4m length. During testing additional gussets were added to bond the slot to the test frame and stop bending of the slots. Further timber posts were also added to support the test frame as buckling of the 4m length was beginning to occur. A maximum load of 20.371 t was applied to the cloth, equivalent to 2.71psi

and near to the 2.8psi static load capacity of the test frame. Figure 8 details the Hydraulic jack loads and vertical displacement of Fabric D during the section 2 tests. Figure 9 details Fabric D fixed in the test frame during loading and Figure 10 shows the tests frame buckling under load.

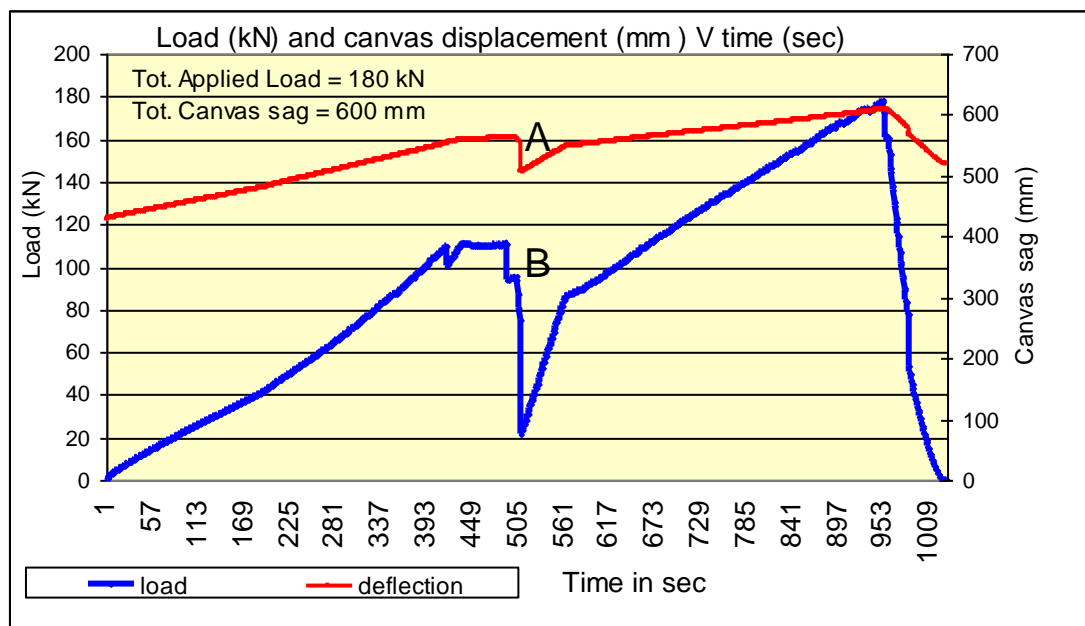


Fig 8 - Hydraulic jack load vs vertical displacement for Fabric D



Fig 9 - Fabric D load distribution in slot



Fig 10 - Buckling of the test frame under load

EXPERIMENTAL MINE AND TEST PROCEDURES

LLEM is one of the world's foremost laboratories in conducting large-scale explosion testing of seals and stoppings and the test area is designed to withstand explosion pressures up to ~700 kPa (~100 psi). Figure 11 shows an expanded view of the stopping test area in the multiple-entry section of LLEM.

Two Flexi-Stop stoppings were constructed in cut-throughs 6 and 7 between B- and C- drifts. There were already seals in the first three cut-throughs from the simulated face and concrete block stoppings in cut-throughs 4 and 5. Before each explosion test a hydraulically operated, track mounted, concrete and steel bulkhead was positioned across E-drift to contain the explosion pressures in C-drift.

Four full-scale explosion tests were conducted in LLEM C-drift in November, 2003. These gas explosions were designed to provide an increasingly higher pressure pulse on the stopping designs during each subsequent test.

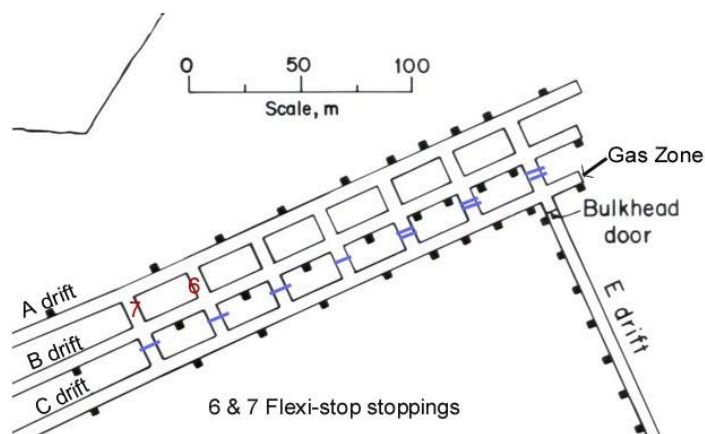


Fig 11 – LLEM Layout

The Flexi-Stop flexible stopping designs were located in cut-through 6 at 167.6 metres and cut-through 7 at 197.8 metres from the face of C-drift. Refer to Figure 11 for details.

For first two tests (test 459# and 460#) a clear plastic diaphragm blocking off C-drift contained the natural gas and air mixture within a 3 m deep by 3.7 m wide ignition zone ($\sim 27\text{ m}^3$). For the last two tests (tests # 461 and # 462), the methane-air ignition zone was extended out to 8.2 m from the face forming a gas volume of $\sim 78.3\text{ m}^3$. A circulation fan inside the ignition zone ensured uniform mixing of the methane-air mixture before the explosions were set off. For tests

460# and 462#, the circulation fan remained operating during the ignition process to provide turbulence and more rapid flame development. The electric match, used as the ignition source, was placed either mid-width within the zone and 0.9 m outbye the face near the door (test # 459) or at mid-height and mid-width near the face (tests 459# - 461#). Double point ignition (one electric match near the center of the face and one located near the right inbye corner) was used during test # 462. Pressures generated were lower when ignition was further outbye the face because the explosion would vent outward as it burned towards the face

As the explosions travel towards the stoppings down C-drift the static pressure was measured at a transducer ~ 4 metres from the face.

INSTRUMENTATION

Each drift has ten environmentally controlled data-gathering stations inset in the rib wall. Each data-gathering station houses a strain gauge pressure transducer that is perpendicular to the entry length (and explosion gas flow) and therefore measures the static pressure generated by the explosion. Pressure transducers were located on the C-drift rib at locations 152.7, 182.3 and 230.7 metres from the face near the Flexi-Stop locations. Most of the pressure transducers were rated at 0-100 psia, with 0-5 V output, infinite resolution, and response time less than 1 ms. A few 0-50 psia transducers were also used.

Although the pressure transducers measured absolute pressure, the local atmospheric baseline pressure was subtracted from the outputted data traces, so that they were gauge pressure values. For some of the explosion tests, the static pressure pulses exerted on each stopping were measured by interpolation of the data from the two nearest C-drift pressure transducers, one inbye and the other outbye the crosscut position. Additional pressure transducers were installed on the C-drift (explosion side) side of the stopping in crosscuts 6 and 7. These transducers were suspended approximately 0.45 m from the mine roof and were located about 0.3 m in front of each stopping. These transducers were positioned perpendicular to the stoppings. The pressure data recorded by these transducers measured the total pressure (combination of static and dynamic pressures) generated on the stoppings during each of the explosion tests. A similar transducer was also mounted from the mine roof ~ 3.3 m behind each stopping on the non-explosion side (B-drift). These B-drift transducers were positioned parallel to the stopping or perpendicular to the explosion path thereby recording only the static pressures caused by any gas flow or air displacement.

The data gathered during the explosion tests were relayed from each of the data-gathering stations to an underground instrument room off C-drift and then to an outside control building.

A high-speed, 64-channel, PC-based computer data acquisition system (DAS) was used to collect and analyze the data.

This system collected the sensor data at a rate of 1,500 samples /s over a 5 s period. The data was then processed using LabView software and presented in graphic and tabular format. The reported data were averaged over 10 ms (15 point smoothing). This PC data analysis system allows the data traces to be expanded in time and pressure (or other sensor value) so that the peak values can be read and recorded precisely. Figure 12 shows a pressure sensor mounted on the C-drift side of the Flex-Stop stopping in cut-through 7.



Fig 12 – Pressure Transducer

CONSTRUCTION OF STOPPINGS

Two flexible stopping designs were constructed in cut-throughs 6 and 7 between B and C drifts at LLEM. These stoppings were designed to withstand overpressures of 14 kPa (2 psi) and 35 kPa (5 psi), the higher pressure stopping being installed in cut-through 6 and the lightweight design likewise in cut-through 7. The stopping in cut-through 6 was located ~ 1.7 metres towards C drift as measured from the center of the cut-through; the cut-through 7 stopping was located ~ 2.4 metres towards C-drift.

Alternatively, the crosscut 6 stopping was located approximately 4.7m into the crosscut as measured from the explosion side entry (C-drift); the crosscut 7 stopping was approximately 3.8m into the crosscut. Crosscut 6 had an average height of 2.17m and width of 5.25m (as measured between the rib slot positions); crosscut 7 had an average height of 2.24m and width of 5.12m.

In October 2003 contractors had shotcreted roof to floor rib slots into both stopping sites using the dry application process. These slots were of 20 mm in width and 200 mm in depth. The roof section of each site had been smoothed with shotcrete. The floor of the roadways was a pad of reinforced concrete laid onto a gravel base overlying the limestone. Figure 13 shows the B-drift side of cut-through 6 stopping cloth being wedged into the shotcreted slot.

The cloth used in each stopping was Fabric D as tested at UOW. The cloth was pre-cut to an appropriate size depending on the dimensions of the opening being used. Each stopping was pre-assembled on the floor. The sewn edges of the cloth forming the roof and the floor formed a loop into which was inserted a steel pipe. The cloth along with the inserted pipes was fed into the slotted RHS steel sections intended to hold the roof and floor sections of each stopping. Each RHS section (top and bottom) had 6 evenly spaced pipes welded to provide a means to anchor the RHS section to the mine roof and floor using 25 mm diameter by 660 mm length resin bolts.

These bolts were embedded 560 mm into 35 mm diameter drill holes and were fully encapsulated. The roof to floor span of the cloth was intentionally oversized so as not to create a pretensioned surface. The cloth material was anchored to each rib by inserting it into the slot and hammering in wooden wedges at regular intervals. Refer to Figure 13. Minova's thin sprayable liner, Tekflex, was then spray applied to the entire stopping periphery. The equipment used for the spraying consisted of a mixer, pump, hoses and a 40 cfm compressor to supply air to the spray nozzle, all in a self contained module.



Figure 13 – Rib Strengthening

Following the first explosion test (# 359), 4 (for cut-through 7) and 5 (for cut-through 6) equally spaced 25 mm diameter by 400 mm length resin bolts were installed through the cloth material and into each rib (embedded 300 mm) on the C-drift side of the rib slots. Figure 14 shows the finished stopping in cut-through 6 and Figure 15 shows the finished stopping in cut-through 7. Note the curve in the roof profile of cut-through 7 shown in Figure 15.



Fig 14 – Finished stopping cut-through 6



Fig 15 – Finished stopping cut-through 7

The stoppings were constructed under conditions analogous to those encountered in an underground coalmine. Because the concrete floor slab in each cut-through had been laid on gravel its stiffness would influence the ability of the stoppings to resist the explosion loads. The under floor aggregate was removed under each bolt hole and replaced with a slurry of gunite. The underground air temperature during the test period was around 11.1° C (52° F) and relative humidity in the range 76-90%.

EXPLOSION TEST RESULTS

A summary of the four explosion test pressures is presented in Table 1, which lists total pressures measured on the C-drift side of both stoppings. Please note that this data is indicative only pending the finalization of a report by NIOSH.

During the first test (# 459) a total pressure of 21 kPa (3 psi) was measured at the sensor immediately in front of the cut-through 6 stopping and 19.6 kPa (2.8 psi) at cut-through 7 stopping. There was little or no damage to the stopping with roof beams and rib slots completely intact.

As mentioned previously rib bolts and square washers were installed in both stoppings before test # 460. During this test a total pressure of 28 kPa (4.0 psi) was measured at the sensor immediately in front of the cut-through 6 stopping and 28.3 kPa (3.9 psi) at the same location in front of the stopping in cut-through 7. On some of the bolts the cloth sheared on three sides of the bolt plates. The rib slots essentially held the stopping rib sections of each stopping in place. The stoppings were essentially intact with no damage to the roof and floor beams and rib slots.

Test #461 resulted in total pressures of 25.9 kPa (3.7 psi) and 25.1 kPa (3.6 psi) on the stoppings in cut-through 6 and 7 respectively. There appeared no further additional damage to either of the stoppings from that observed after test #460.

Test 462# resulted in total pressures of 42.7 kPa (6.1 psi) and 37.8 kPa (5.4 psi) on the stoppings in cut-throughs 6 and 7 respectively. The stopping in cut-through 7 failed when the slot in the roof RHS opened up under the high tension loads on the cloth, letting the cloth go in the central roof portion. This was not an unexpected result for a 14 kPa designed stopping.

The stopping in cut-through 6 failed after the central bolts holding the RHS beam bent and snapped dislodging the upper portion of the stopping.

In all tests the cloth remained intact where it was held within the rib slots. The RHS beams comprising the 35 kPa stopping design did not release the cloth from their slots during the four explosion tests.

Table 1 – Summary of four explosion test pressures conducted at LLEM

Total Pressure kPa (psi)	# 459	# 460	# 461	# 462
Cut-through 6	21.0 (3.0)	28 (4.0)	25.9 (3.7)	42.7 (6.1)
Cut-through 7	19.6 (2.8)	27. 3 (3.9)	25.1(3.6)	37.8 (5.4)

CONCLUSION

Explosion resistant stoppings such as those evaluated in this paper provide protection for coal mine personnel and assets by isolating them from the effects of an explosion that might occur within the workings.

The primary objective was to develop and test a flexible stopping system that would satisfy the requirements of the Queensland Department of Natural Resources and Mines “Approved Standard for Ventilation Control Devices” and to satisfy the requirements of “Coal Mining Safety and Health Regulation 2001, Qld”.

The results achieved in both the static test program and the explosion test program have provided confidence to Minova Australia that the Flexi-Stop system satisfies the legislative requirements. The results provide the opportunity of testing to further improve and optimize the Flexi-Stop stopping designs.

Initial testing has shown that even with the use of high tensile and tear resistant cloths point load failure will still occur when steel bolts and plates are used as the fixing medium. To optimize the load bearing capacity of any stopping system using cloth, it is necessary to develop a fixing system that enables the superimposed loads to be more evenly distributed through the cloth. The RHS beam and internal pipe arrangement developed (patent pending) as part of the flexi-stop system has proven to evenly distribute load on the cloth between the roof and floor and has made 2 and 5psi stoppings possible.

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LONGWALL MINING THROUGH FAULTS AT MORANBAH NORTH

Chris Hanson¹, Brett Moulle², Chris Strawson³, and Andrew Laws⁴

ABSTRACT: The coal mining industry has had to contend with the hazard of geological structure since earliest history. In recent years, there has been considerable success in mining through faults of less than seam displacement, generally consisting of relatively simple structures with a limited zone of influence. Generally, longwall mining through fault structures of greater than seam displacement has been less successful. At Moranbah North, recent experience has demonstrated that both large displacement and structurally complex faulting should not present a barrier to successful and economic development and longwall extraction.

Moranbah North operates its longwall mines in relatively weak strata with varying geological conditions. A fault zone consisting of a series of structures with a combined total displacement of up to 7.5m was mined through in Longwall Panels LW102, LW103, and LW04. Considerable production delays occurred in LW102 associated primarily with maingate end issues. Lessons from this experience have been incorporated in planning and operation to achieve successful mine throughs without significant production delays in LW103 and LW104. In LW103 and LW104, a detailed geological model was developed from surface to seam and in seam drilling and 3D seismic exploration. This also allowed development of an accurate 3D picture of the faulted areas in mid panel. Flight plans detailing the optimum cut horizons to maintain a practical grade, minimise total stone cut, and maintain roof coal beam thickness were incorporated into underground operational fault management plans. In addition to this, installed support was considerably upgraded to ensure stability.

Overall, the decision to mine through the fault in LW103 and LW104 has allowed substantial savings (compared to a longwall relocation around the fault) and minimised resource loss. The ability and confidence to mine through greater than seam displacement structures or zones of structure will continue to be of significant benefit to Moranbah North in maintaining high production output and cost effective resource recovery across the lease.

INTRODUCTION

Located in the Bowen Basin of central Queensland, Moranbah is 200km west of Mackay (Figure 1). Moranbah North is a modern, high capacity longwall mine, operated by Anglo Coal (Moranbah North Management) Pty Ltd and supplies premium quality high fluidity hard coking coal to the export steel market.

Moranbah North to date has mined out four longwall panels LW101 – LW104 shown Figure 2, varying in total length from 2.3 km (LW101) to 3.7 km (LW104). All longwall panels retreat up dip, mining from south to north. Coal is extracted from the Goonyella Middle Seam (GM seam), which varies in thickness from around 3.5m to 6.0m. Longwall extraction aims to maintain a minimum of 1.0m to 1.5m of coal in the immediate roof to maintain stability. Typical heights of extraction vary from 4.0m – 4.6m thick.

An extensional fault zone comprising normal displacement faults with combined displacements varying between 4m – 7.5m has been encountered and mined through in longwall blocks LW102 – LW104. Following a roof fall around the fault zone at the maingate end of LW102, subsequent maingate and tailgate ends have been comprehensively supported with a combination of secondary support (cable bolts, pre-tensioned long tendons, and steel sets.), shuttering and grout pumped false roof, and PUR injection from the maingate end. In addition, a detailed geological model was developed for LW103 and LW104, using surface, and in seam drilling, and 3D seismic exploration. This facilitated the implementation of:

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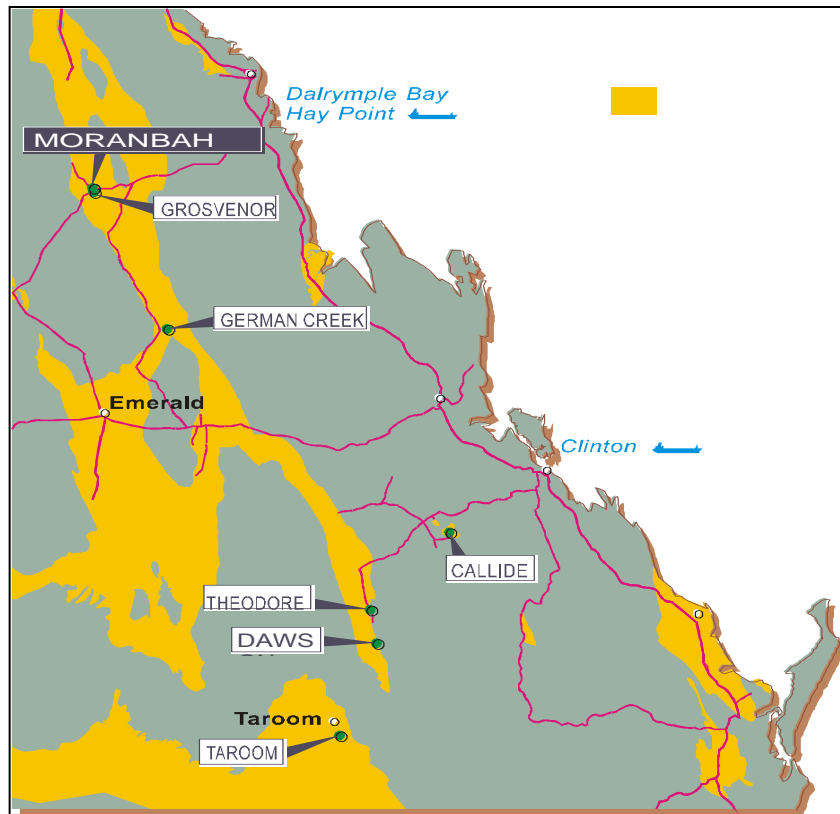


Fig 1 – Moranbah North Location Map

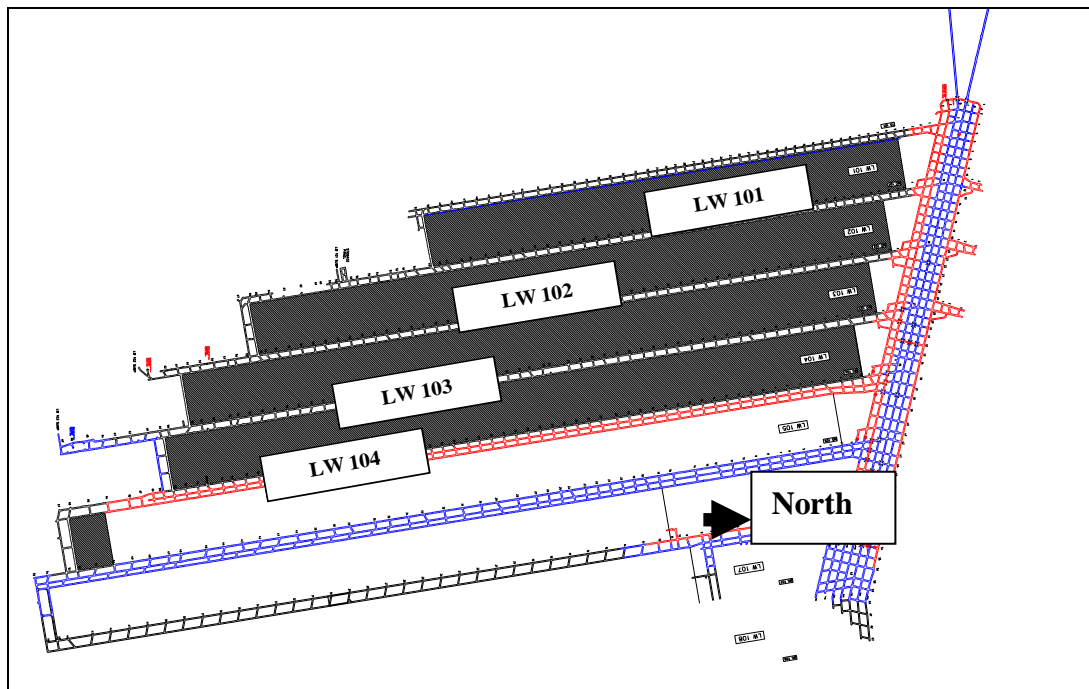


Fig 2 – Moranbah North 100's series longwall panels (101 – 104 shown hatched / longwalls extracted to date).

- Accurate positioning of seam drilling, spiling and pressurised microfine grout injection into the immediate roof of the planned extraction horizon.
- Accurate development of proposed longwall drivage sections (flight plans), detailing the optimum cut horizons to maintain a practical grade, minimise total stone cut, and maintain an adequate roof coal beam thickness; and
- Crew presentations and an operational underground Fault Management Plan, with associated triggers and responses, based on the operational, technical, and maintenance requirements for fault drivage.

This paper details the depth of up front technical and operational planning, and control required to achieve successful longwall retreat through significant (in excess of seam displacement) faulting in the GM seam.

A chronology of fault mine throughs is presented. Review of previous fault mine throughs in each case has allowed lessons to be learned and refinements to be incorporated into future fault mine throughs.

Above all, it is stressed that the successful fault mine throughs at Moranbah North have only been achieved through a collective team effort involving:

- Underground operators, emergency response zone (ERZ) controllers, Shift Managers, production staff and management at Moranbah North through implementing and complying with the Fault Management Plans and installation of required ground support;
- Moranbah North Technical Department for designing flight plans from geological plans, secondary support and spilling design, providing daily underground reconciliations of as mined vs predicted fault positions, and compiling crew presentations and Fault Management Plans;
- Anglo Coal Brisbane Coal Office (BCO) for assistance with exploration and fault modelling based on 3D seismic, in seam and surface drilling, and for providing assistance with ground support design and spilling requirements; and
- External Consultants in various capacities including Geowork Pty Ltd, Strata Engineering (Australia) Pty Ltd, and the CSIRO for micoseismic monitoring through the LW103 fault drivage.

HISTORY

Longwall 102

In mid 1999, development operations in panel 102 unexpectedly encountered a 3.8 metre displacement fault set, downthrown inbye. The zone had not been interpreted at this location from geological modelling despite a 250 metre exploration borehole grid. Some development delays were incurred to regain horizon, install adequate support and concrete critical roadway surfaces.

The longwall mine through is detailed later, however, a failure occurred, which delayed the longwall by 18 days. Back analysis of the delays highlighted the need for improved accuracy of the model, and improved support strategies. Figures 3 and 4 show details of the failure area.

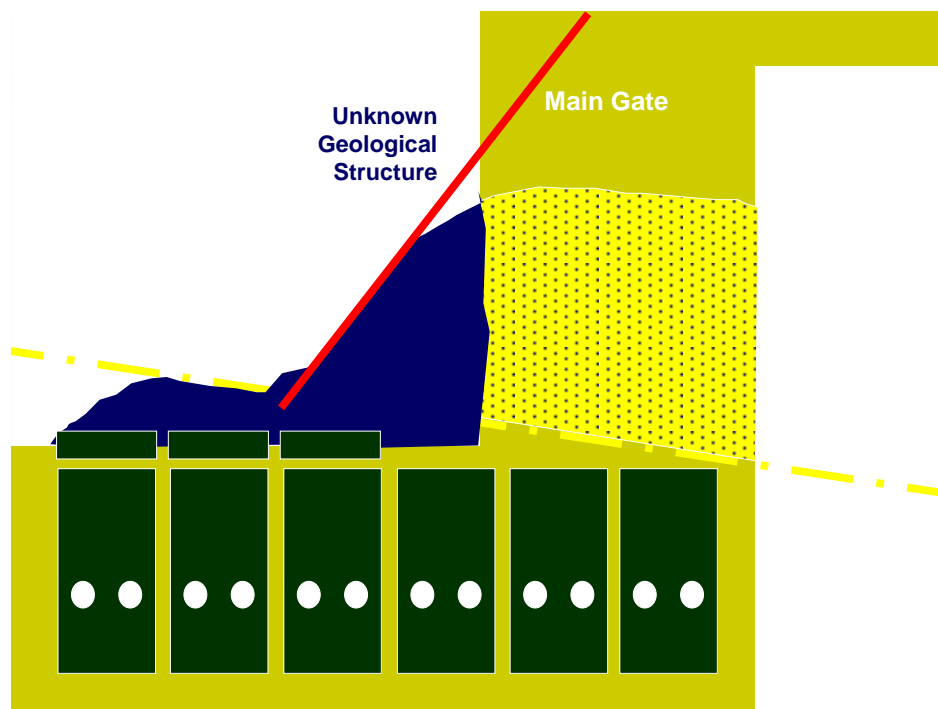


Fig 3 – Plan view of LW102 maingate end roof failure

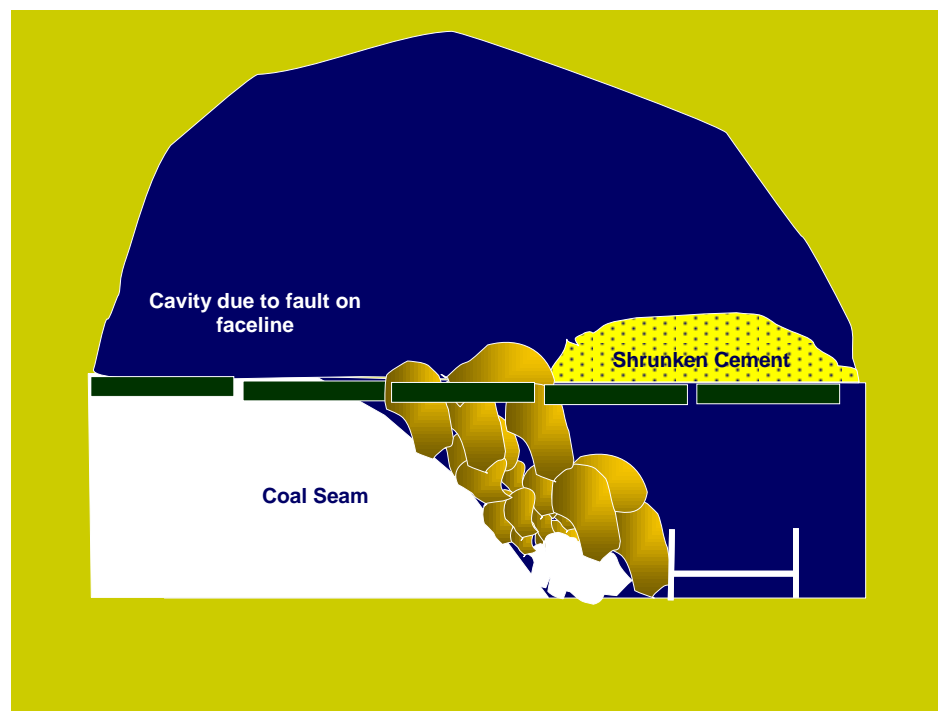


Fig 4 – Section view of LW102 maingate end roof failure

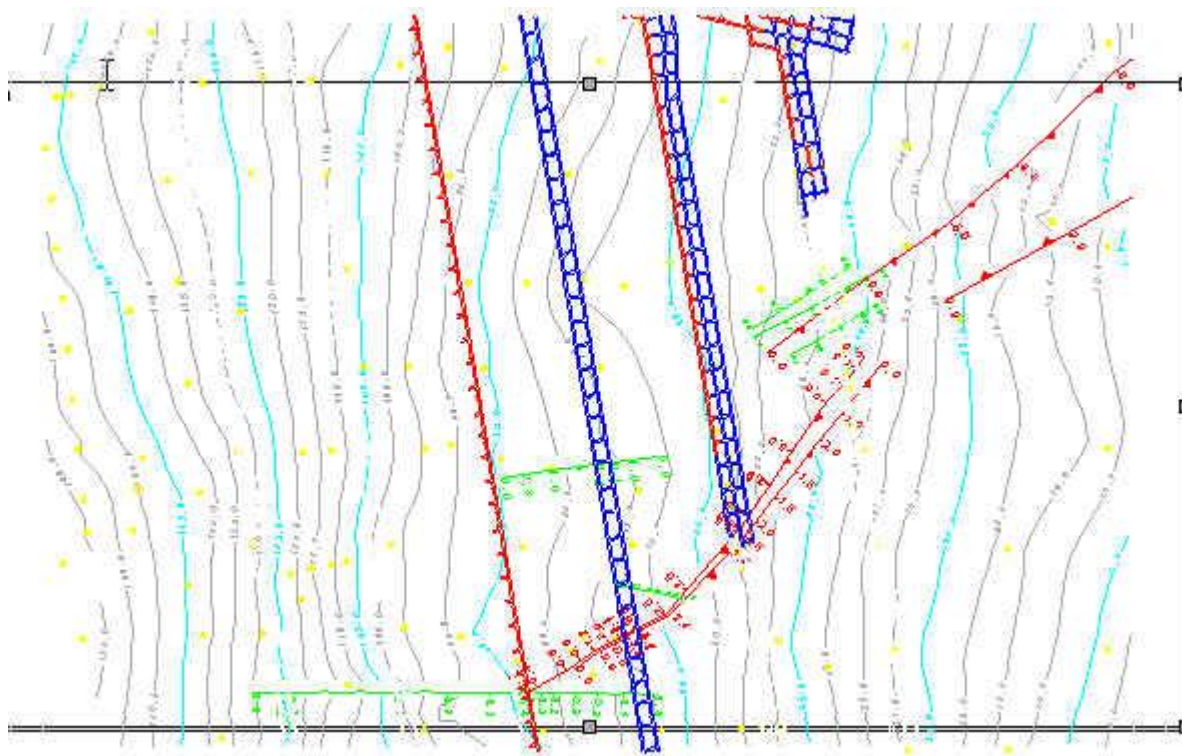


Fig 5 – Maingate 103 fault interpretation following seismic, in seam and surface drilling

Longwall 103

In mid 2000, development in maingate 103 encountered a 5m total displacement fault zone in a location outbye of that predicted (projected from 102). The fault zone consisted of a splayed series of up and down thrown faults with significant displacement (4.9m belt road, 6.7m travel road). The fault intersection away from the predicted location once again highlighted the variable geology and constraints associated with fault prediction in the GM seam from surface drilling and geological modelling only, without utilisation of more sophisticated exploration tools.

In order to avoid results, similar to those experienced in LW102, in future longwall panels, detailed additional assessment and analysis work was completed, including;

- Inseam drilling to confirm fault location and floor RLs;
- 3D seismic exploration and interpretation;
- Detailed roadway re-mapping of the fault zone (see Figure 4);
- Roof and floor drilling in both maingate and tailgate to confirm development roadway roof and floor coal thicknesses;
- Detailed geological mini-model generation on a 1 metre by 1 metre grid;
- Review of numerical modelling for panel 102 mine through;
- Review of panel 102 microseismic monitoring results and conclusions;
- Re design of the support requirements, cross block, as well as in the gateroads.
- Review and confirmation of operational horizon control system utilised; and
- Review of the operational aspects and associated issues with LW 102 fault mine through.

Figure 5 shows a structural interpretation following the use of more sophisticated exploration techniques including seismics, in seam and surface drilling to produce a more detailed and confident geological structural interpretation. Figure 6 shows the detailed mapping following development while Figure 7 an interpretation of the gradient changes.

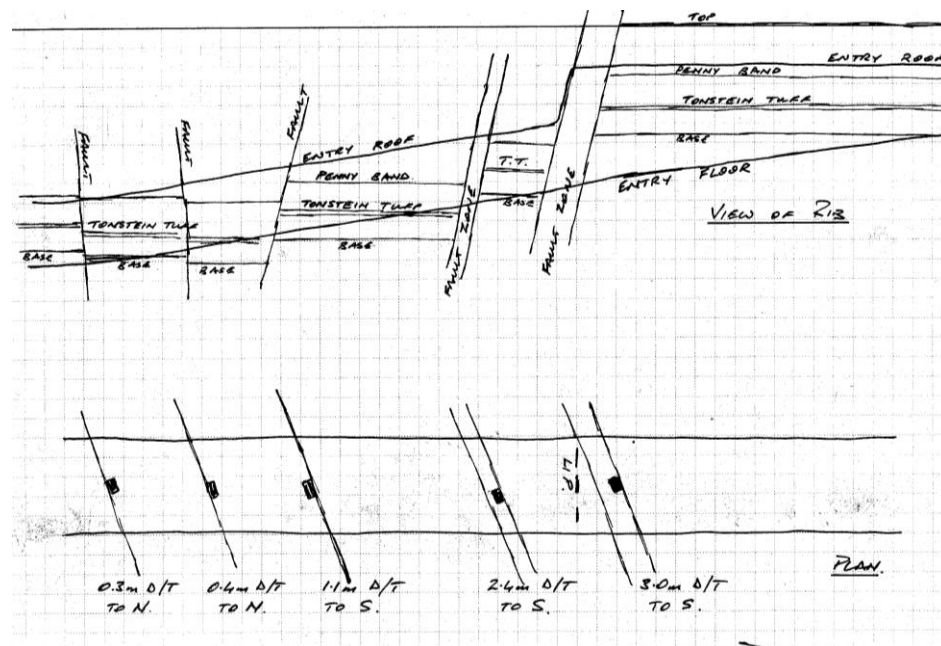


Fig 6 – Maingate 103 mapping of maingate fault system

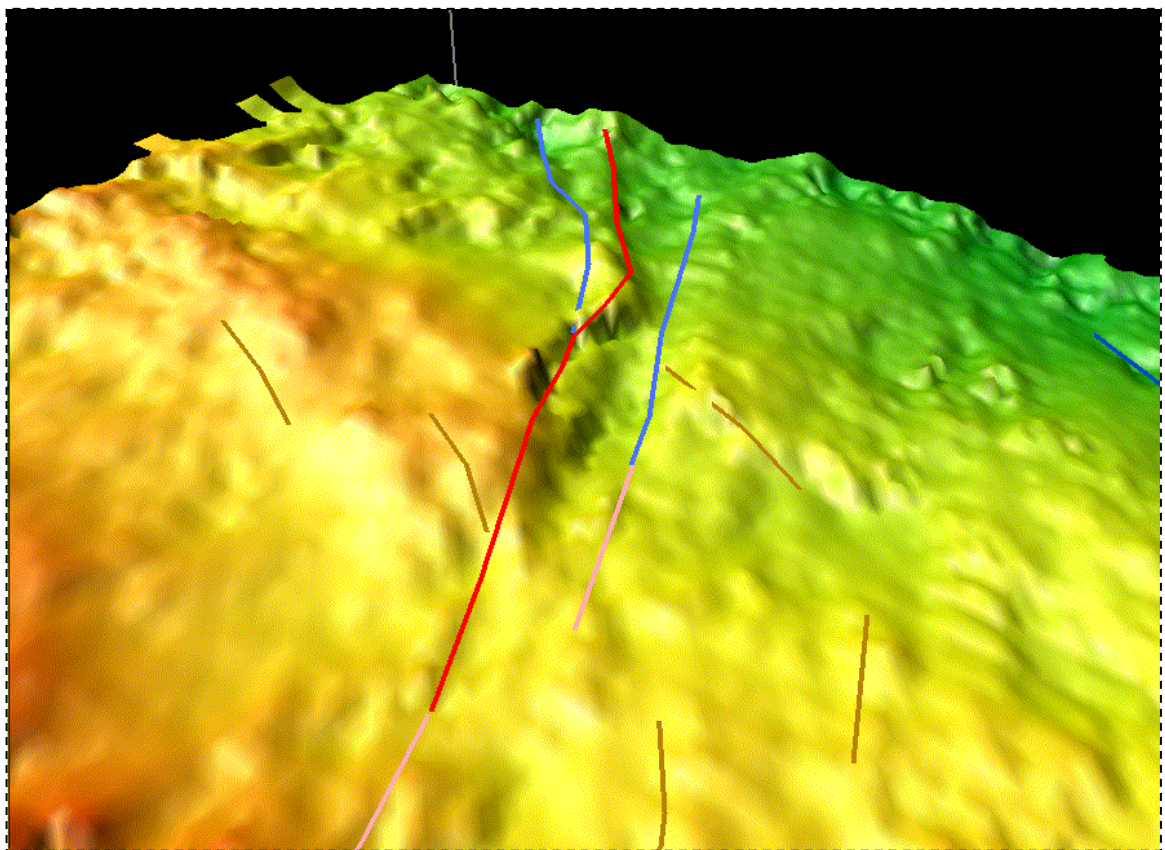


Fig 7 – Seismic (dip gradient change) interpretation of LW103 fault zone

Assessment indicated that the horizon control system used in panel 102 was adequate and appropriate, though more disciplined supervision was required. The ground support strategy implemented for LW103 was aimed to address the failure mechanisms assessed from LW102. Significant additional support beyond that used in panel 102 was installed in both gateroads

Flight plans were developed from the improved geological model, with proposed cut horizons based on:

- Optimising the coal beam roof thickness;
- Minimising the amount of stone cut (reducing dilution and potential damage to equipment);
- Maintaining a practical working grade across the face to conform to the longwall equipment limitations.

In LW103, the plans and sections generated from the geological model proved to be very accurate when reconciling predicted vs actual geological structure mapped on the longwall face. Longitudinal and transverse sections through the zone were used as guidance and provided a visual direction on anticipated ground appearance. This allowed the crews to confidently grade through the fault, with horizon control monitored and recorded, through the use of inclinometers. Figure 8 shows a modelled interpretation and Figure 9 a flight plan developed from the geological model

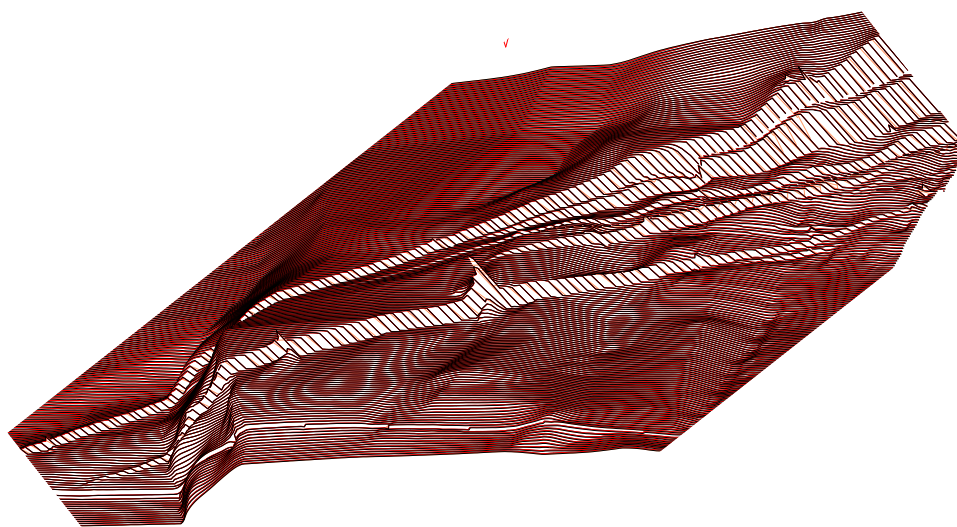
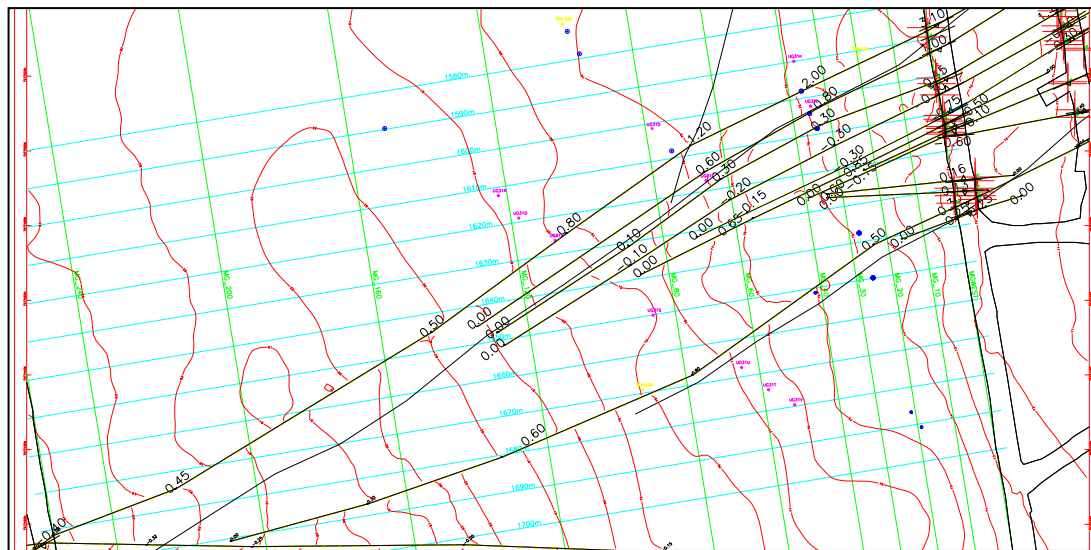
Microseismic monitoring

Two existing microseismic monitoring geophone strings in the panel were recommissioned. The microseismic monitoring was used to measure any potential reactivation of the fault zone as the longwall face approached the area. A system of daily microseismic monitoring, recording and plotting of events in plan view was coordinated on site through the site Technical Department and the CSIRO. Daily plots were distributed to crews for information.

The microseismic monitoring indicated few events located around the fault zone itself, which is likely considering the inability of faults to transmit shear forces across the fault plane. Most significant microseismic events were located 100m – 300m inbye of the longwall face, biased towards the maingate end. Figure 8 shows a typical microseismic plot with the size of event indicated by the size of the circle.

Some mapped outbye faults were re-activated at distances in excess of 500m outbye of active mining. The CSIRO have observed similar behavior at other mine sites.

Results of the mine through indicated that the support strategy was very effective in that no significant fault plane reactivation, roof or rib failure occurred. No movement was recorded above or within the horizon of the false roof. Some delays were experienced due to maingate end equipment failures and soft or broken floor areas, where shields had to be packed to assist elevation of the front of pontoons to negotiate grade. In addition, the floor of the maingate proved to be soft, which allowed the BSL to bury into the floor.



**Fig 8 – Modelled fault zone with survey flight plans and ECS modelled grid (bottom)
– model gridded and interpreted by S. Argent (Anglo Coal Australia Pty Ltd)**

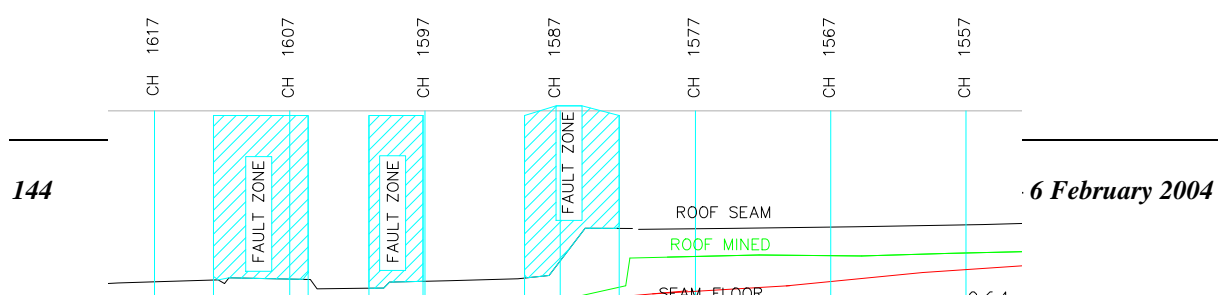


Fig 9 – Flight plan based on geological model (LW103) – A.Varvari

- Ensuring that the maingate roadway provided by development is on grade (for fault driveage), and adequately dimensioned for ease of longwall retreat

outbye around main fault zone and significant microseismic activity inbye

Longwall 104

Based on the success of the LW103 mine through, a similar approach was adopted for the LW104 mine through.

For development (maingate) panel 104, additional exploration drilling and fault interpretation was used to facilitate a pre-planned longwall mining horizon at the maingate end and drive the development roadways on pre-planned grade. This exercise proved very successful in providing a practical grade for longwall retreat mining through the maingate end and eliminating the requirement for installation of a false roof in future maingates.

Geological structure was modeled in the same manner as the previous (LW103) fault mine through and was based on surface and in seam drilling, exploration, mapping, accurate resurvey in the gateroads, and 3D seismic interpretation. The final geological model is illustrated in Figure 11. The LW104 fault zone was found to be greater than the seam displacement over a significantly greater length than LW103 fault zone. A fault zone with 6m – 7.5m total displacement was predicted to run across the entire block over a lateral retreat distance of some 270m.

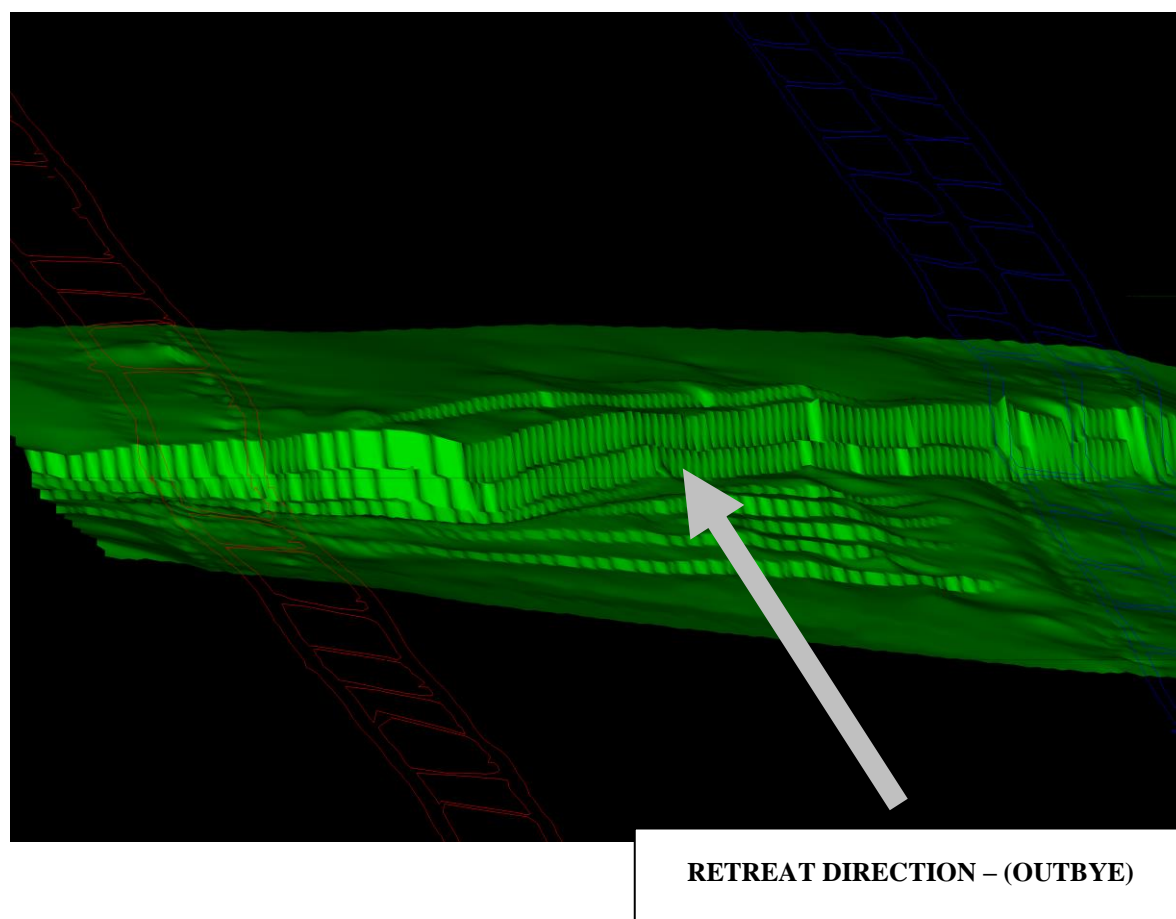


Fig 11 – Modelled fault zone for LW104 – model gridded and interpreted by S. Argent (Anglo Coal)

Flight plans and a detailed Fault Management Plan were compiled in the same manner as for LW103. On site Technical Services personnel mapped the face on a daily basis throughout the fault zone to reconcile the position of actual geological structure against predicted, determine the actual longwall flight paths against design, and advise on corrective action and adjustments to maintain correct flight paths where appropriate.

The actual vs predicted fault locations proved to be reasonably accurate as illustrated in Figure 12.

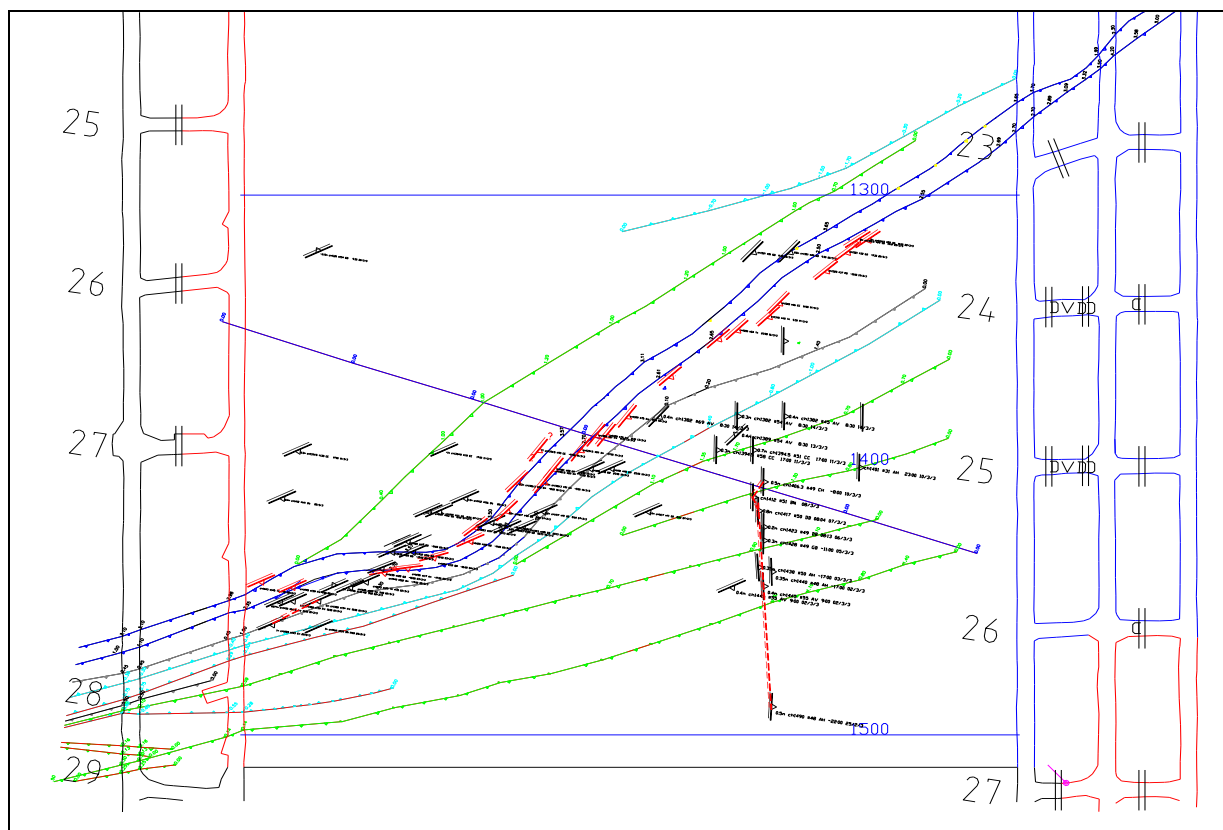


Fig 12 – Actual vs predicted fault structures – LW104 fault zone

Summary – LW104 mine through

As with the LW103 fault mine through, the LW104 fault mine through was successful, particularly in the following regard:

- The planned fault drive through during development included having the belt road at a pre-determined grade. Concreting the floor of the belt road through the fault affected area facilitated easy negotiation of all longwall equipment through the main gate end;
- The fault mine through was safe with no LTIs;
- Minimal strata control issues on the face (one minor roof fall in an area towards the main gate end also associated with horizon control, and soft floor.) during fault mine through;
- In general the cut horizon was maintained to plan and the flight plans and fault management plan was followed. Towards the main gate end a kink developed across the face resulting in stone cutting greater than planned on the main gate side of the fault. However, good work by the crews ensured that this was quickly corrected and evened out by the time the main fault zone had tracked through to the main gate; and
- There was no unexpected mid panel geological structure – the geological plans and sections proved accurate.

SUPPORT AND MONITORING

Summary

The support installed for mining through the fault at Moranbah North, has been modified, and grown, driven by lessons learnt on previous longwalls. In most cases the installation of the support, and the logistics has been more complex, and arduous than initially planned. This has for the most part been due to the support having to fit 'around' the operational constraints of a producing longwall. In future panels where support is required, the installation is planned to be well ahead of the longwall.

An overall monitoring plan has not been used during the respective mine throughs, however, data obtained to date indicate stable maingate, and tailgate conditions, once the correct levels of support were achieved.

LW102

Small throws, and tight faulting in the tailgate of LW102, necessitated only primary bolting, and the use of 6.1m point anchored Flexi Bolts on a 2.2m spacing.

The larger overall displacement, and closer spacing of the faults in the maingate, required additional support. Additional 1.8m roofbolts, with mesh, and point anchored Flexibolts on a 1.8m x 2.2m spacing were used. There was no stabilisation, or consolidation of the fault over the block.

During mining of the last 8m of the fault zone in the maingate, a failure of the rib along the jointing, (running sub parallel to the gate road), allowed a large fall to occur in the maingate, and on the face, see Figure 13. A significant quantity of material had to be removed from this area, and blasting of the stone in the fall was required.



Fig 13 – Open cavity above the gate end supports at the maingate LW102

The grouted false roof, installed to allow the maingate chocks to push forward, came loose, and added to the fall. In addition, the cut height on the face, was 750mm lower than planned, mainly due to soft floor. This exposed more block side rib, possibly assisting in the rib failure. 18 days were lost due to this failure.

The position, and throw of the faults resulted in changes in grade on the face, which in some instances were greater than able to be accommodated by the longwall equipment, and damage to dog bones and other equipment resulted.

Instrumentation

Although Tell tales were installed in the maingate, in the immediate vicinity of the fault, it was not possible to install monitoring devices. The readings on the outbye instruments were normal, and did not provide any warning of the failure that occurred in the maingate.

Learning points:

- Stability of the maingate block side rib is crucial where the faulting intersects the gateroads.
- The provision of an accurate fault 'Flight Plan' is vital to success in mining faults. It is important that all the longwall equipment parameters are fully known and understood. (i.e. in some cases manufacturers figures bear re checking)

LW103

The lessons learnt in LW 102 were to a large extent applied to the fault mine through in LW103. These consisted of:

- Great improvement in the prediction of the position, and nature of the faults / Fault Zone.
- This improvement allowed for confidence in the placement of grouted steel spiles over the block, to stabilize the fault zone. 30mm thick walled boiler pipe was installed in parallel pile holes, drilled in the block, over the fault zone.

A total of 34 holes, varying in length from 15m to 130m and 56mm diameter were drilled at 90 degrees to the roadway. The holes were spaced approximately 1.5m apart, and once the steel pipe had been inserted, were pressure grouted using microfine cement.

The drilling of the holes over an operating longwall belt shown in Figure 14 proved to be difficult and slow. One of the contractors drill steels caused considerable damage to the longwall belt.



Fig 14 – Drilling of the splice holes over the longwall belt. LW103

- No permeability tests, or measurements of the grout quantities were taken, however, all holes were pumped to refusal, and when the area was mined through, grout was visible on the face.
- The injection of microfine cement into these holes provided additional stabilization of the broken ground.
- Due to the unexpected early intersection of the fault in the maingate, the height of the maingate again necessitated the installation of a false roof. This installation was a considerable improvement on that installed in LW 103.

- The support in the maingate was increased. 14m fully grouted, and trussed minicage cables installed in the 'high area', as well as inbye the fault zone. These cables were adequate to provide stability to the maingate during retreat. The support system is illustrated in Figure 15.

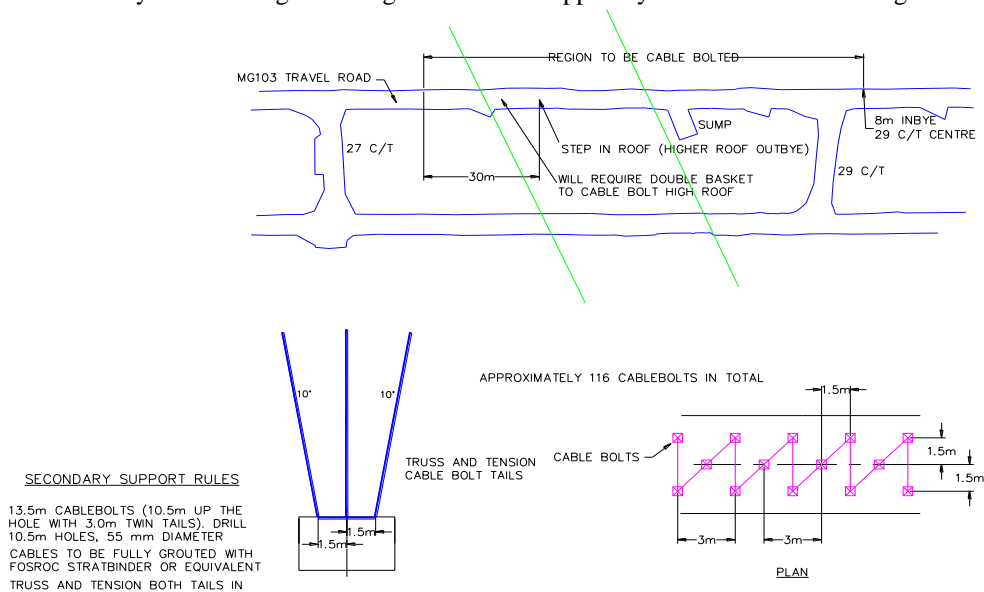


Fig 15 – Maingate cable support in Fault Zone

- In addition to the cable support, it was decided to install steel sets on a 1.2m spacing through the fault zone. These were supported on steel legs on the pillar side, and were bolted into the rib, and supported on timber legs on the block side as illustrated in Figure 16. The steel sets provided the base for the false roof.



Fig 16 – Installing steel sets in the maingate LW103

- The block side rib was reinforced, using PUR injection, and cutable tendons.

- The tailgate received additional support in the form of fully grouted, and trussed cables across each fault, and closely spaced Link and Lock standing supports. A 1.4 x 1.4m Link and Lock was especially produced for installation in the higher areas.
- Inclino-meters as shown in Figure 17 were used on the face (Pan line), allowing the crews to have a simple form of assessment of the attitude of the longwall equipment.



Fig 17 – Magnetic Inclinometer attached to Bretby. Panline LW103

Instrumentation

- A surface extensometer was installed just outbye of the main fault zone with a number of anchors set at various levels in the overburden. The purpose of this surface extensometer was to assess the relative strata movements of the anchors through the overburden. As such the surface extensometer was used as a post analysis tool, rather than a monitoring device capable of triggering a response during mining.
- Tell Tales were installed at various locations in the roof and rib at the maingate end (belt road). The main area of failure in LW102, was the 'False Roof', Tell Tales were therefore installed in this area, and routinely monitored by ERZ Controllers as the longwall retreated through the fault zone. Little or no movement was recorded on the Tell Tales. There were areas where some bagging of the roof occurred, however the roof support ensured stability throughout the fault zone.
- Cross panel and centerline (down block) surface subsidence surveys were undertaken on a frequent basis to assess the impact of the fault zone on the surface subsidence profile. Figure 18 shows a centre line subsidence plot showing the impact of the fault zone on the subsidence profile through time and increased face retreat distance. Surface subsidence over the fault zone is shown in Figure 19.

The increased relative level of subsidence recorded inbye of the fault position at the tailgate end is clearly visible. It should be noted that this area is in close proximity to the roof fall in LW102 MG, where a substantial quantity of material was removed.

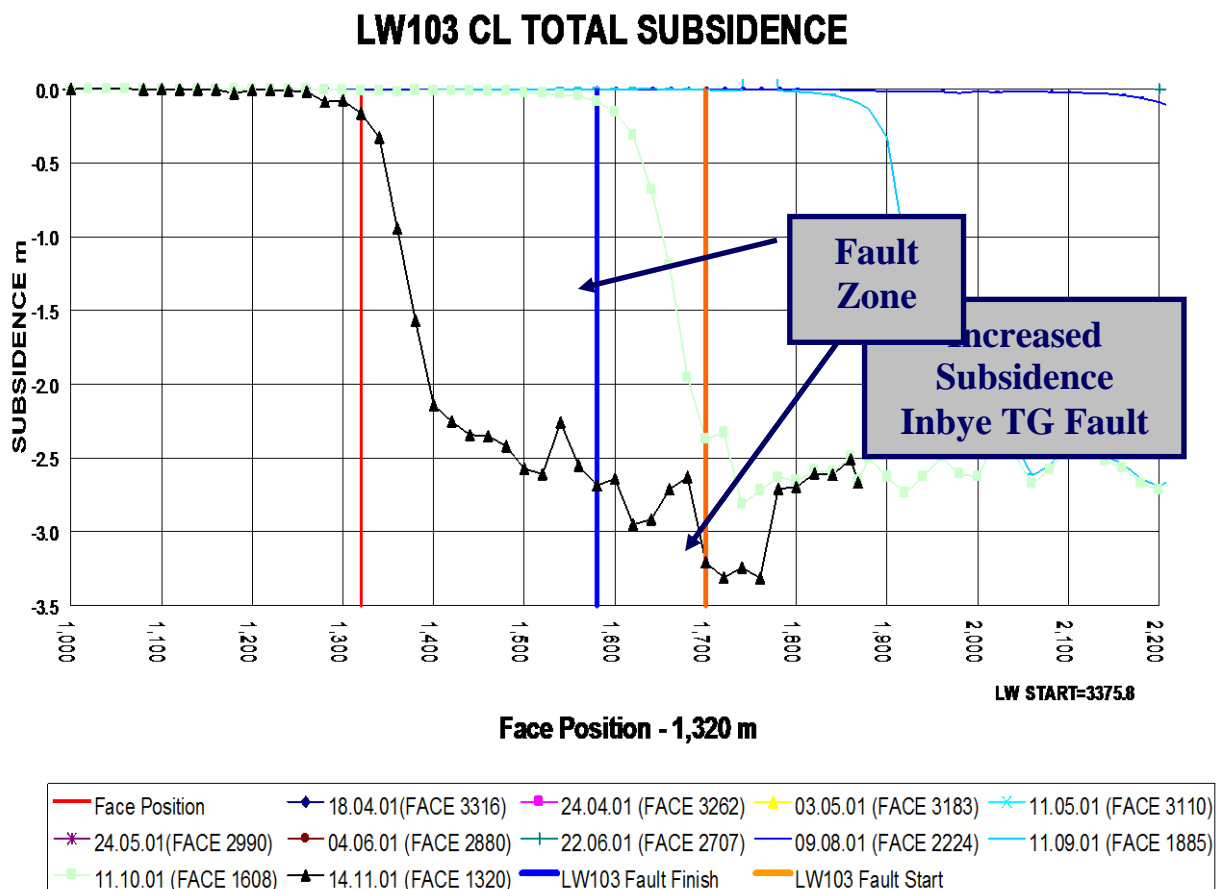


Fig 18 – Centre line surface subsidence plot LW103

Outcomes

- Safely mined through the fault zone, no LTI's, or major damage to Longwall equipment.
- Average longwall production through the faulted zone, 93 500 tonnes per week.
- No support issues in the tailgate, or over the block. No strata control issues on the face during fault mine through, evidence of pressurized grout permeation was visible on the longwall face and this is illustrated in Figure 20.
- Ground support at the maingate and tailgate ends performed well with monitoring indicating adequate ground control.
- The BSL dug into floor in fault 'Ramp up zone', causing delays.



Fig 19 – Surface subsidence over fault zone – LW103

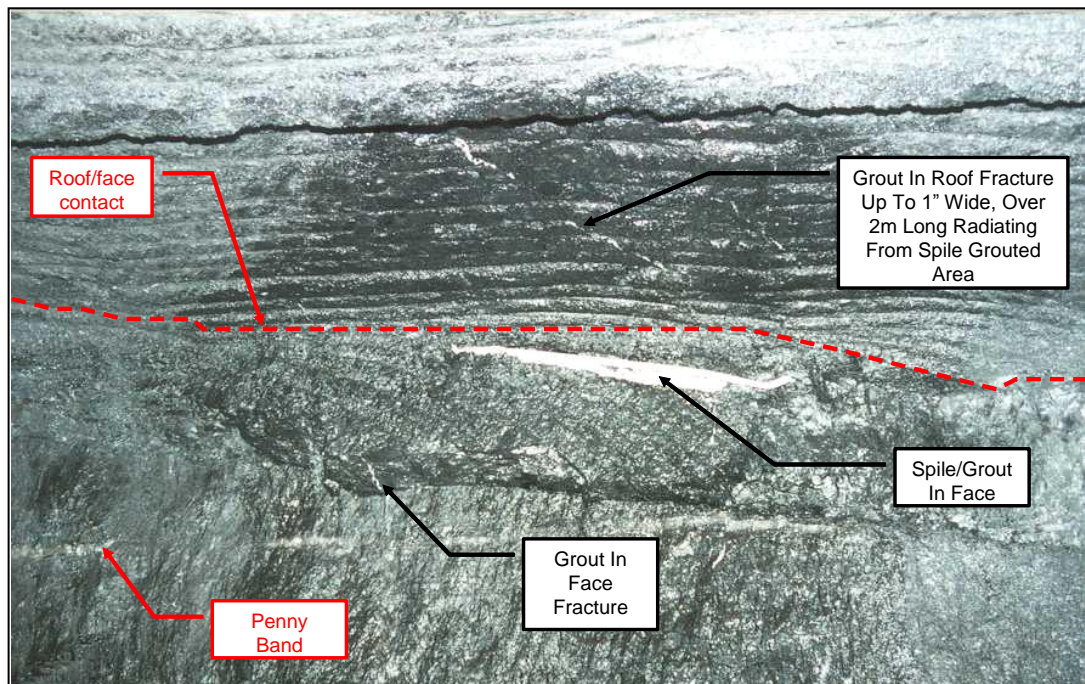


Fig 20 – Grout permeation in face due to pressurized grout injection – LW103

Learning points:

- Soft floor in fault zones, can cause as many problems as soft roof.
- In areas of anticipated dense faulting, and broken ground, it is advisable that some form of 'Over the Face' stabilization be undertaken.
- The accuracy of the geological 'Fault Plan' is important in ensuring that the planned flight path for the longwall equipment is well thought out, and useable
- Maintenance of Longwall Equipment prior to entering the fault zone is important.
- Steel sets may become entangled in the longwall supports in confined areas.

LW 104

With the additional information provided by the successful mining of the fault, in LW103, it was possible to improve further on the support design for LW104. Unfortunately the additional support work had to be done in the tailgate, during normal longwall mining operations. This caused some delays. In addition, the support work done in the maingate, and over the block, were completed over an operating Longwall belt.

With the precise predictions of the fault positions, it was possible to accurately drill fans of spile, and grouting holes over the block, for steel spile, and grouting purposes.

As these holes were generally longer than those in LW103, it was decided to utilize directional drilling to prevent any chance of the longwall intersecting the holes in the roof. There are obvious cost implications, however the improved stability provided by grouted steel spiles in the immediate roof (approx 1m to 2m above the cut horizon) was well worth the expense.

A total 15 holes were drilled from the tailgate, and 38 from the maingate. Again, the holes were 56mm diameter, and 38mm thick walled tube was installed, and grouted. In this instance, the grout usage was monitored, however no permeability testing was possible due to time constraints. In an area identified as being 'soft' when drilled, it was possible to pump more than 500litres (more than hole capacity) of a Thixotropic grout into each of 5 holes. This would indicate voids in the fault zone were filled. The layout of spiled holes in the tailgate is shown in Figure 21 and in the maingate in Figure 22.

The support installed in the maingate consisted of :

- 13m bulbed cables, fully grouted and trussed, in a similar pattern to that used in LW103.
- 6.1m and 8.1m fully encapsulated, and pretensioned Hi Ten cables between the grouted cables, angled over the block . Approximately 40 Hi Tens were used.
- Bolting and meshing of the ribs (pillar, and block side) for a distance of 80m around the fault zone which was approximately 30m long in the maingate.
- Installation of 8m cutable dowels with PUR into the block side rib, to prevent the failure that occurred in LW102.
- A rib spray was applied, more from an interest point of view, to check the performance under friable conditions.
- A concrete floor, with steel guide rails was constructed in the maingate in the 'ramp up' zone.
- The tailgate rib (which was 6m high at one point) was bolted, and meshed on both the pillar and block sides.
- 6.1m long grouted cutable bolts were installed in the block side rib.
- The tailgate was fully cogged (Link and Lock), with extra large footprint cogs used in the high zone. The cogs were on an approximate 1m skin to skin spacing.

All the same systems of Flight Plans, inclinometers, and excellent fault crossing plans were used in LW104.

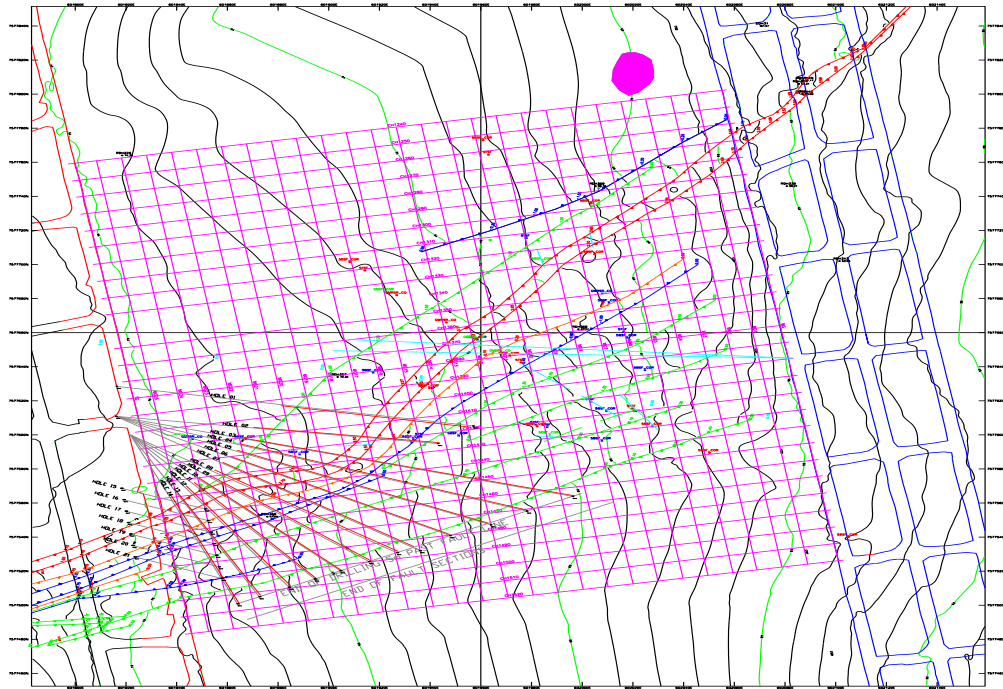


Fig 21 – Tailgate ‘Spile’ plan. LW103

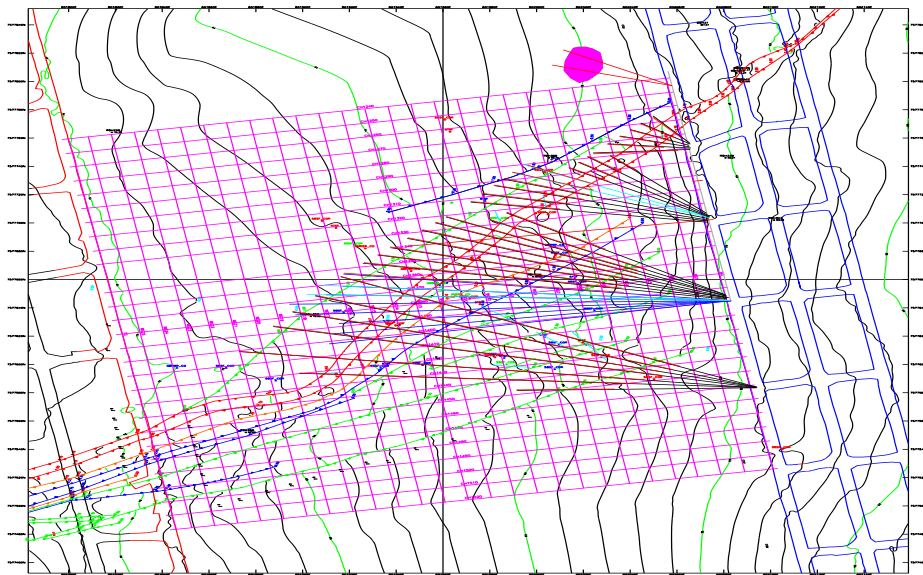


Fig 22 – Maingate ‘Spile’ drilling layout. LW103

Instrumentation

The only instrumentation installed in the fault zone, were Tell Tales on a regular pattern. The maximum displacements measured were 57mm in the Travel road, alongside the fault zone, and 93mm in the Maingate, just ahead of the wall. Generally stability was good.

Outcomes

- Fault mined through with no LTI's.
- Support all worked excellently. The ramp in the maingate worked well.

- Production through the fault zone over budgeted figures for the area. Actual approximately 84 000 tonnes per week.
- Actual faulting very similar to that modeled. Maximum errors approx 2m in position intersected.

Learning Points

- Soft floor conditions require care when negotiating. At one point the powered supports near the maingate, were low in the floor, but with care, were able to be raised to the correct cut level.
- The use of directional drilling proved to be expensive, however provided a good outcome.
- In order to improve the installation of support, and reduce costs, this work should be done well ahead of the face, and if possible, not in a producing tailgate, or over an operating maingate belt.

Summary

Significant faulting has successfully been mined through at Moranbah North primarily due to an excellent collaborative team effort by many people in various areas ranging from:

- Exploration, geological and technical assessment;
- Designing and installing required ground support and mid panel fault consolidation;
- Developing accurate flight plans and an appropriate mining strategy based on technical information;
- Effective communication and training of all responsible individuals in fault driveage strategy prior to implementation;
- Disciplined control and supervision underground by production, maintenance, and operational personnel; and
- Adopting an approach incorporating back analysis and continuous improvement systems.

CONCLUSION

The success of fault driveage to date at Moranbah North has proved that the magnitude of fault displacement alone is not a limiting factor to longwall retreat within the GM seam. Overall, the cost effectiveness of the decision to mine through the fault in LW103 and LW104 has allowed substantial saving (compared to a longwall relocation around the fault) and minimised resource loss. The ability and confidence to mine through greater than seam displacement structures or zones of structure will continue to be of significant benefit to Moranbah North in maintaining high production output and cost effective resource recovery across the lease.

NEW DEVELOPMENTS IN AUSTRALIAN COAL PRODUCTION

Ken Talbot¹

ABSTRACT: With Australia as the largest exporter of coal in the world, the challenge for the mine developers is to develop a mine which can be profitable. Coal reserves, market demand and capital, forms the basis for successful mining development.

INTRODUCTION

It is important that the fundamentals for a new mine development are right. The mine developer then has the best likelihood that the development will be economic.

Marshall (1974) stated that the old age of mining was based on the following three criteria:

- Reserves;
- Market; and
- Capital.

All three above stated criteria are a prerequisite to a successful mine development.

HISTORY OF THE INDUSTRY

The Australian Coal Industry today is the largest export industry in the world.

From an export perspective, the industry has a relatively young history as follows:

Phase 1

Serious development of the export market occurred during the 1970's, led principally by Kembla Coal and Coke in New South Wales and Utah Development Company in Queensland. The exports of significant volumes primarily coincided with the development of coking coal products suitable for use in the Japanese steel mills.

Phase 2

The industry was influenced by world oil shocks, which occurred 1974 and 1979. As a consequence the attitude of customers changed. No longer was the supply of raw materials guaranteed.

Phase 3

During the 1980's customers were increasingly seeking security of supply of relevant raw materials. It was common for customers and trading related companies to actively participate in development of production capacity. Long-term contracts were introduced to further encourage mine development. Unfortunately many operations were over capitalised and were only justified in the context of an ever-increasing coal price scenario.

Phase 4

During the 1990's the coal price reality did not meet the expectations of the 1980's. The industry was forced to rationalise in an effort to provide a satisfactory economic result for its shareholders. Further the industry has undergone a significant ownership consolidation phase also driven by the requirement for improved economics. As an example, Coates (2002) advised that in respect to thermal coal or steaming coal, four major exporters

¹ Macarthur Coal Ltd

accounted for about 42% of supply from Australia. Coates (2002) further advised the same four global suppliers in 2002 were responsible for 70% of Australia's supply capacity.

TODAY'S INDUSTRY STATUS

The Australian Coal Industry in calendar year 2002 exported approx 200 million tonnes of coal. Approximately half of the export tonnage was thermal or steaming coals and the other half was coking coal products. The industry trend is quite different for both market segments as follows:

Thermal or Steaming Coals

Thermal or steaming coal is mainly used for electricity generation. The coals are rated primarily on energy content for pricing purposes. The steaming coal market has become a commodity driven market.

With the increasing exports of steaming coal from countries such as China and Indonesia, the additional supply has provided a competitive environment to the pricing and utilisation of Australian based coals. Customers to date in this decade have tended towards shorter term purchasing arrangements as a consequence of having multiple sourcing options for coal supply.

Coking or Metallurgical Coal

The driver of the world's steel industry at this time is the growing industrial demand occurring within China. China's industrial growth has been significant over the last 5 years and the crude steel production capacity is now a reported 180 million tonnes. China's increasing industrial growth has now resulted in boom times for consequential supply of raw materials to the steel sector. As a comparison the crude steel production capacity for the Japanese steel mills is approximately 100 to 110 million tonnes per annum. Currently, demand is exceptionally strong for:

- Iron ore;
- Coke;
- Coking coals; and
- Coals suitable for the Pulverised Coal Injection process.

Most observers around the world are now predicting a sustainable growth scenario for China's industrial sector. As a consequence customers are again seeking security of supply of raw materials similar to the 1980's.

FACTORS AFFECTING MINE DEVELOPMENT

Development of a new mine represents a rare opportunity for the mine developer to apply latest thinking in working practices, method of mining, industrial and employee agreements together with the way the company chooses to do business. Factors affecting mine development include:

Access to Viable Reserves

Marshall (1974) reported that the old age of mining was to mine the best coal first, in order to repay capital.

From the viewpoint of the Bowen Basin, its best and most economic coal has been mined to date. The reality is today's viable reserves generally are at greater depths of cover.

A challenge to open cut operators in the Bowen Basin in the early 1980's was to extend their operations beyond a threshold of 60 meters depth of cover. Today it is common for open cut miners to be operating at 100 meters to 200 meters depth of cover. Similarly, companies are forced to undertake underground mining at increasing depths of cover. This trend is occurring not only in the Bowen Basin but in New South Wales as well.

Changing Coal Market

In relation to thermal coals, the ultimate use of coal continues to be constrained by the greenhouse gas debate, despite being the cheapest form of fossil fuel available to an overseas power station. In the future, customers will favour coals with an improved environmental status; that is, relatively low emissions of oxides of nitrogen and sulphur.

In relation to the steel industry, customers are expanding the use of poor grade coking coals and increasing the use of pulverised coal injection technology. As a consequence, mine developers are being encouraged to give priority to the establishment of production capacity for these coal types.

Market Pricing Volatility

If it was a perfect world conceivably business could be based on a fixed price in Australian Dollar terms. It would allow for business to be planned with a satisfactory risk profile, having a cost base in the same currency. The price of coal historically has fluctuated in US Dollar terms by as much as +/-20%. Some companies operating in Australia are global companies that have a preference for US dollar income, which provides a natural hedge for their respective global business. Other companies operating in the Australian Coal Industry are Australian companies whereby they report to their shareholders in Australian Dollar terms. During calendar year 2003 the Australian Dollar exchange rate relative to the US currency has appreciated by in excess of 30%. Such fluctuations in revenue provide an added challenge to mine operators to develop viable operations with a risk profile that is acceptable to investors.

MACARTHUR COAL STORY

Macarthur Coal undertook an Initial Public Offering (IPO) in July 2001. The purpose was to raise funds to grow the business.

Strategy

Macarthur Coal's strategy is to develop new coalmines within the Australian Coal Industry based on the identification of market segments that the company believes will grow faster than the general coal market. As part of the strategy the company has placed an important emphasis on exploration to identify new projects.

Market Segment

Macarthur Coal and its joint venture partners at the Coppabella and Moortvale mines supply approximately 40% of the global supply for low volatile coals, used for Pulverised Coal Injection (PCI) technology in the world's steel mills. Australia and Queensland particularly remain the principal supply source.

The continued use of low volatile coal for pulverised coal injection represents an economic benefit to customers, whereby it reduces its dependency on coke. Low volatile coals are attractive due to their high carbon and energy content.

Assets

Macarthur Coal is the major joint venture participant in the Coppabella & Moortvale Joint Venture. Ownership is as follows:

• Macarthur Coal Limited	73.3%
• CITIC Australia Pty Ltd	7.0%
• Marubeni Coal Pty Ltd	7.0%
• Nissho Iwai-Nichimen Australia Limited	7.0%
• Kawasho International (Australia) Pty Ltd	3.7%
• Nippon Steel Trading Co Ltd	2.0%

Production capacity of the Coppabella and Moortvale mines is almost 6 million tonnes per annum. Both mines have been developed as stand alone mines with their own infrastructure comprising a coal preparation plant and a balloon loop. Both operations are open cut with substantial future underground resources.

Macarthur Coal believes that the fundamentals are right for both projects, whereby both projects are open cut, relatively shallow compared to other operations in the Bowen Basin, close to port and close to under utilised infrastructure. The company has sought to minimise its capital investment through the use of contractors, both in the mining and coal preparation plant process.

Exploration

Macarthur Coal has the rights to in excess of one billion tonnes of coal resources in the Bowen Basin (see Figure 1). The main centre of activity is the Coxendean sub-basin, which has been confirmed as a major extension to the Bowen Basin of Queensland. The Coxendean Sub-basin contains the Olive Downs, Codrilla and Wilunga Projects. The company has confirmed the existence of coal measures over 25 kilometres of strike length at less than 100 meters depth of cover within the Coxendean sub-basin. The priority is to identify open cut reserves at this time. However the company recognises the potential for the above coal measures to be mined by highwall underground techniques.

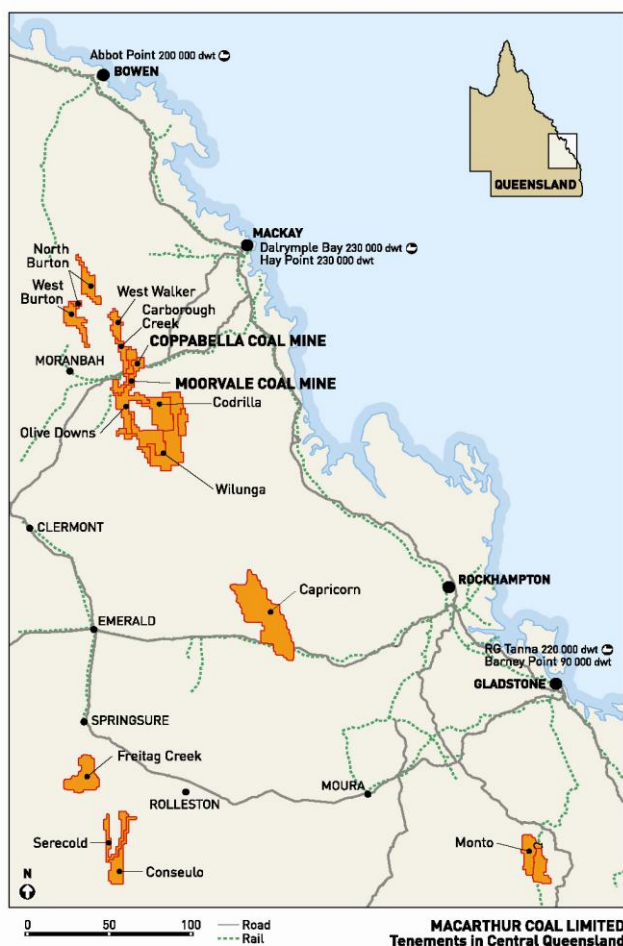


Fig 1 – Location of Macarthur Coal Mines

CONCLUSION

Mine development can be a rewarding and challenging experience. However, business is subject to substantial fluctuations in sales price and exchange rate. Under the circumstances, the overriding challenge for mine developers is to maximise competitiveness. The industry cannot rely on historical practices as a pre-requisite to competitiveness and must continue to develop new and more efficient ways to conduct our business.

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INNOVATIVE TECHNIQUES FOR DETECTION AND CONTROL OF UNDERGROUND SPONTANEOUS COMBUSTION OF COAL

Sheng Xue¹ and Hongyi Cui²

ABSTRACT: In recent years, the frequency and intensity of spontaneous combustion of coal in Australian underground coal mines has shown worrying signs of increasing. To enhance spontaneous combustion management capabilities in the Australian coal industry, a study was undertaken to investigate three innovative techniques that have been successfully used in China. These three are the radon detection technique, the infrared technique and the colloid injection technique. The radon detection technique is for remotely locating the areas of underground spontaneous combustion from a surface location. The infrared detection technique is for locating the spontaneous combustion within a short distance. The colloids injection technique is for controlling spontaneous combustion after it is located. This paper describes these techniques including their principles, operations, applications in China, and applicability in Australia.

INTRODUCTION

Spontaneous combustion (sponcom) is a significant hazard in coal mines worldwide, including Australia and China. If not detected at its early stage and appropriate controls are not employed, sponcom can lead to fires, explosions, asphyxiation, loss of life, equipment and resources.

In China, more than 90% of the fires in coal mines are the result of sponcom. 54.9% of the state-owned coal mines and 29.1% of the locally owned coal mines are classified as sponcom prone mines. Every year, there are about 300 sponcom-induced fires in coal mines in China. It is estimated that sponcom of coal in mines consumes 100 million tonnes of coal each year. To overcome the problem of sponcom, Chinese coal mines, in cooperation with research institutes and universities, have put substantial manpower and financial resources to undertake the fundamental research. Their aim is to develop techniques for preventing, detecting, locating and controlling the sponcom, and to implement these techniques in the Chinese coal industry. These efforts have led to an increased understanding of sponcom, the development of a number of innovative techniques, and a substantial reduction of sponcom-induced fire hazards.

In Australia, sponcom has been and continues to be a major hazard of underground coal mining. Over the last thirty years there have been over 250 reported or recorded incidents of sponcom in New South Wales and Queensland. The economic and social costs to the Australian coal mining industry in particular are very high. As evidenced over the last couple of years, the frequency and intensity of sponcom in Australian underground coal mines is showing the worrying sign of increasing. How to deal with sponcom is one of the major focuses of the Australian coal industry and the relevant governmental agencies.

In 2002 CSIRO and Yankuang Group Corporation in China agreed to undertake a study to investigate a number of innovative techniques developed and used in China for preventing, detecting, locating and controlling sponcom of coal (Xue and Cui, 2003). The applicability of these techniques in Australian coal mines were also investigated. The techniques selected for investigation include radon detection technique, infrared detection technique and colloid injection technique.

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² Yanhuang Group Co. Pty, China

RADON DETECTION TECHNIQUE

Principle

In radioactive processes, particles or electromagnetic radiation are emitted from the nucleus. The most common forms of radiation emitted have been traditionally classified as alpha (α), beta (β), and gamma (γ) radiation. The radioactive decay will change one nucleus to another if the product nucleus has a greater nuclear binding energy than the initial decaying nucleus. The difference in binding energy (comparing the before and after states) determines which decays are energetically possible and which are not. The excess binding energy appears as kinetic energy or rest mass energy of the decay-products. The energy emitted in the radioactive decay can be detected, measured and used as information to determine the concentration of the nucleus, its decay mode as well as its other properties. This is the principle of nuclear-based detection techniques.

U-238 is a common rare element in rock strata, as is its decay product Rn-222 (radon) and radon progeny. Radon has a strong diffusion ability. Activated carbon, silica gel, polyethylene and some other materials can easily adsorb radon and its progeny. This property enables radon and its progeny to be easily collected from the surface with a container coated with these adsorbents and analyzed. Experimental test data show that when the coal is heated up, the emanation rate of radon from overlying strata will increase. This relationship, combined with radon radioactive and its unique properties form the fundamental principle of radon-based detection techniques for coal sponcom.

Operation and Analysis Procedures

The operation and analyzing procedures with the radon detection technique are described below.

- Designing surface measurement grid. Firstly deciding the survey area underground and then selecting the surface area right above the underground area to be investigated. This is followed by design of surface measurement points. The measurement point spacing can be 5 m, 10 m, 15 m, or 20 m depending on requirements of site conditions.
- Placing test cups. An auger machine can be used to dig the holes. The hole is 30 cm in diameter and 30-40 cm in depth. Once the hole is dug, an alpha test cup (Figure 1) is placed in the hole upside down.
- Analyzing the cups. The cups are buried in the holes for at least 4 hours. The alpha cup is then recovered and inserted into a CD-1 radon detector for testing (Figure 1). The CD-1 radon detector shows counts per minute reflecting the concentration of radon and its progeny.
- Processing test data. A program is used to process the test data. It produces a 3D map of abnormal CPM values, and the location of "high-temperature" area(s) (Figure 2).

Characteristics

- It can remotely locate abnormal temperature areas. Its accuracy for locating the sponcom center is 90 %.
- Detection depth is up to 800 m.
- It is of high suitability, low cost and easy to operate.
- It has a high reliability and is largely unaffected by external factors.

Applications

The radon detection technique has been used in more than 25 underground coal mines in China with great success. For example, the coal seam in Caili mine is prone to sponcom. The mine had 40 cases of sponcom from 1967 to 1997. Efforts to control sponcom had been hampered by not knowing the exact location of sponcom. The radon technique has been used in LW2334, LW2349 and LW2313. An area of 66,000 m² was surveyed and locations of sponcom were detected with this technique.



Fig 1 - Alpha cup and CD-1 radon detector

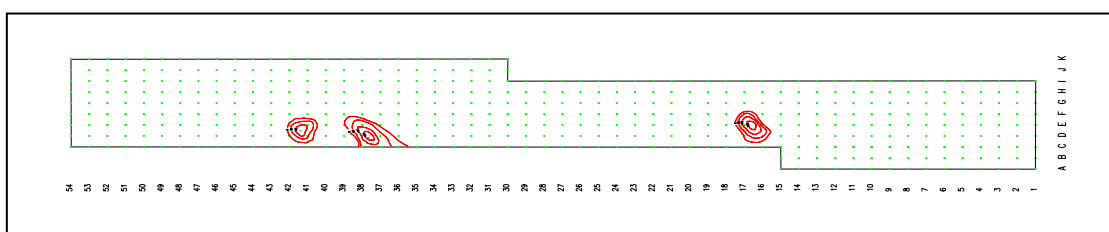


Fig 2 - Hot spots detected with the radon technique

INFRARED DETECTION TECHNIQUE

Principle

All warm (warm defined as being above 0 Kelvin in temperature) objects' atoms, molecules, and electrons are always in motion, vibrating and radiating (emitting) infrared waves, forming in infrared radiation field. As the object's temperature increases, the intensity of the radiation increases. The radiation field can be characterized by its energy, momentum, direction and other information. Like any other objects, a coal seam is also emitting infrared waves. If there is a sponcom in the coal seam, this should be reflected in the characteristics of its infrared radiation field. The infrared radiation field can be used to determine the existence of the sponcom and its extent through establishing the relationship between the field and its source and monitoring the characteristic change of the field.

Operation and Analysis Procedures

The operation and analyzing procedures of the infrared detection technique are described below

- Layout of measurement points. Measurement stations are set along the gateroad to be surveyed. Each measurement station covers a number of adjacent measurement points in the ribs, roof and floor (Figure 3).
- Field Measurement. An infrared detector is used to measure the strength of the energy field of infrared radiation and surface temperatures of the measurement points. If a zone of abnormal strength is recorded, then repeated measurements in that zone are undertaken.
- Data Analyses. A program is used to process measurement data. The program has three main functions: (1) graphic outputs of strength profiles of infrared radiation, (2) identifying any abnormal strength of radiation caused by sponcom, and (3) determining the position and temperature of the

sponcom by the inverse calculation of heat conduction based on the heat conduction mechanism of coal strata.

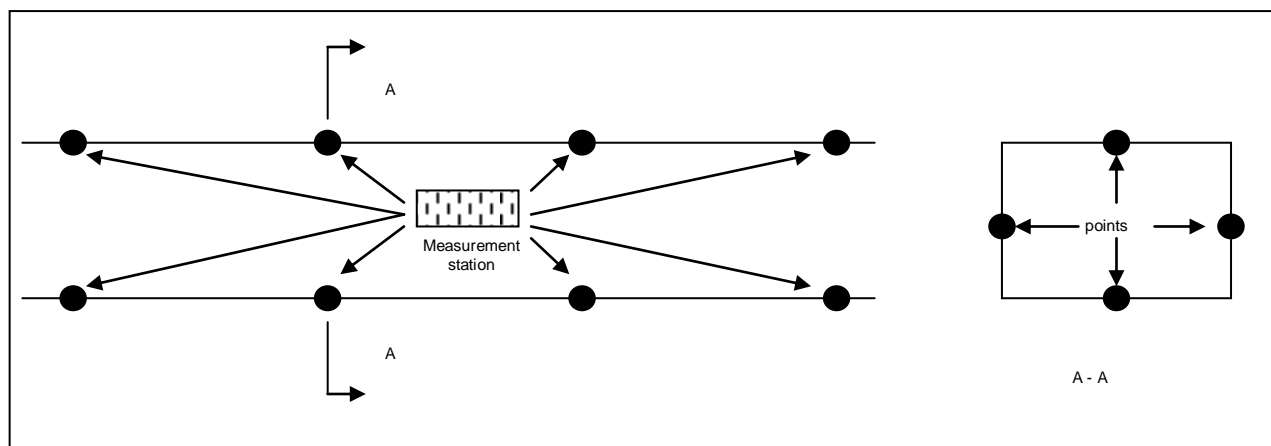


Fig 3 - Schematic layouts of measurement stations and points with the infrared technique

Characteristics

- It is remote and requires no direct contact to detect sponcom of coal.
- Maximum detection depth of 10 m in a coal pillar
- 90% accuracy for detecting the location of a sponcom
- It can detect a coal sponcom with temperature at 130° C and above
- Suitable for detecting “hot spots” in coal pillars and areas adjacent to roadways

Applications

The infrared technique has been successfully used in about a dozen underground coal mines in China for locating coal sponcom. For example the coal seam in Baodian mine is prone to sponcom. This technique was used to detect the suspected heating spots in the coal pillar and roof of two roadways in longwall #5308. A total of 1140 measurement points were selected along the 1900 m long roadway and two heating spots in the roof were successfully detected with this technique (Figure 4).

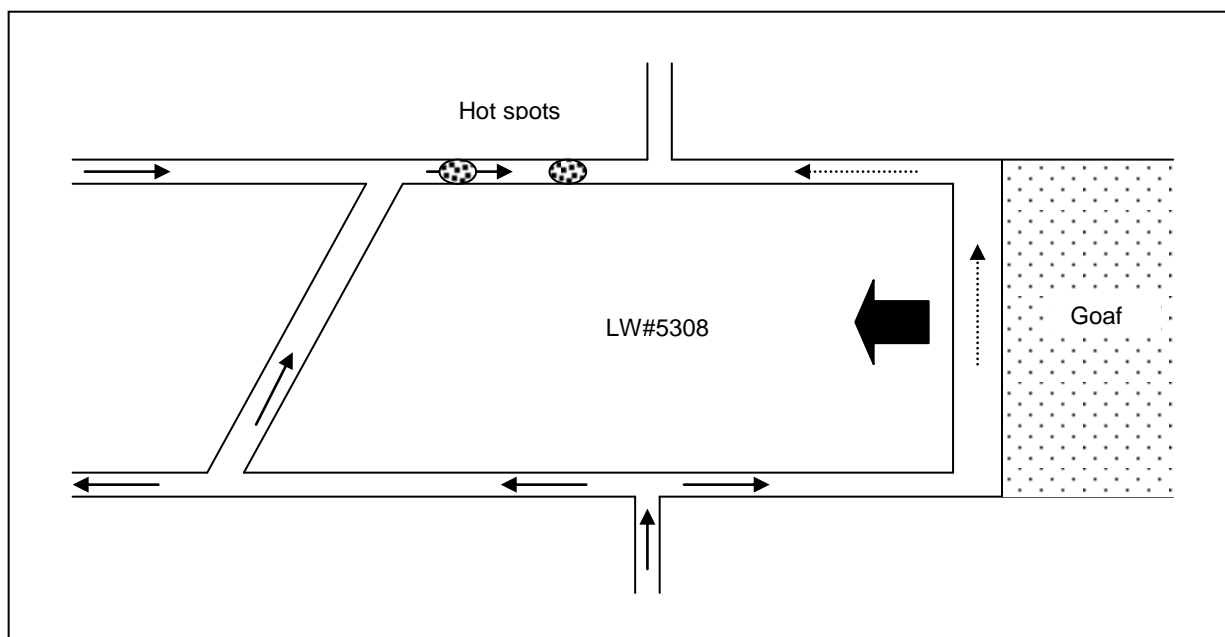


Fig 4 - Hot spots detected in LW#5308 of Baodian mine with the infrared technique

COLLOID INJECTION TECHNIQUE

Types and Characteristics of Colloids

Colloids developed for the control of sponcom can be broadly divided into three categories: gels, large-molecule colloids and compound colloids. The gels consist of a base material, an additive for fast gelatinization, and water. The large-molecule colloids are composed of large-molecule materials and water. Adding some additives for enhancing mechanical strength in the gels or large-molecule colloids makes compound colloids.

Operation systems

Two systems have been developed for the colloid injection technique for sponcom control: an underground-based system suitable for controlling small-scale sponcom and a surface-based method suitable for controlling large-scale sponcom. The main equipment used in the underground-based system includes a movable colloid mixer and pumping station. Required materials are fed into the station and mixed to make the colloid and then pumped into the area of sponcom via pipelines. Figure 5 shows the flow chart of the system.

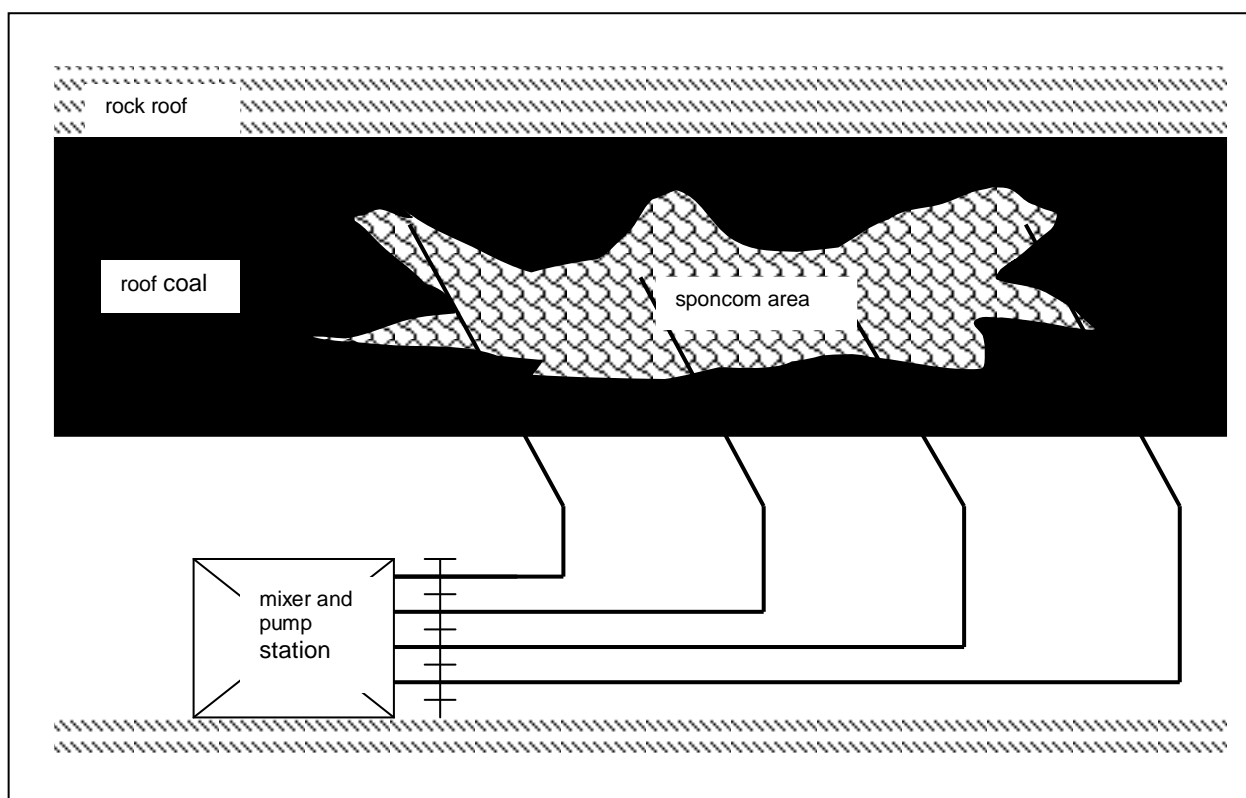


Fig 5 - An underground-based system for colloid injection

To control large-scale sponcom, a surface-based system should be used. A surface-based mixer blends base materials with water. A pipeline is then used to deliver colloids from surface to underground, the pipeline is connected to a number of borehole drilled from underground workings into the area(s) of sponcom. A small mixer and pumping station located underground is used to mix water with the additive for fast gelatinisation and pump the mix into the pipeline. Figure 6 shows the flow chart of the system. The system is capable of delivering 30-100 m³/h of colloids into sponcom spots.

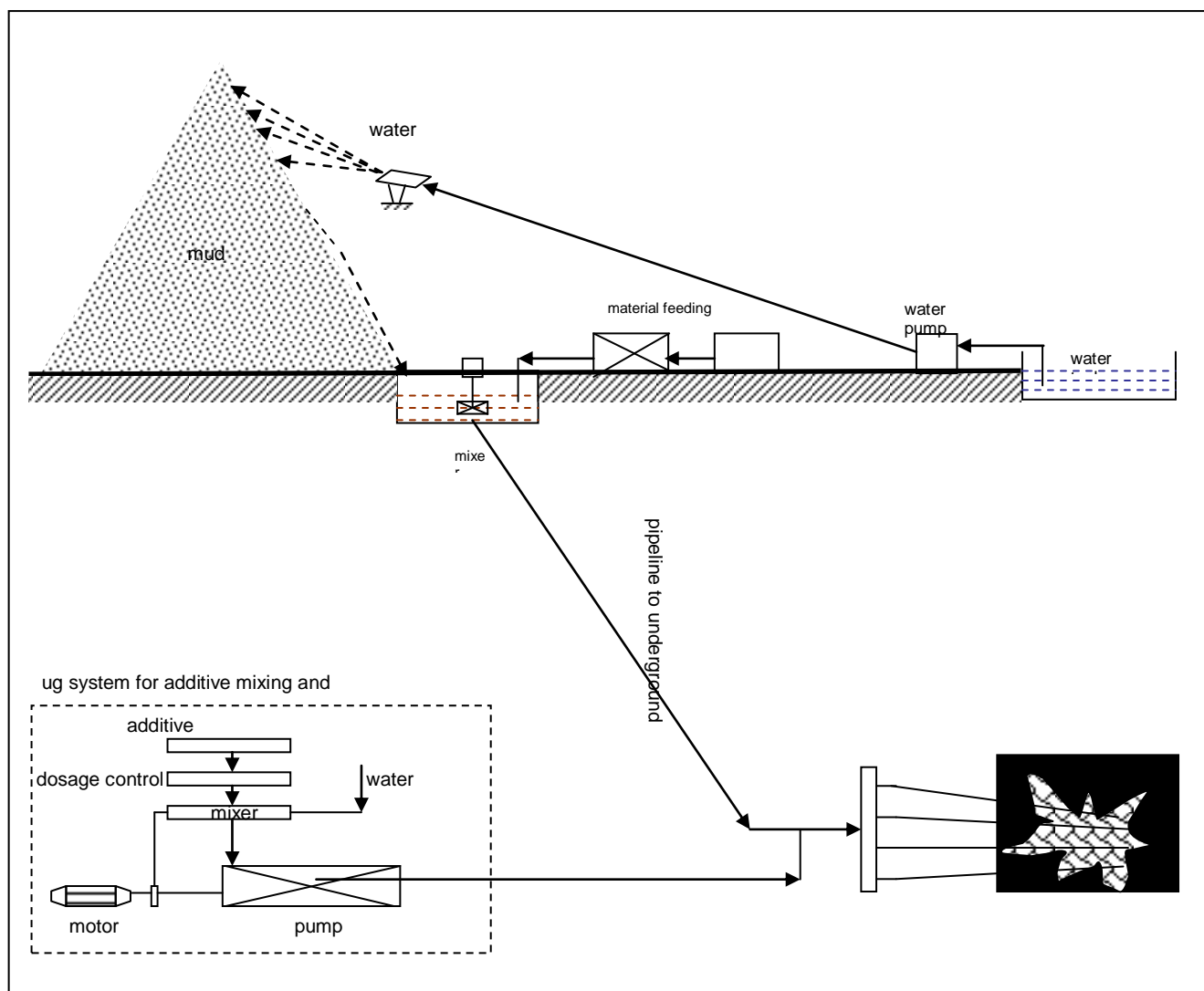


Fig 6 - A surface-based system for colloid injection

Characteristics

- Speedy control of sponcom. For a small-scale sponcom in a coal pillar it may take only a few hours to control, even for a large-scale sponcom in goaf it may take a couple of days at maximum.
- Colloid can be solidified in fragmented coal, resulting in the blockage of leakage passes and hence stopping poisonous gases flowing out from the passes. At high temperatures a colloid gives off a very small amount of water steam (unlike water injection technique) and therefore there is no possibility of explosion of water gas and no risk of injuries resulted from high temperature water steam.
- To date there has been no reoccurrence of sponcom in an area treated with colloid.

Applications

The colloid injection technique has been successfully applied to control about 100 cases of coal sponcom in Chinese coal mines. For example, in March 1993, sponcom occurred in longwall face #11501 of Wangcun mine. Sponcom occurred in a large goaf area immediately behind chock #9 to chock #13. Several control techniques including ventilation pressure balancing, water injection and slurry injection were used with no success. As a last attempt to avoid sealing of the face the colloid injection technique was applied. The area was injected with 100 m³ gel, and sponcom was controlled.

APPLICABILITY IN AUSTRALIA

The Radon detection technique is operated from the surface. There are no specific requirements for its operation condition. Since there is no competing technique available in Australia, this makes the technique quite attractive. Its limitations include that targeted surface areas have to be accessible by operators and if sponcom occurs in multiple seams the technique needs to be used with other techniques for accurately locating the sponcom.

The Infrared detection technique is operated from underground openings. For pillar heatings, the infrared technique may be more sensitive than gas detection. However it cannot detect sponcom below 130° C or if sponcom occurs more than 10 m inside a coal pillar.

There is no specific operation requirement with the colloid injection technique. However for the large-scale surface system it requires some surface infrastructures such as a water tank, mixer and pipeline. It should be noted that the area of sponcom has to be relatively close to an underground opening so that injection boreholes can be drilled into the area.

These three techniques are technically and operationally feasible to apply in Australian coal mines, and should be included in the current collection of approaches available for prevention, detection, locating and controlling sponcom of coal in Australian coal mines.

CONCLUSIONS

The three techniques investigated in this study offer the Australian coal industry the potential to enhance its capacity to deal with the issues of preventing, detecting, locating and controlling sponcom of coal. This study has identified the following advantages of these techniques over existing technique associated with sponcom:

- These techniques enable an integrated approach from detection to controlling.
- Radon technique provides the only solution for remotely locating underground sponcom from surface
- Infrared technique offers an alternative for detecting sponcom, particularly in coal pillars
- Colloid injection technique can be used in conjunction with other inertisation techniques to control sponcom.

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SAND PROPPED HYDRAULIC FRACTURE STIMULATION OF HORIZONTAL IN-SEAM GAS DRAINAGE HOLES AT DARTBROOK COAL MINE

Rob Jeffrey ¹, Christian Boucher ²,

ABSTRACT: Longwalls 107, 108 and 109 at Dartbrook Coal Mine contained coal with a high gas content and low permeability. Horizontal in-seam drain holes were found to have low gas production rates compared with drainage rates in previous panels. Hydraulic fracture stimulations, using water and sand, were therefore carried out in three boreholes in Longwalls 109 and 108 at Dartbrook to assess the effectiveness of sand propped fractures in stimulating gas drainage from in-seam boreholes.

Boreholes 108-10-10 and 108-7-1 were stimulated with 20 and 10 fractures respectively and, on average, 100 kg of sand was placed into each fracture. The fractures placed into LW 109 were to be mined and mapped, but operational constraints precluded mapping of these fractures.

The stimulations produced a significant increase in gas drainage rates from the two boreholes. Hole 108-10-10, which ran perpendicular to the major joint system in the seam, increased its early gas rate by a factor of about 180 while hole 108-7-1, which was drilled parallel to the joint set, increased its rate by about 22 compared to pre-stimulation rates. The stimulated gas rates continuously increased for several weeks and the higher rates were sustained for the entire period the holes were monitored.

Based on the higher stimulation effect achieved in hole 108-10-10 (drilled perpendicular to the jointing) compared with hole 108-7-1 (drilled parallel to the jointing), target drainage holes drilled perpendicular (north-south) to the jointing are better stimulation candidates.

Fracture modeling suggests the sand proppant bank may extend to 15m from the borehole. The unpropped portion of the fracture may extend to more than 40m. A purpose-built fracturing system was developed and used at Dartbrook to stimulate holes that covered most of LW109. This full-scale enhancement of gas drainage was successful and allowed efficient mining of that panel.

INTRODUCTION

Hydraulic fracturing is applied routinely in the petroleum industry to stimulate oil and gas wells. Typically, a stimulation treatment consists of a clear pad fluid injected at rates and pressures sufficient to initiate and extend a hydraulic fracture. Once a fracture of sufficient size and width has formed, proppant is added to the fluid and pumped into the fracture as slurry. The proppant is carried into the fracture and, after the injection stops, serves to prop the fracture open to form a permeable channel in the reservoir through which the hydrocarbons can be produced. As the well is produced, the propped hydraulic fracture connects the wellbore to a large surface area of the reservoir and provides a conductive pathway to carry the oil or gas back to the well. Such treatments are commonly done to stimulate coal seams to accelerate production rates from coalbed or coal seam methane wells. Both vertical and horizontal oil and gas wells are also candidates for fracture stimulation.

Proppants are selected based on the requirements that they are permeable and strong enough to prop the fracture open without crushing. Round, spherical and sieved sand is the lowest cost proppant material commonly used and is able to withstand fracture closure stresses of up to about 35 MPa. In higher stress environments, more costly but stronger resin-coated or ceramic proppants are used.

¹ CSIRO Petroleum

² Dartbrook Coal

Previous experience at other sites

Several projects to investigate propped fracture stimulation of drain holes have been carried out in Australia and overseas. The earliest work that we are aware of was carried out in Queensland by the Department of Mines. In late 1979 and early 1980 the Department conducted a four month long research project to investigate hydraulic fracture stimulation of horizontal drain holes (Croft, 1980). The project's intent was to design and test a system for pumping fluid and sand into horizontal holes at pressures and rates sufficient to induce hydraulic fracturing. The plan required the pump and other equipment to be located underground next to the coal rib. A number of equipment developmental problems were encountered and no fracture treatments were successfully performed. However, the concept of stimulating horizontal holes drilled into the seam from underground is, essentially, the same as have recently been undertaken in the trial at Dartbrook.

Fracture stimulation work is now done in horizontal wells in the petroleum industry on a fairly frequent basis (Walker, Ehrl, and Arasteh, 1993; Weijers, et al., 1992). Such wells are drilled into conventional reservoir materials (sandstones or limestones) and are stimulated by fracturing for the purpose of establishing a better connection with the reservoir. Multiple fractures can be formed during injections into long sections of open horizontal wells in a single treatment and most of the fractures formed tend to be located over the first third of the horizontal section of wellbore (Grieser, Wiemers, and Hill, 1999).

When a long section of a borehole is treated without attempting to isolate a small zone, the hydraulic fracture will initiate and grow from the point or points where pre-existing weaknesses in the borehole exist. Once a fracture starts growing, it typically will continue to extend at a lower pressure than required to initiate other fractures along the borehole. Therefore, trying to treat a long section of hole in one injection may, in the worst case, only produce one major hydraulic fracture in the entire open section of the hole.

To partially overcome this problem, diverting agents, which are materials added to the fluid that act to block the entry to the fractures, are sometimes pumped at various times during these treatments in order to divert the fluid and proppant from an established fracture into a new area of the wellbore. Control over the number and position of fractures formed in open-hole treatments like this is limited and the diverting agents have the potential to damage the permeability around the wellbore. Initiating multiple fractures or extending and propping too few fractures, are problems that can be avoided by limiting the length of the interval being treated, usually by casing the well and then selectively perforating a small zone for each treatment. Treating a long open hole section of a horizontal well does have the significant advantage of not requiring packers to be set at a number of positions along the wellbore.

In 1993, a project was undertaken to perform several hydraulic fracture stimulations in horizontal drain holes at the Soldier Canyon Coal Mine, in Utah (Kravits, 1993). Five propped fractures were placed along a 2505 foot-long horizontal borehole. Gas production from the hole increased by 46 percent and then, over a period of about 4 weeks, declined back to pre-stimulation rates. Stress conditions at this site in Utah are suspected to have allowed the hydraulic fractures to grow vertically into overlying and underlying rock, resulting in poor stimulation of the coal seam. Use of plastic beads rather than conventional sand proppant also caused problems in mixing, pumping, and placement of proppant.

ACARP project C4033 was undertaken to trial propped fractures for stimulation of horizontal in-seam boreholes (Jeffrey, 1999). In June 1996, field tests were carried out at Central Colliery near Middlesbrough, Queensland. Pumping and mixing equipment was located on the surface and the fluid and sand were pumped down an HQ borehole to a cut-through in the 306 maingate at Central. From the cut-through, the fluids were carried into a horizontal borehole drilled into the 307 panel. A straddle packer system was deployed in the horizontal drain hole. Water and sand bypassed the packers at injection pressures well below fracture initiation pressure.

The bypass is thought to have occurred via fractures or structures in the coal that ran along the length of the packers at the test site. Axial fractures along the packers may have formed because stress conditions, in combination with pressure exerted on the borehole by the inflated packers, were sufficient to split the borehole. Seam conditions at this site were difficult and several attempts were made to drill other holes without success. These results illustrate that some seams may not be suitable candidates for fracture stimulation by running packers in open hole sections and may require significant modifications to the hole completion or fracturing procedure.

IN-SEAM DRAINAGE AT DARTBROOK

At Dartbrook coal mine, the Wynn upper seam is extracted by longwall mining methods. The longwall operations extract 4m of the 28m thick mega seam, leaving a coal floor and roof. Figure 1 contains a plan showing Longwalls 8 and 9. The fan-array holes shown, that are drilled essentially across the panels, are the standard in-seam drainage holes used at Dartbrook. The holes drilled for the sand propped hydraulic fracturing work are shown with thicker lines and are aligned mostly along the panel length. Fractures placed in these holes would then extend east-west across the hole and parallel to the panel width.

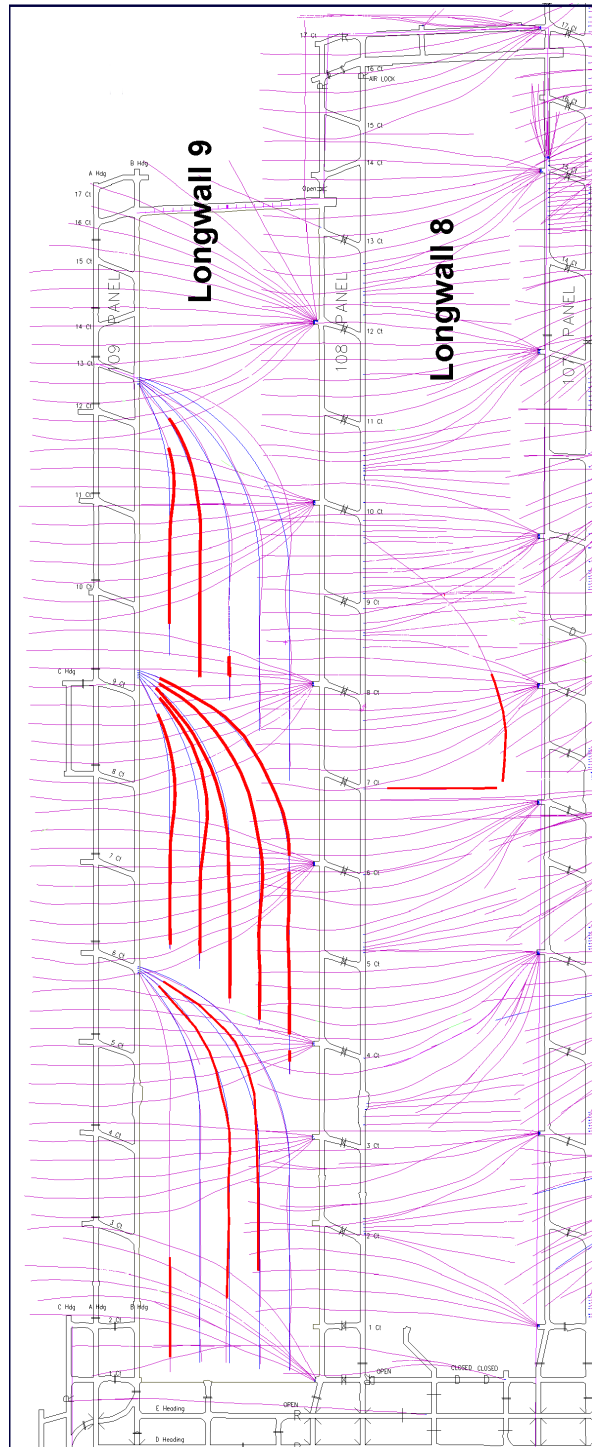


Fig 1 - Longwall 9 with hydraulic fractured holes indicated.

In-seam holes in panels mined before LW7 typically produced gas ranging in rate from 1 to 4 litres per minute per metre of hole. However, in LW7, 8 and 9 this rate was reduced, because of low permeability coal, to 0.1 to 0.3 litres per minute per metre of hole. Coal permeability was estimated to be about 0.02 to 0.03 md while gas content in these three panels was 8 to 9 cubic metres per tonne, composed of 90 percent carbon dioxide and 10 percent methane. This drainage rate was too low to effectively reduce the gas content to a content of 5 cubic metres per tonne, which is known to be low enough to avoid gas delays during longwall mining.

HYDRAULIC FRACTURE STIMULATION OF DRAINAGE HOLES

A measure of the effect of fracture stimulation on gas drainage can be obtained by comparing the productivity index, J_0 , for the unstimulated hole to the productivity index, J , for the stimulated hole. The productivity index is defined as:

$$J = \frac{q_g}{(\bar{p} - p_{wf})}$$

where q_g is the gas rate, \bar{p} is the average reservoir pressure, and p_{wf} is the borehole flowing pressure (Lee, 1989). This expression for J is only correct for produced fluids that are of small compressibility (such as oil or water) but is applied to gas production to give order-of-magnitude estimates. The effect of stimulation is usually illustrated by plotting the ratio J/J_0 , which is called the stimulation ratio, as a function of another parameter such as time. Figure 2 contains a plot of J/J_0 against time constructed for a set of reservoir and fracture parameters and is presented to qualitatively illustrate the magnitude of fracture stimulation on gas production. For low-permeability reservoirs, the stimulation ratio during unsteady state flow (which can last for long periods of time) can easily exceed 10 or even 100 in value.

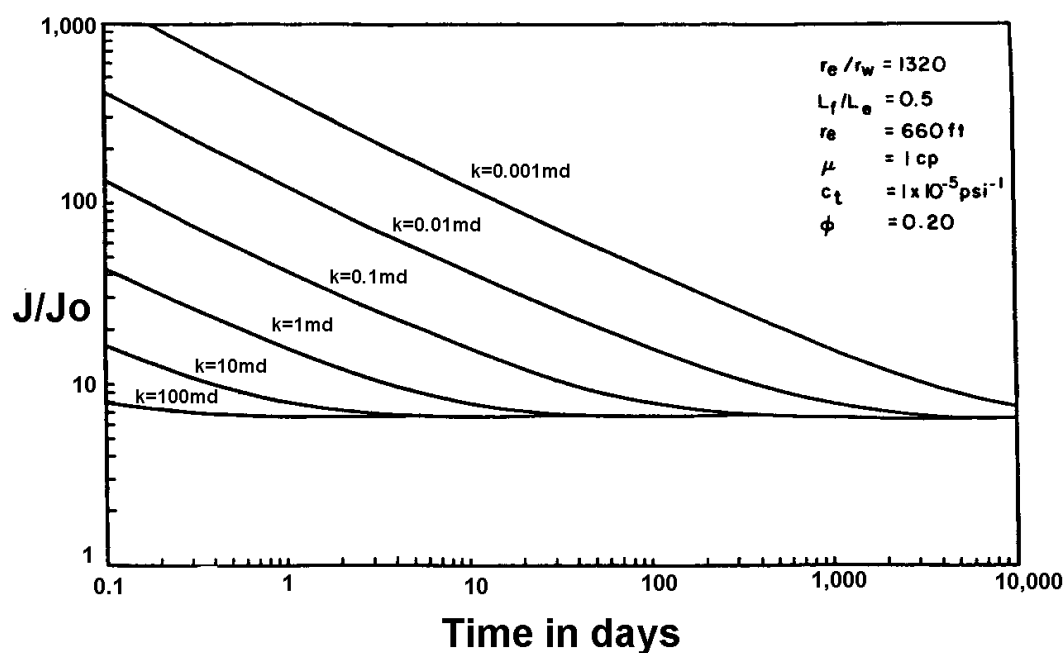


Fig 2 - Productivity ratio with time for different reservoir permeabilities (after Lee, 1989)

Water Only Fractures

In late 2001, a program of hydraulic fracture stimulation of horizontal drain holes was started at Dartbrook. The initial program used small water-only treatments placed at 3m intervals along several boreholes in Longwall 107¹. Typical treatments consisted of 5 minutes of injection at 160 to 170 litres per minute (800 to 900 litres total volume). The gas rate per metre of borehole was increased by these treatments by a factor ranging from 2 to 3. However, this stimulated gas rate was too low for the coal to be drained in time available and the stimulated gas rate dropped off too quickly after the drainage started (Gray, 2002).

A trial using hydraulic fractures that included placing a sand proppant into the fractures was therefore undertaken to determine if the additional fracture conductivity produced by propped fractures would provide the additional degree of stimulation needed. At the same time, the trial allowed an assessment to be made of the equipment needed to carry out such treatments from underground.

STIMULATION TRIAL IN LW108

A system of equipment to undertake the sand-propped fracture trial, from the maingate side of LW8, was assembled from existing CSIRO Petroleum equipment and mobilized for use underground at Dartbrook.

The major joint system in the coal at Dartbrook is subvertical and strikes at about 110 degrees. However, this joint system strikes at about 90 degrees in LW8 and 9¹. This joint system typically runs parallel to the maximum horizontal principal stress in the seam, which is also the direction that hydraulic fractures grow. The hydraulic fractures were, therefore, expected to be subvertical, growing along this joint direction. The borehole drilled at cut-through 10 was designed to run perpendicular to the expected fracture direction over its last 100m of length while the hole drilled from cut-through 7 was drilled more parallel to the expected fracture direction (see Figure 1). Fracture stimulating these two holes was designed to provide a comparison of the stimulation effect as a function of borehole orientation. Purpose-drilled boreholes targeted for this type treatment would best be drilled so the fractures form across the borehole axis, but extensive pre-existing drainage holes exist in the seam. A comparison of these two boreholes with different orientations was useful in determining if the existing drainage holes could be successfully stimulated or if new boreholes with the preferred orientation should be drilled to optimize the stimulation process and effect.

The hole at 10 cut-through was fracture stimulated with 20 fractures over its last 62m of extent (dark portion in Figure 1) with twenty fractures placed along this hole at 3 m. In contrast, the fractures placed in the hole at 7 cut-through should have been more aligned with the hole. A total of 10 fractures were placed over 88 m of this hole. In both cases, about 100 kg of 30/60 mesh sand proppant was pumped into each fracture formed.

Hydraulic Fracture Growth and Proppant Distribution

A numerical hydraulic fracture model has been used to approximately match the pressure measured during the fracture treatment at 327.7 m in the borehole at cut-through 10. The injection rate used in the treatment was 250 litres per minute. Coal and site properties used for this match consisted of a Young's modulus of 3500 MPa, Poisson's ratio of 0.34, minimum horizontal stress of 5.4 MPa, and coal permeability of 0.03 md. Figure 3 contains a plot of the pressure match obtained and Figure 4 shows the model-predicted fracture growth with time. Figure 5 shows the model-predicted proppant distribution. The propped fracture is predicted to have a conductivity of 10 md-m with this conductivity extending out to 15 m each side of the borehole.

Most of the proppant, as indicated in Figure 5, lies within 15 m of the borehole, but the fracture conductivity is enhanced out to 50 m. The model predicts the fractures grow vertically to about 7m in height.

Gas rates from each hole were monitored on a regular basis from the time they were drilled. The results of this trial were striking. Gas rate from both holes increased dramatically, with the hole drilled from 10 cut-through showing the largest increase. Figure 6 shows the specific gas rate history for the holes, expressed as gas rate per metre of hole. Not only did the gas rate increase significantly, but the increased rate was maintained for months. The curves shown are for gas rate per metre based on the entire hole length and for the length of the holes

¹ Byrnes, R. Private communications, April 2002

actually stimulated. The rate in the later case increased by a factor of 180 for hole 108-7-IF1 and by a factor of 22 for hole 108-10-10L. A stimulation ratio, J/J_0 , may be somewhat larger than the factors given based on the raw gas rates. The producing pressure of the flowing holes is maintained constant by the vacuum applied to the system while the mean reservoir pressure decreases slowly as gas is extracted, resulting in an increase in J compared to the raw gas rate. Results of this trial justified a drilling and sand-propped fracture stimulation campaign for LW9.

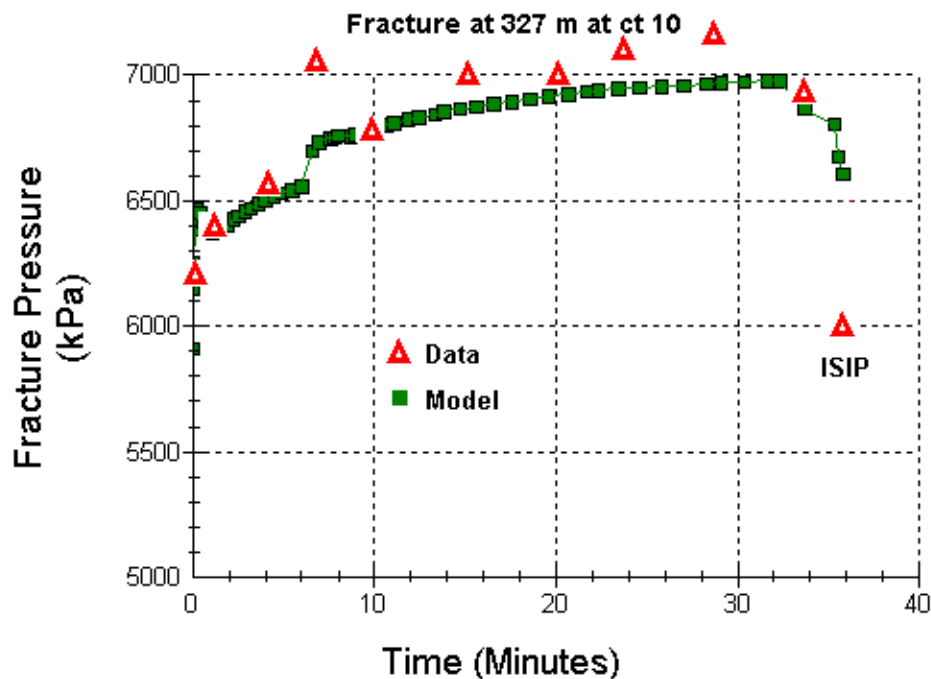


Fig 3 - Fracturing pressure versus time for the treatment at 327m at cut-through 10.
The red squares are measured pressure.

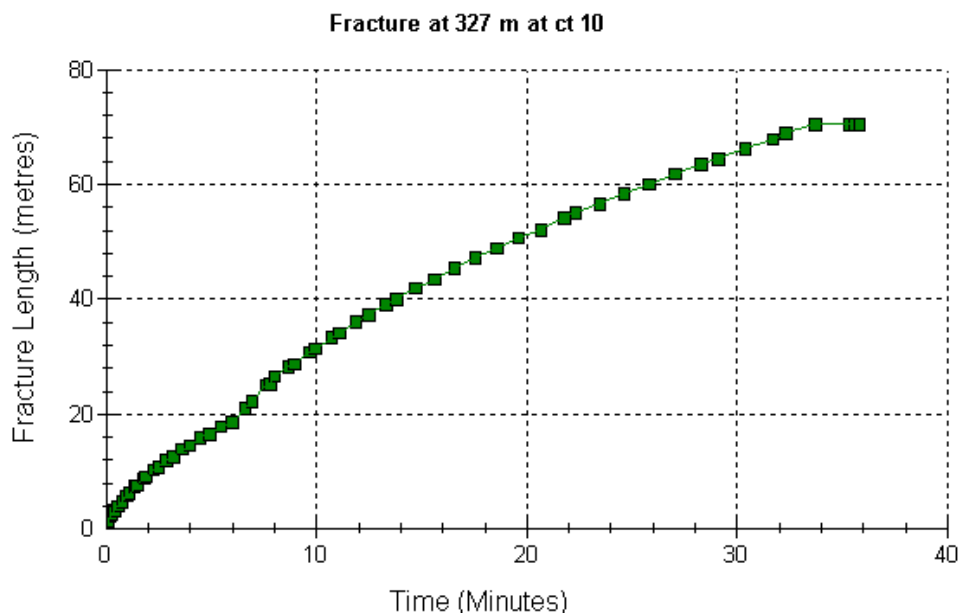


Fig 4 - The growth of the fracture with time for the treatment at 327 m at cut-through 10. The fracture grows to 32 m size in 10 minutes which is approximately the growth rate indicated by the observations of water at the rib during the last treatment at 7 cut-through.

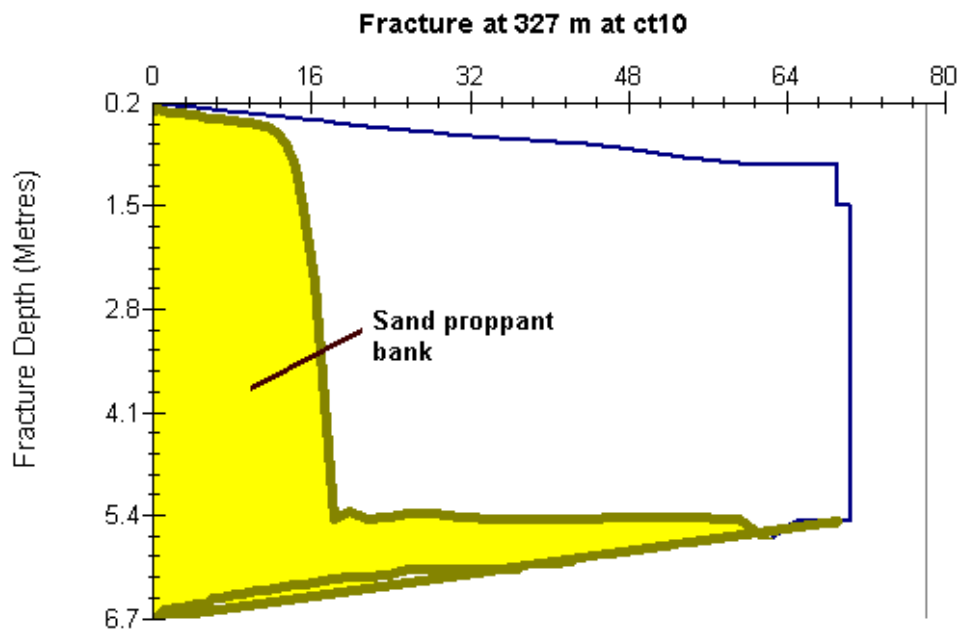


Fig 5 - The model-predicted proppant bank for the treatment at 327 m at cut-through 10. 160 kg of sand was placed in this treatment.

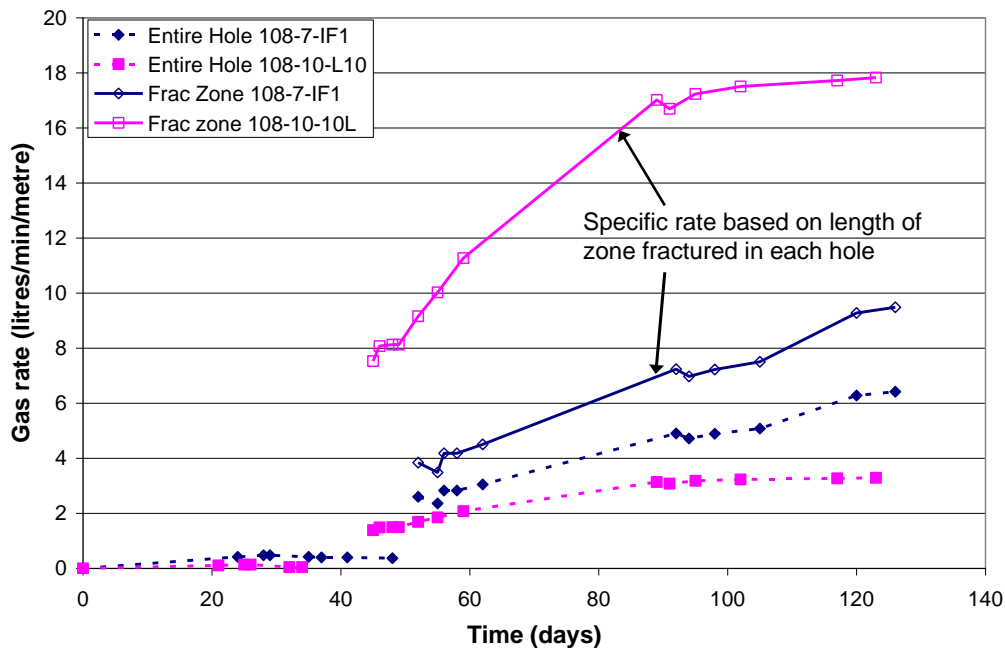


Fig 6 - Measured gas rate before and after sand-propped fracture stimulation. The fracturing occurred during the period in the data that is not connected by a line.

STIMULATION AND DRAINAGE OF LW9

As anticipated, hole 108-10-L10 which was drilled along the axis of the panel gave the best stimulation results. In addition, holes with this orientation allow few holes to be drilled to effectively cover the panel with propped hydraulic fractures, since the fracture propagate in the direction of the maximum horizontal stress which runs east-west across the panel. Therefore, the drain holes drilled in LW9, for the fracture treatments, were drilled to align with the long axis of the panel (see Figure 1). The holes were drilled from three locations, at cut-through 13, 9 and 6 with five holes drilled from each site and steered so they were spaced about 35m apart across the panel.

By the time the hydraulic fracturing in LW9 had started, the longwall was retreating. Approach of the longwall face limited the time available to fracture and drain all the holes that had been drilled so some holes were not treated and others had fractures placed every 6 m rather than every 3 m along them. Sections of the fractured holes that crossed near pre-existing fan-array holes were skipped over to avoid the possibility of the hole breaking out into the nearby existing hole. Such breakout could damage packers inflated there or lead to cross flow between the fractured hole and the older fan array hole. Table 1 contains a summary of the fracture treatments carried out in LW9 during this project. A total of 624 fracture treatments were carried out to stimulate 3174 metres total of 11 drain holes. Figures 7, 8, and 9 show the specific gas rate for each hole.

Table 1 - Summary of fractures in LW9 at Dartbrook

Hole number	Number of fractures	Hole length fractured (metres)	Fraction of hole stimulated (percent)	Average fracture spacing (metres)	Sand per fracture (kg)
109-13-F1	54	195	72%	3.6	78.3
109-13-F2	50	294	85%	5.9	81.2
109-13-F3	4	18	5%	4.5	100.0
109-9-F1	44	258	85%	5.9	86.1
109-9-F2	54	297	90%	5.5	84.7
109-9-F3	108	366	94%	3.4	95.6
109-9-F4	104	413	94%	4.0	83.4
109-9-F5	77	486	95%	6.3	86.8
109B-6-F1	14	110	26%	7.9	
109B-6-F3	59	373	94%	6.3	71.6
109B-6-F4	56	364	93%	6.5	75.2
Totals/ Average	624	3174			84.3

Overall the gas rate from the unstimulated holes averaged 0.74 litres per metre per minute and 3.6 litres per metre per minute after stimulation. The average stimulation factor was, therefore, 4.9 for all three cut throughs. These averages include hole F3 at 13 cut through, which had only 5% of its length stimulated and hole F1 at 6 cut-through, which was stimulated over only 25% of its length. With more complete coverage of the holes, a stimulation ratio above 5 would be expected to result for conditions in LW9. In Figure 8, holes F1 and F3 show signs of being blocked by debris or sand which then is cleared at 110 to 120 days on the graph. Similarly, F1 and F4 at 6 cut-through (Figure 9) appear to be blocked. The gas from these holes has, evidently, found its way into holes F2 and F5, which were not stimulated but show a large increase in gas rate after the hydraulic fracturing work in the other holes at this site. Such a connection between holes can be formed either through a number of propped hydraulic fractures or directly by a hole-to-hole intersection.

Some of the sand proppant was produced back from the fractures into the drain holes. A portion of this sand was carried into the gas drainage pipe work. Sand production can be controlled by adding sand stabilizers to the proppant when it is pumped. It may also be worthwhile to consider cleaning the entire hole after the fractures are placed since most proppant is produced soon after the stimulation work is carried out. Sand production also becomes less of a problem as the closure stress on the fractures increases.

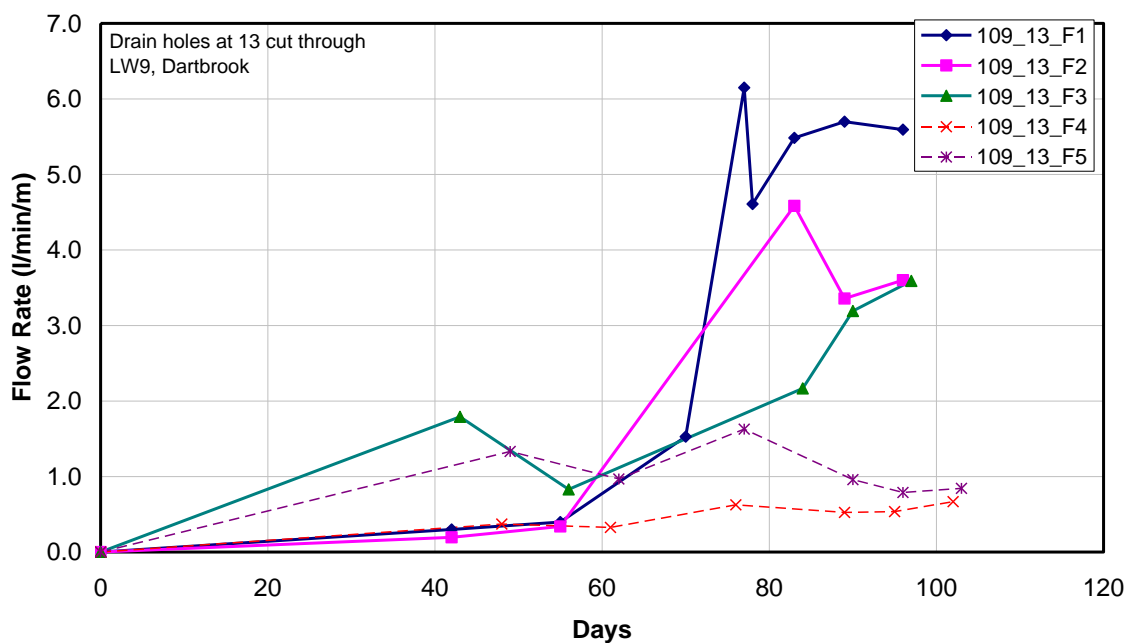


Fig 7 - Specific gas rate from drain holes drilled at cut-through 13.
Dashed lines indicate holes not fracture stimulated.

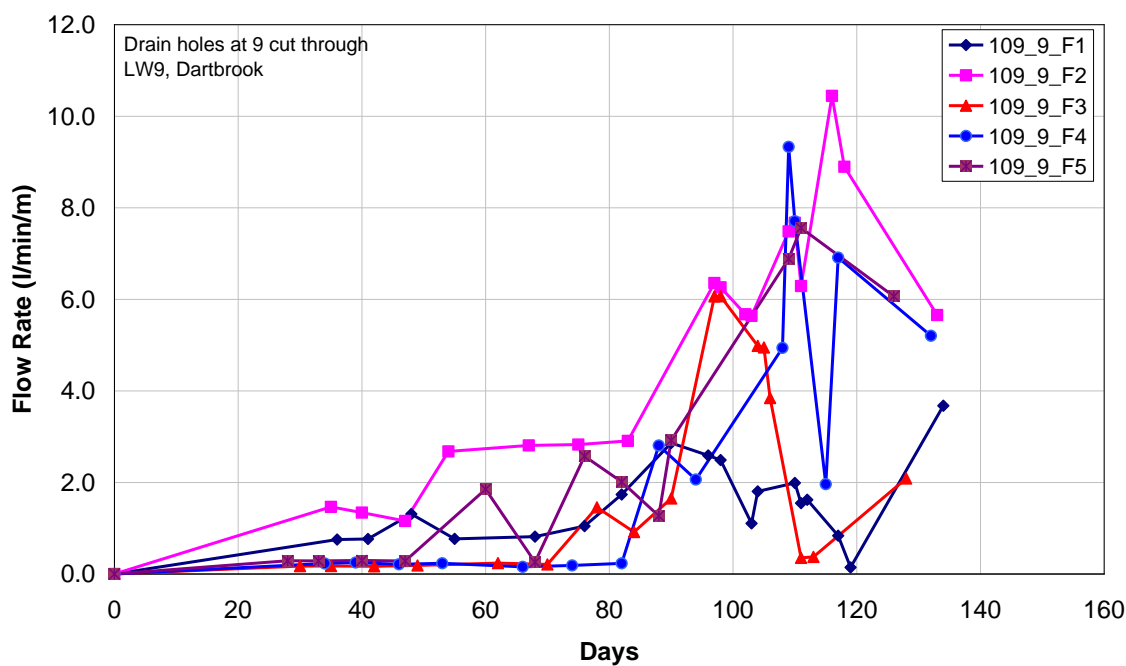
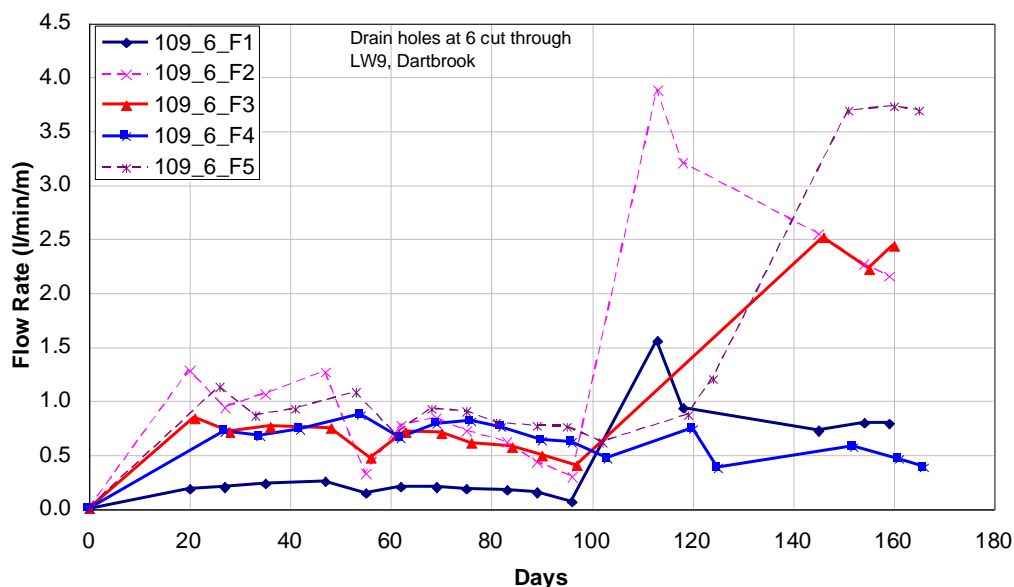


Fig 8 - Specific gas rate from drain holes drilled at cut-through 9. All holes were stimulated.



**Fig 9 - Specific gas rate from drain holes drilled at cut through 6.
Dashed lines indicate holes not fracture stimulated**

DISCUSSION AND APPLICATION TO OTHER MINES

The effect of sand propped hydraulic fractures on the gas drainage rate depends on the seam parameters such as permeability and thickness and on the hydraulic fracture parameters, such as fracture conductivity, propped length and spacing between fractures. If coal seam permeability is higher, the fracture conductivity and propped length must be increased (by pumping more sand into each fracture) if the same stimulation ratio is to be obtained. This fact partly explains why the stimulation work in LW9 provided a stimulation ratio of about 5 while at the trial site in LW8 the stimulation factor was 22 and 180. The coal in LW8 was less permeable and pre-stimulation gas rates were lower than those in LW9, but no change was made to the amount of proppant in each fracture and the average spacing between fractures was actually increased in LW9 (because of the approaching longwall face). Nevertheless, the stimulated drainage holes in LW9 were deemed successful, producing a faster drainage rate which allowed mining of LW9 without any gas-related delays occurring. A stimulation factor of 5 implies that, for example, 1 month of stimulated drainage will drain the same gas volume (but probably from different parts of the seam) as 5 months of unstimulated drainage.

The coal strength and the stress conditions at Dartbrook are such that horizontal drain holes can be drilled with little difficulty and these holes, by and large, remain stable. Good hole conditions made possible the use of standard open-hole inflatable packers to carry out the fracturing work. Initial packer life in this project was considered less than desired and adjustments to the standard inflation pressure and the downhole tool were implemented to increase the number of fracture treatments that could be obtained per packer. The packer manufacturer also supplied stronger packers. In addition, the locations of existing fan array holes were known and, as part of the fracturing procedure, packers were not set near points where the old and new boreholes were close to one another. The holes drilled for the fracturing project were positioned in the upper part of the working section of the seam, to avoid intersection with existing fan array holes. With these changes, packers lasted for about 60 fracture treatments before failing. Higher setting pressure or less stable borehole conditions, which exist at many other mines, will lead to shorter packer life and higher costs for carrying out each fracture stimulation. In many cases, without some form of hole stabilization, it may not be practical to use open-hole packers to place the fractures.

Alternative hole completion methods, such as casing the hole before fracturing, are being investigated (Jeffrey and Mills, 2002) in order to allow sand-propped hydraulic fracture stimulation to be carried out under a greater range of sites and coal seams conditions.

CONCLUSIONS

The trial in LW8 demonstrated that sand-propped hydraulic fractures were effective in stimulating gas drainage rates from horizontal drain holes and the best stimulation resulted for holes drilled so that the fractures extend in a direction perpendicular to the hole axis.

The sand-propped hydraulic fracture stimulation work carried out in LW9 at Dartbrook, on average, increased the rate of gas drainage by a factor of about 5. The increased rate of gas drainage allowed the gas content to be reduced in LW9 to a level that was sufficiently low so that no gas-related delays were experienced in mining the panel. This was a considerable improvement over conditions experienced in mining the previous panel, which was not drained with the aid of sand-propped stimulated drain holes.

ACKNOWLEDGEMENT

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STUDY OF THE MECHANISMS OF COAL AND GAS OUTBURSTS USING A NEW NUMERICAL MODELING APPROACH

Xavier Choi¹ and Mike Wold²

Abstract: During mining or roadway development, the distribution of stress and pore pressure in the coal face and rib around the new opening will change. These changes are usually dependent on the mining history and are related to the rate of roadway development, geometry of the opening, the pre-mining stress and reservoir conditions, the strength of the coal, the adjacent rock strata and major geological structures, and the permeability of the coal. Quasi-static yielding of coal is usually observed at regions of high stress concentration. However, under certain conditions, dynamic failure of coal in the form of an outburst can occur.

The occurrence of coal and gas outbursts and the way they evolve will depend on a number of factors and processes. Under varied mining conditions, some of the factors and/or processes may play a more important role in outburst initiation than others. It can be misleading to attribute the cause of an outburst to a particular factor or process. This is partly because some of the processes are highly non-linear; outburst occurrence may depend on how these processes evolve and interact. The problem becomes more complex because natural heterogeneity of the coal and geological structures also play an important part in the outburst mechanisms.

In the modelling studies presented in this paper, an outburst is considered to consist of three distinct stages: pre-initiation, initiation and post-initiation (or outburst evolution). During the pre-initiation stage, deformation of the yielded coal occurs in a quasi-static manner. Initiation is referred to as the moment in time when the deformation behaviour of the coal/rock/gas system suddenly transforms from being quasi-static to dynamic. The post-initiation stage, or outburst evolution, is characterised by the release of a substantial amount of gas and violent ejection of coal fragments into the mine opening. A description of the numerical modelling approach is given in this paper, and model results, including the post-initiation deformation and fragmentation of the ejected coal, are presented. The effects on outbursts of gas content, gas composition, pressure gradient, coal strength and other factors are discussed.

INTRODUCTION

During mining or roadway development, a change in stress and pore pressure around the new opening occurs. At some stage, an outburst may be initiated which can be related to the size of the opening and/or the proximity to some geological structures. When outbursting occurs, the rock/coal/gas system transforms from a stable to an unstable state with the release of a significant volume of gas over the duration of the outburst. Some mechanisms that enable the system to overcome the energy barrier provided by the strength of the coal must exist. They can be related to major geologic structure, natural heterogeneity, and/or the state of the system. Although an outburst may sometimes be referred to as "instantaneous", the system may, in some outbursts, go through several meta-stable states before reaching the final equilibrium state, and the whole process may take many seconds or minutes. As coal is a soft rock, the amount of strain energy stored as a result of elastic deformation is limited. In order for coal to fail in a violent manner, another form of energy needs to be available. This is supplied by the adsorbed gas within the coal matrix and the compressed gas in the void space such as the cleats and other natural fractures. Pre-mining gas drainage is effective in preventing outbursts by removing this source of potential energy.

After an outburst has been initiated, the outburst coal starts to deform at a high strain rate, and the fragmented coal will tend to behave as a particulate material. Depending on the gas pressure and the availability of new sources of free gas, the gas can cause further failure and fragmentation of the coal, and the expanding gas will

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provide a drag force to propel the fragmented coal further into the mine opening. During an outburst, fluid flow in the outburst coal will transform from flow in a fractured porous medium to dense particle flow where the consideration of inter-particle contact and collision is important, and then to dilute particle flow as the ejected coal fragments and particles move further away from each other. At this stage, the momentum transfer between the coal fragments and free gas emitted from the mine face becomes important.

NUMERICAL MODELLING APPROACH

Numerical modelling of outbursts requires the quantification of a number of processes and factors and their interactions shown in Figures 1 and 2. These include gas desorption, mass transfer between adsorbed gas and free gas, flow of water and gas within the cleats, macropores and other large scale fractures, coal deformation and failure, coal fragmentation, gas dynamics and transport of outburst coal (particle flow).

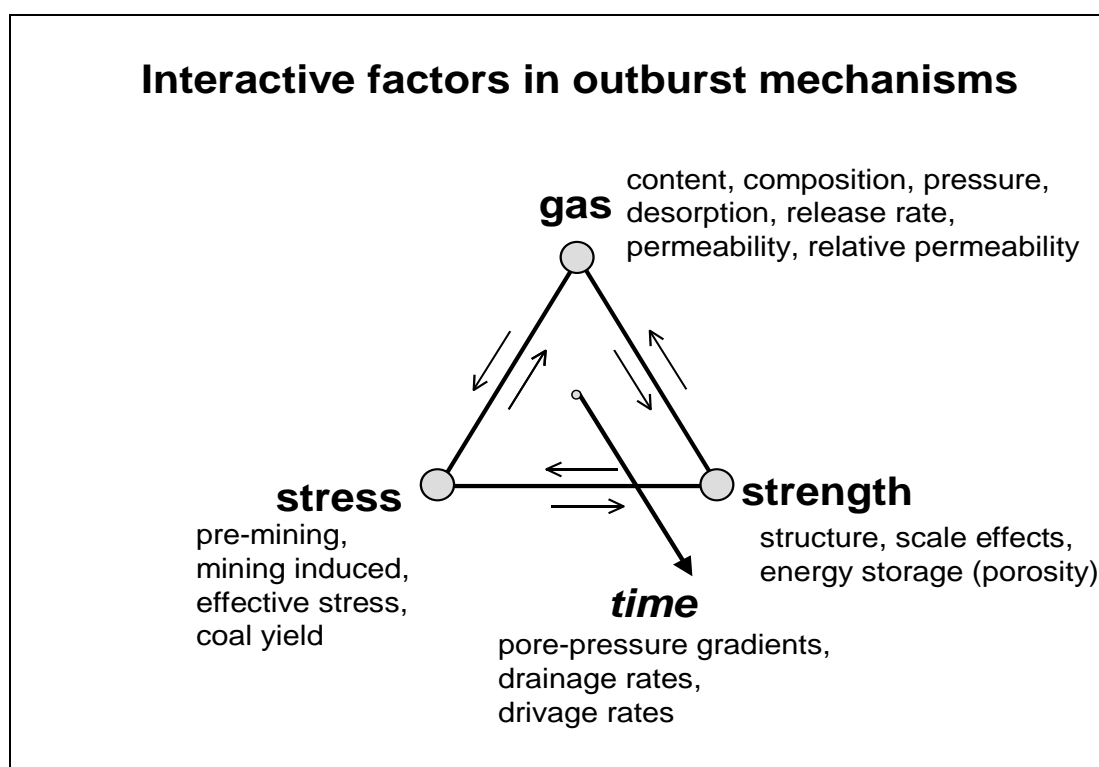


Fig 1 - Schematics of interactions of reservoir, geomechanical and time rate variables contributing to outburst mechanisms

A numerical model for outburst initiation was developed by linking a geomechanical model (Choi, 1984; Choi et al., 1991, 1992; Choi and Tan, 1998) with a coalbed methane reservoir simulator (Spencer et al., 1987; Stevenson et al., 1994; Stevenson, 1997). The model has been applied to study the mechanisms for outburst initiation (Choi and Wold, 2001a and 2001b) and to identify which are the key variables in outburst initiation, and which are the less important variables (Wold and Choi, 1999). The capability of the numerical model for outburst initiation has been extended to include development of additional constitutive models for the important factors and processes that occur during the post-initiation evolution of outbursts (Choi and Wold, 2002).

In the modelling studies, pre-initiation quasi-static yielding type failure of coal is distinguished from the post-initiation failure of outburst coal, as the latter is much more dynamic and violent, involving fragmentation of the coal and rapid release of gas. The quasi-static yielding failure is modelled using macroscopic plasticity theory with softening, and the fragmentation process is modelled using continuum damage mechanics. The effects of particle size on the rate of mass transfer between adsorbed gas and free gas during fragmentation are studied using both the more conventional two-stage desorption diffusion processes and a new hypothesis based on the kinetics of desorption. The violence of the model outburst is estimated using the momentum of the outburst

coal. The potential volume of gas that can be released immediately after an outburst is estimated based on the amount of desorbable gas in the ejected coal and the gas flux from the faces.

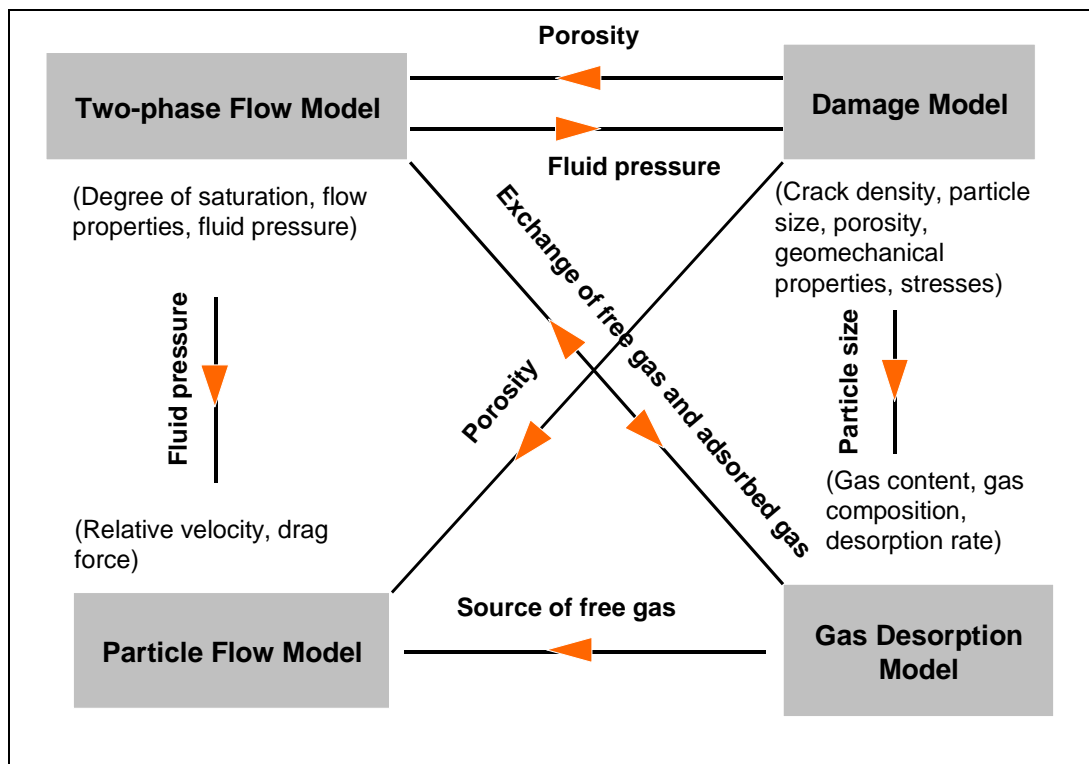


Fig 2 - Schematics of coupling of major processes in modelling outburst evolution

GAS ADSORPTION AND DESORPTION

As most of the gas in coal exists in the adsorbed state on the surface of micropores within the coal matrix, it is important to understand the gas adsorption/desorption properties of coal and the mechanisms of mass transport between adsorbed gas and free gas, and how the adsorbed gas may become available as free gas to provide the energy during an outburst.

The adsorption properties of coal are normally represented by the adsorption isotherm which shows the amount of gas adsorbed at a certain temperature and partial pressure. For a single component gas, the Langmuir adsorption isotherm (Langmuir, 1918) is usually used for coal and is given by

$$\frac{V}{V_L} = \frac{p_g}{P_L + p_g} \quad (1)$$

where V is the volume of gas adsorbed per unit mass of coal at pressure p_g , V_L is the Langmuir volume, which is the volume of gas needed for monolayer coverage of the surfaces of the micropores per unit mass of coal, and P_L is Langmuir pressure, defined as the pressure at which the volume of gas adsorbed is half the Langmuir volume.

The extended Langmuir model (ELM) is the most common approach used to represent multicomponent gas adsorption in coal. This is based on direct extension of the single component isotherm to a multicomponent system giving the analogous expression

$$\frac{V_i}{V_{Li}} = \frac{b_i p_{gi}}{1 + \sum_{j=1,n} b_j p_{gj}} \quad (2)$$

where subscript i represents the i^{th} gas component, b is the reciprocal of P_L , and n is the number of gas components.

After initiation, the way an outburst evolves can be strongly influenced by the amount of free gas in the outburst coal. It is therefore important also to understand the kinetics of the adsorption and desorption processes, and the rate of transformation of adsorbed gas into free gas during an outburst.

In the Langmuir treatment of the kinetics of gas adsorption and desorption, equilibrium condition is obtained by equating the rate of adsorption to the rate of desorption as follows:

$$\frac{P}{\sqrt{2\pi mkT}} (1 - \theta) = \frac{e^{-Q/kT}}{\tau_o a} \Theta \quad (3)$$

where P is gas pressure, m is the mass of the gas molecule, k is Boltzmann constant, T is absolute temperature, θ is fraction of available sites occupied ($0 \leq \theta \leq 1$), Θ is total surface area of adsorption sites, Q is energy of adsorption, a is the area of an adsorption site, and τ_o is the average resident time.

It was observed by Groszek (1982) that, at very low pressure, very little energy is required for the adsorbed molecules to move away from the interface, and the rate of desorption can be very high.

In the conventional approach, the rate of mass transport between adsorbed gas in the matrix and free gas in the cleats and other fractures is assumed to involve a two-step process corresponding to desorption/diffusion within the coal matrix, followed by long-range flow in the cleat system. Desorption of gas from the coal surface occurs at a much faster rate than the diffusion of the gas through the coal matrix. The rate of diffusion is often assumed to be sufficiently represented by Fickian diffusion which is usually the rate-limiting step in the desorption process.

If the coal matrix can be assumed to be made up of spheres with a characteristic radius (Ancell et al., 1979, 1980), the rate of mass transport between free gas in the cleats and adsorbed gas within the coal matrix is given by

$$\frac{c_d}{r^2} \frac{\partial}{\partial r} \left(r^2 \frac{\partial c}{\partial r} \right) = \frac{\partial c}{\partial t} \quad (4)$$

where c_d is the micropore diffusion coefficient, c is gas concentration, and r is radial distance from the centre of the sphere. The rate of mass transfer therefore depends on the characteristic radius of the spheres, the micropore diffusivity, the partial pressure of the free gas in the cleats, and the gas content (or concentration of gas in the matrix spheres).

However, considering that small fragments and fine particles can be generated very rapidly during an outburst with loss of confinement and rapid drop in pressure around the particles, rapid gas desorption will occur. At the time when the particles have just been formed, it is assumed that the gas content and pressure within the particle is uniform. An amount of free gas resulting from the formation of the new surfaces will be available, together with the gas in the bulk phase existing in the pores that are connected to the new surfaces. The adsorbed gas in the very small particles may transform into free gas in a very short time. For the larger particles, a high pressure gradient will exist within the particles, and viscous flow, in addition to Fickian diffusion, will become an important mass transport mechanism within the matrix. Within the particles, it is assumed that the viscous flow can be adequately represented by Darcy's law :

$$u_i = -\frac{k}{\mu} \frac{\partial p}{\partial x_i} \quad (5)$$

where u_i is apparent velocity, k is permeability, μ is viscosity, x_i is spatial coordinate.

In 3-dimensional Cartesian space, taking into account desorbed gas as a source for free gas, and assuming that the change in porosity and gas density over a small time step can be ignored, the continuity equation is given by

$$\frac{\partial V}{\partial t} = \frac{1}{\rho_s} \nabla \cdot u_i = \left(\frac{V_L}{P_L + p_g} \right) \left(\frac{p_g}{P_L + p_g} \right) \frac{\partial p_g}{\partial t} \quad (6)$$

where ρ_s is the density of coal.

Equation (6) can be used as the source term in the mass balance and momentum equations to calculate gas flux and drag force acting on the coal fragments.

COAL DEFORMATION AND FAILURE – CONTINUUM STATE

During mining of a roadway, stress concentration will occur in certain regions around the mine opening. Under sufficiently high stress, initiation and growth of microcracks can occur. Initially, this will tend to occur in a smooth and gradual manner, and the differences between the states at each stage of progressive failure can be very small, with a smooth transition from intact to failed state. Failure at this stage may manifest itself as plastic deformation resulting from distributed damage evolution, strain localisation, subcritical crack growth and coalescence, and degradation of mechanical properties including strength and bulk and shear moduli. The degree of “plastic” damage just prior to outburst occurrence can have important influence on the fragmentation process during post-initiation outburst evolution. However, the plastic damage is rate-independent while the post-initiation damage (or fragmentation) is strongly rate-dependent. It is therefore important for the two distinct stages of failure to be modelled using different approaches.

The Mohr-Coulomb criterion is used to represent the shear failure of coal. In the modelling studies, after yielding has occurred, the total strain is assumed to consist of three components: the elastic, plastic and damage components.

$$\varepsilon = \varepsilon^e + \varepsilon^p + \varepsilon^d \quad (7)$$

Damage (or material degradation) is expressed as a function of total plastic strain.

$$D = 1 - e^{-\alpha \varepsilon_p} \quad (0 \leq D \leq 1) \quad (8)$$

The elastic properties of the coal continues to change as the degree of damage increases.

$$c_d = c_o (1 - \chi D) \quad (9)$$

$$\phi_d = \phi_o (1 - \beta D) \quad (10)$$

$$E_d = E_o (1 - \gamma D) \quad (11)$$

$$\nu_d = \nu_o (1 - \kappa D) \quad (12)$$

where c is cohesion, ϕ is angle of internal friction, subscript o represents the initial (undamaged) material property, subscript d represents the damaged material property, and the values for α , χ , β , γ , and κ have to be determined in the laboratory under triaxial compression to beyond the peak strength, under strain-controlled conditions.

COAL FRAGMENTATION

When an outburst has been initiated, macroscopic continuum damage mechanics is adopted to model the creation of new cracks, and the propagation and coalescence of new and existing cracks. Although it is less rigorous than the micro- and meso-damage approaches, it provides a computationally feasible alternative for some complex problems such as the modelling of rock fragmentation. The constitutive models are based mainly on experimental observations at the macroscopic scale (Grady and Hollenbach, 1979; Kipp et al., 1980). It is assumed that the fragmenting coal can be treated as an isotropic, continuous and homogeneous material with pre-existing microcracks. The outburst coal is considered to deform at a high strain rate, and confinement will be lost after the coal has detached from the face. The coal will continue to deform under the influence of the free gas in the pore space, and is subjected mainly to tensile stress induced by the compressed gas. The propagation of the pre-existing microcracks occurs mainly under mode I (opening mode), and damage is isotropic. Based on the above assumptions, damage can be sufficiently represented by a scalar variable. It is also assumed that a critical strain criterion can be used to predict the onset of fragmentation. The assumption of isotropy enables the use of a model which is tractable and requires only a few model parameters that are physically meaningful. Most importantly, their values can be measured in the laboratory.

In the model, the degree of damage to the outburst coal is a function of strain rate. The coal is assumed to be completely destroyed (or lose its strength and stiffness) when the crack density has exceeded a certain critical value for a particular type of coal. This is reflected by the values of effective bulk modulus and Poisson's ratio both approaching zero. The size of the fragments is assumed to be directly related to the maximum strain rate experienced during the loading history. Based on the above assumptions, the model can therefore be represented by the following equations to provide an initial estimate of particle size. Subsequent size evolution may depend on other factors such as collision among particles, but they are not considered here.

$$F = 1 - e^{-\alpha C_d} \quad (13)$$

$$\nu_e = \nu(1 - \beta F) \quad (14)$$

$$\text{where } \lim_{C_d \rightarrow \infty} \beta = 1 \quad (15)$$

$$K_e = (1 - \gamma F) K \quad (16)$$

$$r = \omega \left(\frac{K_{IC}}{E_{eff} R} \right)^{2/3} \quad (17)$$

$$c_d = f(\epsilon_{dp}) \left(\frac{K_{IC}}{E_{eff} R_{max}} \right)^2 \quad (18)$$

C_d is crack density, F is regularised damage parameter, ν is Poisson's ratio, K is bulk modulus, K_{IC} is mode I fracture toughness, r is radius of fragment, R is strain rate, R_{max} is maximum strain rate during loading history, ϵ_{dp} is cumulative irreversible strain due to plastic deformation and damage, and subscript e refers to effective material property. α , β , γ , ω , and E_{eff} are empirical constants and material properties that need to be determined through laboratory measurements.

Gas dynamics and transport of outburst coal

Particle flow refers to the two-phase (particles and fluid) flow where solid particles are immersed in a continuous carrier fluid (liquid or gas). These correspond to the fragmented coal and free gas during an outburst in which the fragmented coal is expelled into the mine opening by the gas. In modelling particle flow, two conditions have to be distinguished – dilute flow (the solid particles are highly dispersed) or dense flow. For

dilute flow, the solid and fluid mixture may be treated as single-phase fluid with modified rheological properties. However, for dense flow, which is more related to the early stages of coal ejection from the face, the mixture has to be treated as a two-phase medium, taking into consideration the interaction between the coal fragments and the continuous fluid phase.

As the number and size of the particles (or coal fragments) are not constant, but constantly evolving, and due to the very large number of fragments, it is almost impossible to model them explicitly. Instead, the coal fragments and the carrier fluid are treated as two super-imposing continua. For such a case, the particle flow can be represented by the Navier-Stokes equations for compressible gas, and the interaction between the coal fragments and the carrier fluid can be represented by suitable interfacial exchange terms in the momentum equations

$$-2\mu_s \nabla \cdot (\phi_s \varepsilon(u_s)) + \rho_s \phi_s (u_s \nabla) u_s = -\phi_s \nabla p_f + M + \phi_s F_b \quad (19)$$

$$-2\mu_f \nabla \cdot (\phi_f \varepsilon(u_f)) + \rho_f \phi_f (u_f \nabla) u_f = -\phi_f \nabla p_f + M + \phi_f F_b \quad (20)$$

The drag force (Ergun, 1952) of the gas on the coal fragments (assumed to be spherical) is given by

$$F_d = f(n, Re) \frac{\pi}{4} d_p^2 \frac{\rho_g u_r^2}{2} \quad (21)$$

where

$$Re = \frac{u_r d_p \rho_g}{\mu_f} \quad (22)$$

ϕ is volume fraction, u is velocity, μ is viscosity, ρ is density, ε is stretch tensor, p is pressure, M is interfacial momentum transfer, F_b is body force, n is local porosity, Re is Reynold's number, d_p is average particle diameter, u_r is relative velocity, subscript f is fluid phase, and subscript s is solid phase.

To model the violence of outbursts, it is necessary to calculate the initial momentum and the initial drag force of the expanding free gas on the motion of the coal fragments. After expulsion from the face, the coal fragments are travelling at reasonably high speed and desorbing gas at the same time. Further fluid-particle interaction is not tracked. It is considered that the initial momentum gives a good indication of the violence of the event.

MODEL DESCRIPTION

All the model studies were conducted using a 2-dimensional geometry. The seam is at a depth of 500 m. The roadway is 5 m wide. The coal was assumed to be homogeneous. Heading advance of 25.0 m was modelled in five stages, each of 5.0 m, and each of uniform time increment to simulate a rate of 14.0 m/shift. Geological structures such as faults and dykes were included in some of the studies (see Figure 3). The input values for some of the model parameters are shown in Tables 1 and 2.

Table 1 - Values of parameters used in strain softening model

α	χ	β	γ	κ
100.0	1.0	0.0	0.0	0.0

Table 2 - Values of parameters used in fragmentation model

Critical strain (fragmentation)	α	β	γ	f (ε_{dp})
0.3	1.0	0.0	1.0	ε_{dp}

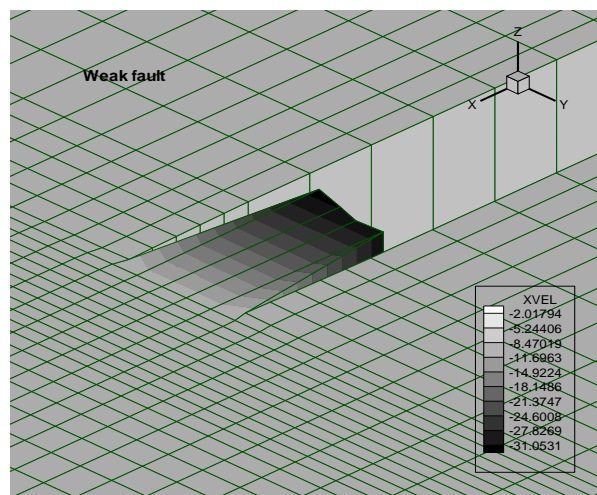


Fig 3 - Velocity (m/s) contours during model outburst when mining through weak fault.

MODEL RESULTS

The Effects of Coal Strength

It appears that, if the strength of the coal is high enough, outburst will not occur (see Figure 4). Also, the degree of violence seems to decrease with an increase in coal strength (see Figures 4 and 5). Furthermore, model failure of the very strong coal occurred mainly in a small region behind the face, and stress arching in the intact coal further back behind the face provided a stabilising mechanism against outburst initiation.

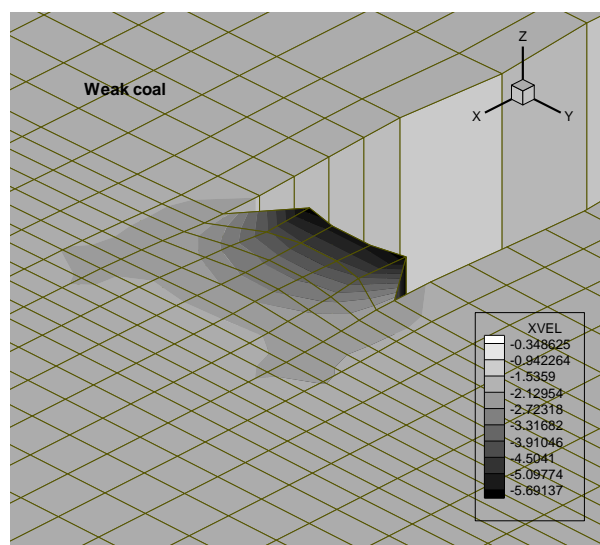


Fig 4 - Velocity (m/s) contours during model outburst in weak coal

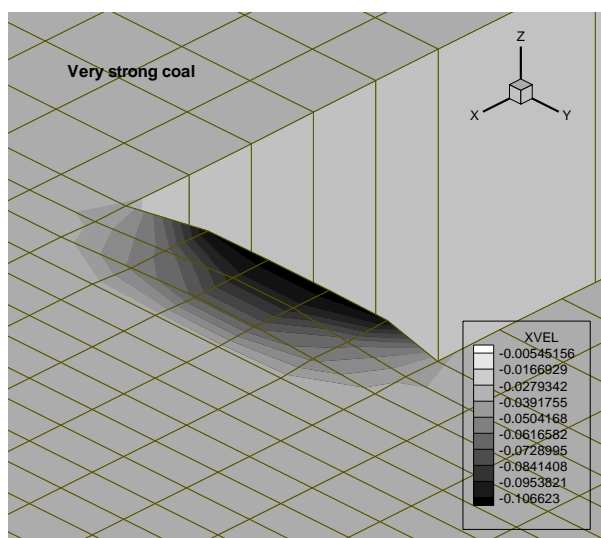


Fig 5 - Velocity (m/s) contours during model outburst in very strong coal

The Effects of Fragment Size and Gas Desorption Rate using the Conventional Approach

The effects of fragment size on the rate of desorption and the mass transport of desorbed gas to the cleats were first modelled by varying the desorption time constant using the conventional approach (see Equation 4). It can be seen from Figures 6 and 7 that the difference in pressure gradient at the face between seams saturated with either CO₂ or CH₄ is less than 10%.

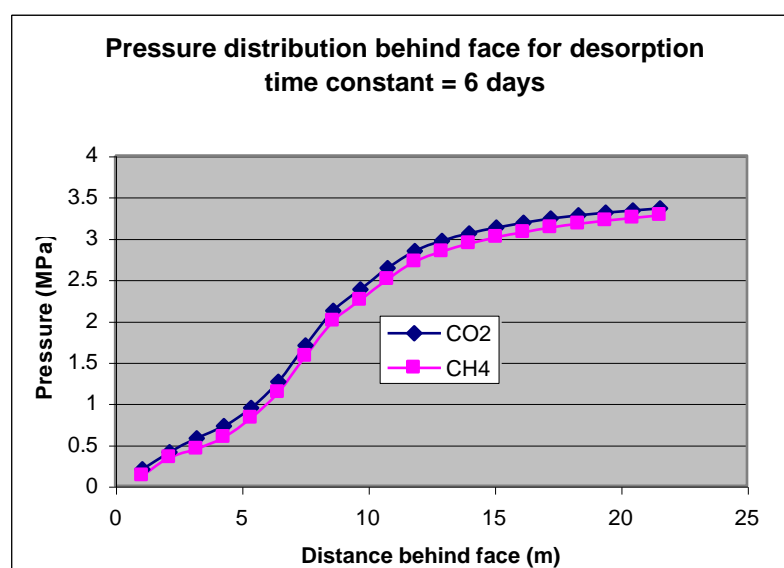


Fig 6 - Pressure distribution behind face for desorption time constant = 6 days

This does not provide a good explanation for the observed difference in violence between CO₂ and CH₄ outbursts. Also, there was significant decrease in the pressure and pressure gradient behind the face as the desorption time constant was reduced from 6 days to 8.5 seconds. This would imply that, based on geomechanical considerations, an outburst is less likely to be initiated with a decrease in desorption time

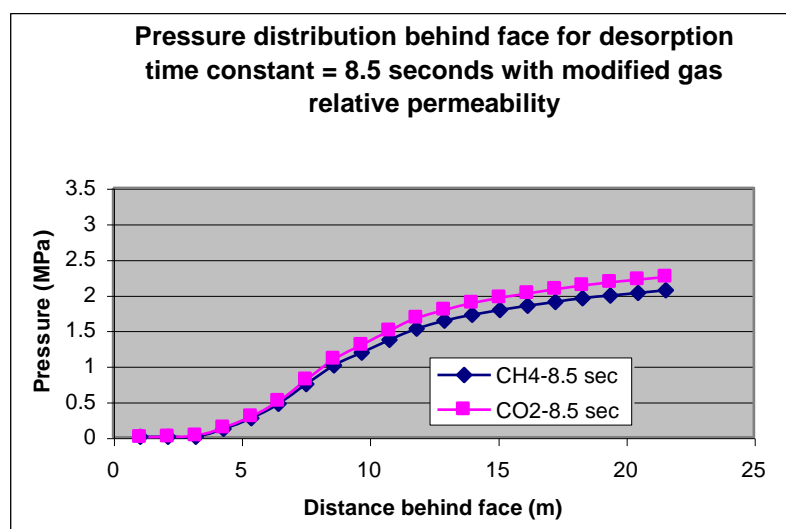


Fig 7 - Pressure distribution behind face for desorption time constant = 8.5 sec with modified gas relative permeability

strain rate, and the voids in the coal will expand rapidly, leading to almost full saturation of void space by gas and significant increase in permeability and the relative permeability factor. It was found in the modelling studies that this could lead to a very significant increase in drag force with a great influence on the post-initiation behaviour of the outburst coal. The results suggest that the likely impact of an increase in desorption rate is on post-initiation behaviour during outburst evolution.

The Effects Of Gas Diffusion Mechanism And Gas Composition Using The New Approach

If we assume that desorption is almost instantaneous and that viscous flow within the coal fragments becomes a very important mass transport mechanism under a high pressure gradient, the gas flux at the face for 100% CO₂ can be up to 3 times that of 100% CH₄ in a coal seam with the same initial reservoir pressure. From the model results, based on the gas flux computed at the face for the weak coal, relative velocities between the fragments and the gas were estimated to be about 5 ms⁻¹ and 15 ms⁻¹ respectively for the cases of CH₄ and CO₂, ignoring the greater amount of CO₂ that might be available due to the creation of the new surfaces. The effects of the higher drag force on outburst violence (increased momentum resulting from greater acceleration) is shown in Table 3. This may explain why CO₂ outbursts are in general more violent than CH₄ outbursts. However, the model results still need to be validated against data from laboratory tests or field observations. The model results suggest that CO₂ may not increase the likelihood of an outburst, but that its influence is mainly to increase the violence of an outburst after it has been initiated. It should be noted that, based on the model results, the controlling factors for outburst initiation are pressure and pressure gradient, and the former is directly related to gas content through the desorption isotherm. This would imply that, depending on the gas adsorption properties of the coal, the reservoir conditions for outburst initiation may correspond to different gas contents. Unless there is significant influence of CO₂ on coal strength, or the amount of dissolved CO₂ in the pore fluid can provide a significant amount of free gas during an outburst, the model results would imply that, if other geological factors are similar, CO₂ outbursts would occur at a higher gas content compared to CH₄ outbursts, and that CO₂ outbursts, once initiated, are likely to be more violent.

Table 3 - Acceleration caused by drag force

Radius of Particle (mm)	Relative Velocity (ms ⁻¹)	Drag Force (N) * 10 ³	Acceleration (ms ⁻²)
5	15	7.22	9.84
5	10	4.81	6.56
5	5	2.40	3.28

The effects of geological structures

For structures such as weak faults with high permeability, because of enhanced connectivity to adjacent coal (see Figure 8), pressure in the structure can stay high when the face of the advancing roadway is very close because of the inflow of gas from the coal to the structure. This can lead to piping and the formation of cavity aligned along the structure as was observed in some of the samples during the laboratory cavity completion tests (Wu et al., 1996). Outburst of this kind of structure can be violent because of the large amount of gas that is available to drive the outburst.

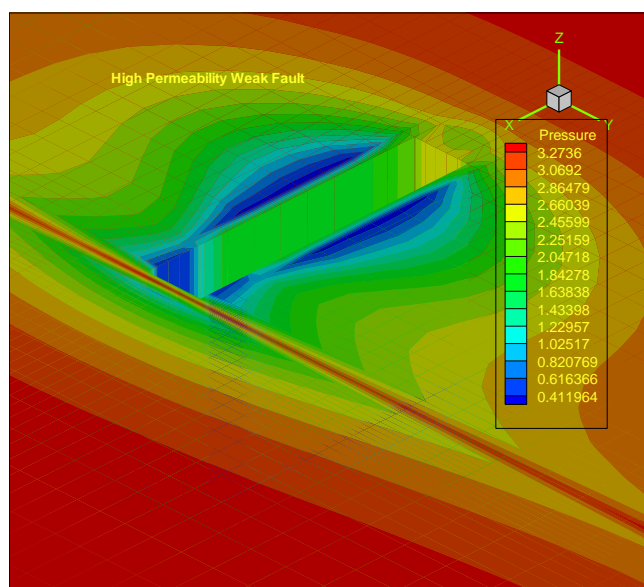


Fig 8 - Pressure distribution in weak fault behind face

Volume of outburst coal and gas

Damage reduces the strength and stiffness of the coal. By using the values for the parameters in the damage models and the computed fragment size as shown in Table 4, the volume of outburst coal appears to be smaller for stronger coal (see Figures 9 and 10). The volume of released gas associated with an outburst can be estimated based on the amount of desorbable gas in the ejected coal and the gas flux at the faces. However, before applying the model for quantitative prediction in the field, the model parameters have to be calibrated for different types of coal, taking into account size effect and heterogeneity.

Table 4. Values of parameters used in fragmentation model and predicted range of particle sizes

ω ($s^{-2/3}$)	(K_{IC}/E_{eff}) (\sqrt{m})	Coal type	Smallest particles (m)	Largest fragments (m)
1.0	0.1	Weak	0.093	1.0
1.0	0.1	Uniform	0.115	1.91
1.0	0.1	Strong	0.171	2.18
1.0	0.01	Weak	0.02	0.22
1.0	0.01	Uniform	0.025	0.41
1.0	0.01	Strong	0.037	0.47
1.0	0.001	Weak	0.0043	0.046
1.0	0.001	Uniform	0.005	0.088
1.0	0.001	Strong	0.008	0.101

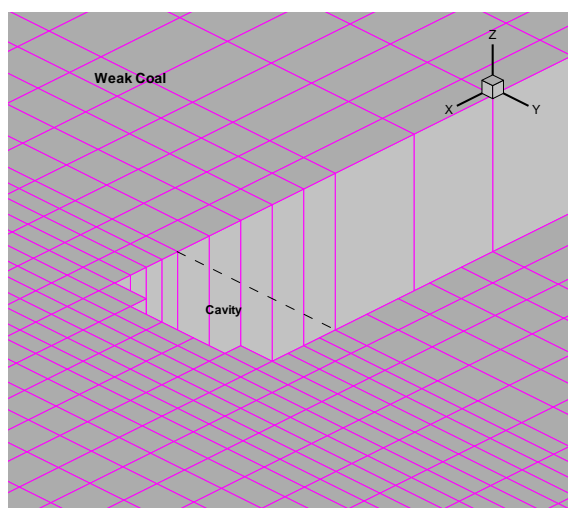


Fig 9 - Model prediction of cavity size in weak coal under model conditions

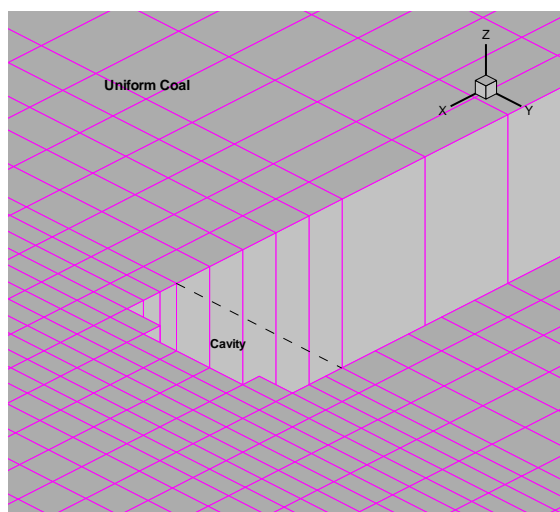


Fig 10 - Model prediction of cavity size in uniform (medium strength) coal under model conditions

DISCUSSION OF RESULTS

The current practice for outburst control and prevention is by gas pre-drainage to below a threshold value of gas content (THV). This can have significant impact on the rate of mine development and production. It is well recognised that permeability is a key factor governing the efficiency and time scale of gas drainage. On the time scale of heading development, i.e. hours or days, low permeability has an important impact on the depth of face drainage and the development of gas pressure gradients, and therefore on conditions for outburst initiation. However, once the outburst event has initiated, virgin permeability is of relatively less significance to the mechanism and the violence of the event.

The model results suggest that strong coals are less prone to outbursts than weaker coals. However, the friability and the rate of damage accumulation under increasing load can affect the rate of strain energy release and stress redistribution. This can have an impact on both the likelihood and the degree of violence of outbursts. Changing the desorption time-constant has little impact on outburst proneness, however, it affects the post-outburst behaviour by providing energy from the expanding gas to drive an outburst and transport the outburst coal through greater drag force. Outbursts associated with geological structures appear to be more violent compared to outbursts in uniform coal.

With respect to gas composition, the model results do not show that outburst will occur at lower gas content for CO₂ compared to CH₄, and in that respect, are compatible with the data collected by Lama (1996), regardless of the superposed THV's. However, at the same initial reservoir pressure, the more violent nature of CO₂ outbursts compared to CH₄ outbursts may be due to the higher rate of mass transport of desorbed gas from the matrix pores to the cracks and other fractures as the coal fragments. In the model, the volume of released gas associated with an outburst can be estimated based on the amount of desorbable gas in the ejected coal and the gas flux at the faces. A mechanism that has been ignored in the current model is the change in sorption capacity of intact coals when they have failed either in shear or tension. It is reported in the literature that the sorption capacity of sheared coal can be much higher than that of intact coal. Although yet to be confirmed through laboratory studies, this could be one of the reasons why the amount of gas released during an outburst is apparently higher than the amount of gas available based on the gas content of the ejected coal. Also, in the modelling conducted to date, the amount of dissolved CO₂ in the pore water has been ignored; it is perhaps worthwhile to estimate the amount of CO₂ that could have been released from solution when the pressure dropped from the initial reservoir pressure to atmospheric during an outburst. The possible influence of CO₂ on the strength and deformation behaviour of coal has also been ignored. If CO₂ does reduce the strength of coal as reported in the literature, it can increase both the proneness to outburst and the degree of violence. This remains to be confirmed by laboratory tests.

CONCLUSIONS

A model has been developed which takes into account the major processes and mechanisms that can influence both outburst-proneness and post-initiation outburst behaviour. The model has been applied to simulate the effects of gas diffusion mechanisms in the coal matrix, coal strength, coal damage, geological structures and gas composition on outbursts. The model is able to provide estimates of the amount of outburst coal and released gas if laboratory data is available for the model input parameters.

The model results do not indicate that coal seams rich in CO₂ are more outburst prone than seams rich in CH₄. On the contrary, because of the higher adsorption capacity of coal to CO₂ relative to CH₄ at the same partial pressure, the results suggest that outbursts tend to be initiated at higher gas content for CO₂ compared to CH₄. On closer examination, the observational data collected by Lama (1996) do not appear to show unequivocally that outburst will occur at a lower gas content for CO₂ compared to CH₄.

The model results suggest that CO₂ outbursts tend to be more violent mainly because of the greater adsorption capacity for CO₂ under the same partial pressure compared to CH₄. After an outburst has been initiated, there is a higher rate of mass transfer of CO₂ from the adsorbed state to the free state, the greater amount of free CO₂ provides more energy to fragment the coal and a greater drag force to act on the fragmented coal, leading to a more violent outburst compared to CH₄ (The possibility of CO₂ causing a reduction in the strength of coal as another contributing factor needs further study).

The likelihood of outbursts appears to be less in stronger coal. Also the degree of violence decreases with an increase in coal strength. It appears that, if the strength of the coal is high enough, the outburst process can be subdued.

For the assumed properties of the structures in the models, the outbursts associated with geological structures such as dykes and faults tend to be more violent compared with uniform coal.

Damage reduces the strength and the stiffness of the coal. The results show that, during an outburst, a substantial amount of energy can be released suddenly, together with transfer of load to the adjacent more intact material. This can cause the adjacent coal to fail and burst. This would suggest that the likelihood of outbursts may increase when the coal is more friable. (The influence of permeability of more friable coal needs further investigation).

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THE INFLUENCE OF GAS ENVIRONMENT ON COAL PROPERTIES- EXPERIMENTAL STUDIES ON OUTBURST CONTROL

Naj Aziz¹, Ian Porter¹ and Farhang Sereshki¹

ABSTRACT: The volumetric changes in the coal matrix (Coal Shrinkage) and permeability properties under various gas environment conditions were studied in the laboratory. The shrinkage and permeability of coal were examined with respect to changing gas type and confining pressures. The shrinkage tests were carried out in high-pressure bombs while the permeability study was conducted in a specially constructed high pressure chamber. Methane (CH₄), carbon dioxide (CO₂), nitrogen (N₂) and a 50% mixture of CO₂/CH₄ gas were used in the study. The tests showed that under different pressure levels gas type affected permeability and shrinkage characteristics of coal.

INTRODUCTION

The composition of the gas stored in coal is highly variable, ranging from pure methane to pure carbon dioxide. These variations are mainly related to the geological structure and depth of the coal deposit. The matrix structure of coal is characterized by both micropores <2 nm and macropores >50 nm in size. The storage of methane in coal structure occurs in two different forms, firstly by sorption into pores and microfractures, and free gas. Almost 95% of stored gas in coal is in the adsorbed state as a monomolecular layer on the surfaces of fissures cracks and cleavages and only a small percentage (<5%) is in free state. The level and easiness of gas sorption from coal seams is influenced by moisture, temperature, structure, porosity and a permeability of coal. Methane and other gases will flow out of the coal pores if there is a pressure gradient acting as a driving force. However, the easiness of gas removal from coal is dependent upon the type of the gas and coal petrography and according to Bartosiewicz and Hargraves (1985) coal has higher a permeability to methane than to carbon dioxide.

Another aspect of gas removal from coal is coal matrix volume change. According to Gray (1987) the shrinkage of coal matrix associated to desorption opens up the cleats and results in an increase in coal permeability. Gray also noted that the degree of coal shrinkage with respect to overburden stresses can also influence coal porosity and permeability. Harpalani and Chen, (1992) showed that there was a linear relationship between the coal matrix volumetric strain and the quantity of gas released. Also St George and Barakat, (2001), in their studies on New Zealand coal, found that the shrinkage coefficients of coal matrix in CO₂ gas sorption was in excess of 4 times of CH₄.

Clearly, there remains considerable interest in evaluating the permeability and coal matrix volume changes in Australian coals. Accordingly, the programme of study reported in this paper is intended to show the influence of gas type and pressure on both the coal permeability and the volumetric changes in an Australian coal. The tests were made under various gas types and gas pressure changes. The permeability and volume change experiments were conducted in separate apparatus specifically designed and constructed for each test.

COAL PERMEABILITY TESTS

The permeability measurement of coal, under different loading conditions and gas type, was studied in purpose designed pressure chamber. A general schematic diagram of this apparatus is shown in Figure 1. Constructed at the University of Wollongong, the equipment incorporated facilities to carry out the following:

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- Apply and monitor axial load on coal samples placed in the pressure chamber.
- Monitor strain in coal by using strain gauges mounted on the sample.
- Charge and maintain circumferential gas pressures around the coal sample, and
- Monitor gas flow rate through the coal at suction.

The gas pressure chamber consisted of a rectangular prism of cast iron with removable front and back plates. Its dimensions were 110 mm x 110 mm x 180 mm. The viewing windows were made of 20 mm thick glass in a cast iron frame. Access to the chamber was possible by unbolting the front steel frame with the glass window. A total of 24 bolts secured the frame to the chamber. The chamber was made leak proof by inserting packers between the frame and the box as well as fitting "O" rings around the loading shaft situated at the top of the chamber.

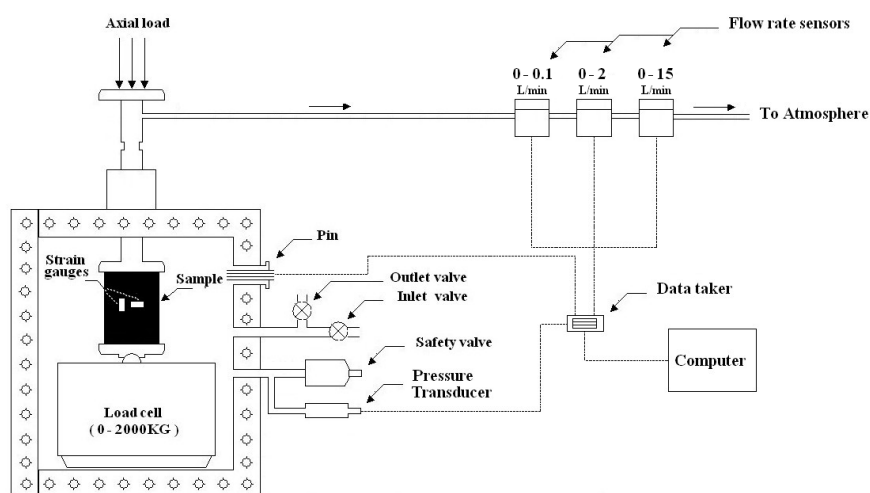


Fig 1 - Schematic layout of the permeability test equipment

Inside the gas pressure chamber were two load plates. The top plate was connected to the universal thrust socket via the central axis, which transferred the applied axial load to the sample. The bottom plate had a hemispherical, concave seat, which rested on a ball bearing at the top of a 40 kN capacity load cell. The bottom plate assemblage served first to transfer the load from the sample to the load cell, and also to pivot and align the coal sample in the event that the top and bottom surfaces of the coal sample specimen were not parallel. Lips on the loading plates prevented any lateral movement of the coal sample during testing.

The 54mm diameter, 50mm long core samples were first centrally drilled with a 5.6 mm hole. Two sets of axial and lateral strain gauges were mounted on the sides of each sample. The coal sample was then placed between the loading plates inside the pressure chamber and axially loaded. Loading of the coal sample was achieved via a universal torque. Gas was then charged into the sealed pressure chamber at a pressure of 3 MPa and maintained constant for a period of one week to allow the coal to be sufficiently saturated. The strain was recorded for this period. It was found that there was a little change in strain after this time. Change in the sample axial and lateral dimensions due to gas sorption were monitored by two sets of strain gauges. Once the sample was fully saturated, the release valve was opened and released gas passed through various flow meters of different flow rates consisting of (a) low flow range 0-100 ml/min, medium range (0-2 l/min) and high range (0-15 l/min). Information from load cell, strain gauges, flow meters and others were monitored in a data-taker data-logger connected to a PC Unit.

The programme of permeability tests consisted of testing each coal sample in a variety of mine gases and under different axial loading conditions and gas pressures. Figure 2. shows the sequence of changing gas pressures and sample loading conditions:

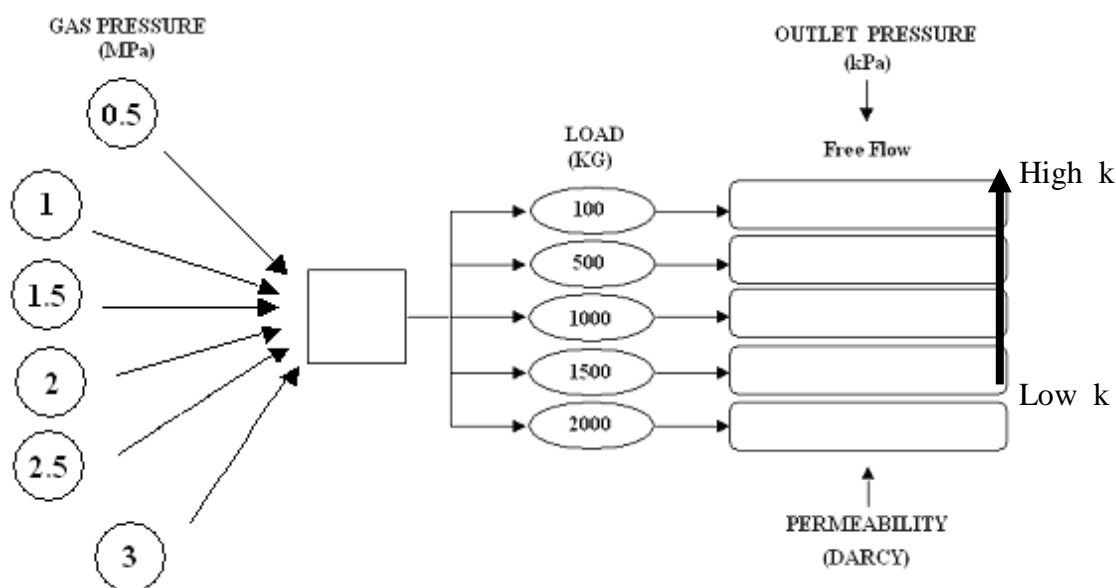


Fig 2- Sequence of changing the pressure and load in the permeability test

The permeability of the sample was calculated using the following Darcy's equation:

$$K = \frac{\mu Q \ln \left(\frac{r_o}{r_i} \right)}{\pi \ell (P_o^2 - P_u^2)}$$

where:

K = permeability (Darcy)
 Q = rate of flow of gas (cc/sec⁻¹)
 r_o = external radius of sample (cm)
 r_i = internal radius of sample (cm)

ℓ = height of the sample (cm)
 P_o = absolute pressure in the chamber (bars)
 P_u = absolute pressure in the outlet (bars)
 μ = viscosity of CH₄ (0.001087)

RESULTS

Figures 3a –3d show the effect of changing the gas pressure (reciprocal) and axial loading conditions on the permeability of coal. Gases used for each test were nitrogen, then methane, carbon dioxide and CO₂/CH₄ (50%, 50%) mixture.

As can be seen, the permeability of coal is highly stress-dependent. Permeability decreased with increasing stress in all gases tested. At the lower mean gas pressures levels the permeability reduction was much more than at higher pressures. It was also observed that the permeability of coal in nitrogen was more than methane (CH₄), carbon dioxide and the CO₂/CH₄ mixture. The lowest permeability was measured for CO₂, which was expected because of the higher adsorption capacity of coal to CO₂. The adsorption of CO₂ takes place mainly in the internal surface of pores and cleats (cracks) of the coal matrix, resulting in lower flow rates. The adsorption capacity of coal for methane was generally lower than carbon dioxide resulting a higher permeability conditions for methane. The permeability of coal to CO₂/CH₄ (50/50 %) mixture was closer to the carbon dioxide than methane. According to Xue and Thomas (1995) varying the ratio of CO₂/CH₄ causes the permeability of coal to change. As it is shown in Figure 3 when the axial load was applied to the coal sample the permeability decreased accordingly as the movement of gas becomes restricted as a result of the applied mechanical load causing the cleats and fractures to tighten or closed (Tarasov, 1960).

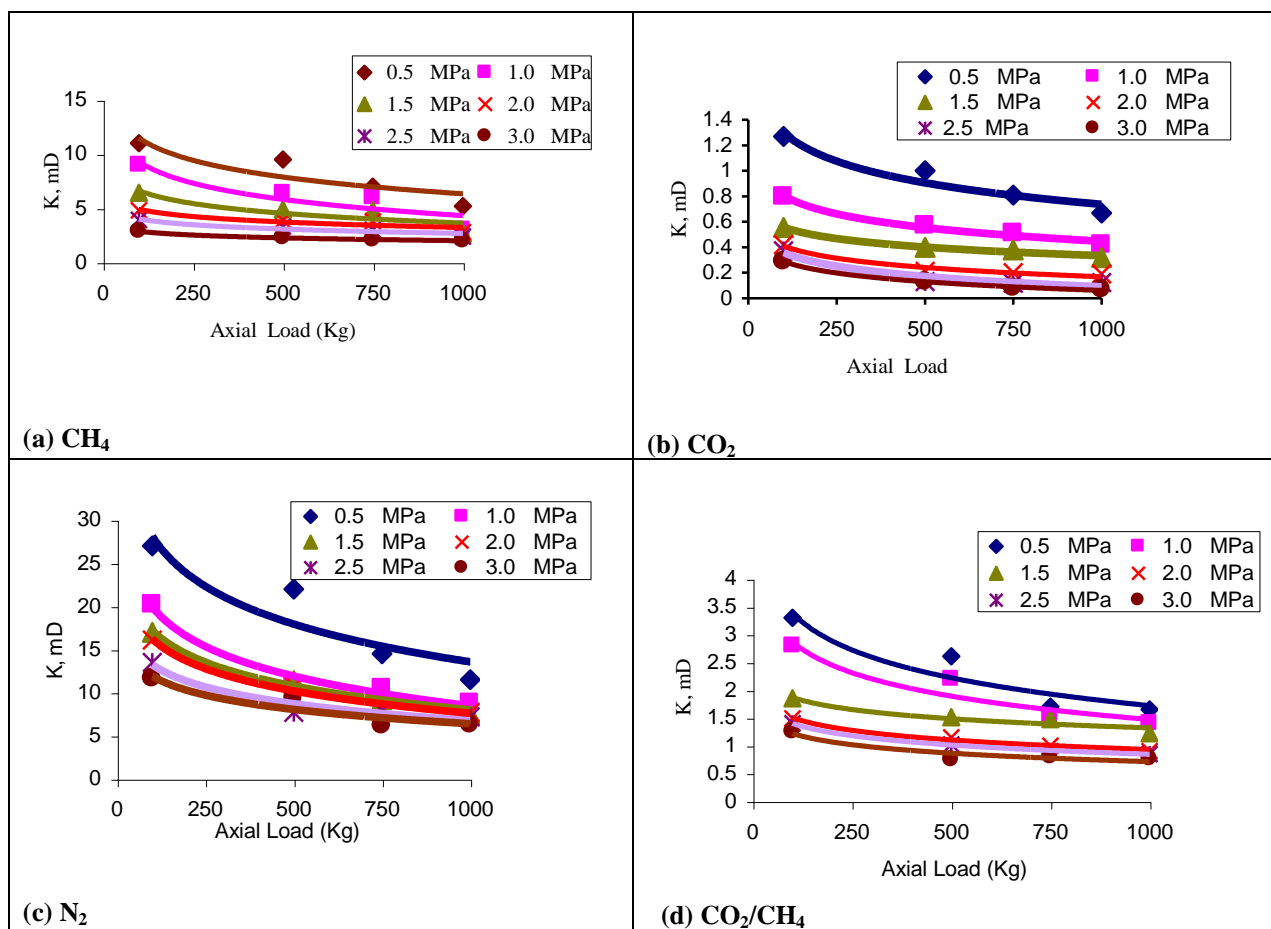


Fig 3 - Effect of stress on permeability of coal in different gases

COAL SHRINKAGE TEST

The volumetric change tests were carried out using a modified pressure bomb of the adsorption/desorption apparatus as described by Lama and Bartosiewicz (1982), and later by Aziz and Ming-Li (1999). A pressure transducer, shown in Figure 4, was mounted on each bomb. Coal samples were sealed in gas bombs and pressurized to saturation level at 3 MPa. The sample containers (bombs) were immersed in a water bath, but were isolated from the water bath by copper sleeves to keep them dry. A thermostatically controlled water bath (with a stirrer) allowed the coal samples to be kept at the desired temperature (25 °C).

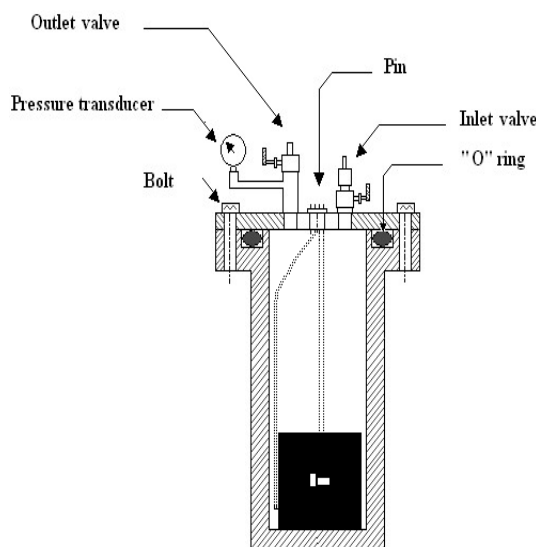


Fig 4 - Sample container (Bomb)

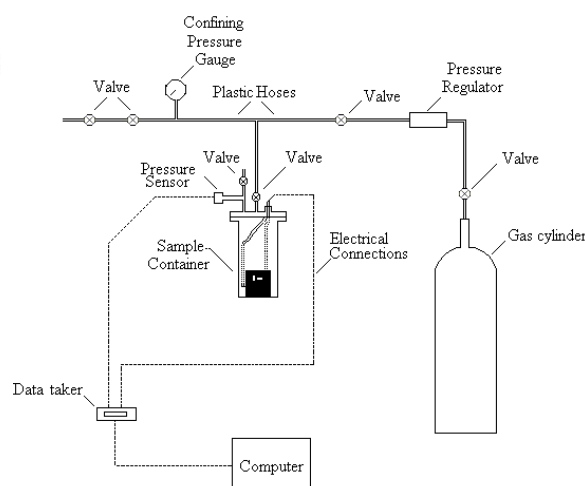


Fig 5 - Schematic diagram of apparatus for testing volumetric change in coal

On the lid of each bomb two types of valves, an isolation and a quick release valve were connected to a gas supply cylinder via a manifold and pressure regulator. To evacuate the gas, a vacuum pump connected, to the manifold, applied suction to the line, expelling any residual gases or air from the system. With this approach, it was possible to bring the pressure to near zero absolute pressure. Pressure release valves enabled the control of pressure and regulated the pressure in each bomb. The whole system capacity was designed to ensure up to 3 MPa absolute pressure and a temperature up to 40 °C. The bomb lid was attached to the body by six bolts with the bomb being sealed perfectly using an 'O' ring in the top of the bomb. Before, the coal samples were placed in the bombs, four strain gauges were mounted on each sample surface to monitor axial and radial strains on coal size due to gas sorption. The mounting of the strain gauges was carried out in accordance with the International Society of Rock Mechanics (ISRM) standard. As shown in Figure 5, data was collected in a datataker (DT500), which later was connected to a PC for recording and analysing. Pressure meters were used to indicate the bombs inlet gas pressures.

Initially, one core sample was placed in each bomb and then vacuumed for the first 24 hours. The bomb was then charged with an appropriate gas type until a maximum pressure of 3 MPa was reached. Once coal was saturated, the gas was then discharged at incremental steps of 0.5 MPa, and the changes in the volume of coal was monitored and recorded in a PC. There were intervals of 2 hours between pressure changes. After finishing one set of tests for a gas type, the bomb was evacuated and the same procedure was repeated for other gases.

Changes in the volume of coal matrix were calculated using the average of the two strains in the axial and radial directions. The shrinkage coefficient (C_m), is defined as the rate of change of coal matrix volume to the change in gas pressure and is given by (Harpalani and Chen, 1997):

$$C_m = \frac{1}{V_m} \left(\frac{dV_m}{dP} \right)$$

where:

V_m = Matrix volume	(m ³)
dV_m = Change in volume	(m ³)
dP = Change in applied pressure	(MPa)
C_m = Shrinkage coefficient	(MPa ⁻¹)

The influence of an incremental reduction of gas pressures on a pressurized coal sample of 3 MPa, for Burton coals are shown in Figure 6. The trend in incremental increase in coal column as a result of gas pressure drop is shown. As in pressurization, the level of volume change is greatest in a carbon dioxide environment, followed by the 50% mixture CO₂/CH₄, then CH₄ and then N₂ gas.

RESULTS

Figure 6 shows the relationship between applied gas pressure and volumetric change in coal. The coal sample was initially charged to a maximum pressure of 3 MPa. The changes in coal volume were monitored in increments of 0.5 MPa. As can be seen, the reduction in coal volume is different for different gas medium. A minimal change in coal volume was measured with nitrogen while a CO_2 environment produced the highest volume change. Obviously, the influence of CO_2 reflects an strong affinity of the gas for coal. As coal adsorbs CO_2 more strongly than methane, it is thus likely the high rate of gas storage in coal is accommodated with the increase in coal volume. Clearly the change in coal volume can be more than five fold in CO_2 in comparison with the methane environment. The relative change in coal volume in mixed CO_2/CH_4 environment is between pure CH_4 and CO_2 , but the mixture proportions influenced the degree of volume change.

CONCLUSIONS

The experimental study reported in this paper has demonstrated that increasing stress tends to close the cleats and reduce permeability within the coal. Also, the degree of influence is dependent on gas type and pressure. Permeability of coal was found to be highest in nitrogen and lowest in CO_2 gas. Coal samples have been shown to expand on gas sorption and shrink during gas desorption. The level of coal shrinkage was affected by gas type and pressure. Carbon dioxide gas appears to cause the highest volume change and nitrogen has the least effect. This is understandable in view of the fact that coal has higher affinity for carbon dioxide gas than other gases tested. Obviously, the changes to coal permeability and volume are likely to be different for different coals and this issue is currently the subject of an ongoing research by the authors.

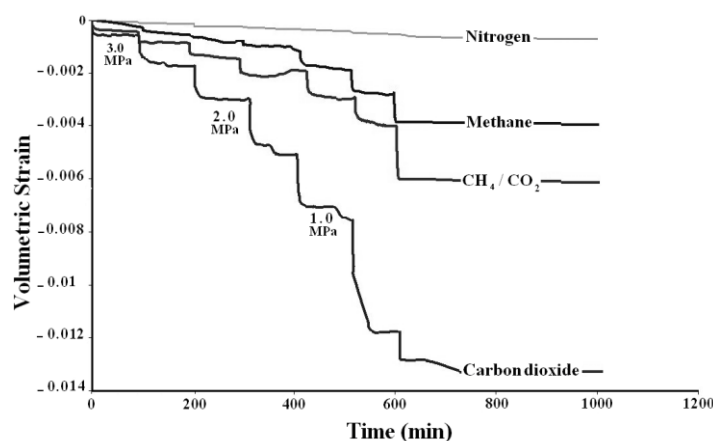


Fig 6 - Volumetric strain for different gases and pressure reductions at increments of 0.5 MPa

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INVESTIGATION INTO THE EXTENT AND MECHANISMS OF GLOVING AND UN-MIXED RESIN IN FULLY ENCAPSULATED ROOF BOLTS.

Richard Campbell ¹, Richard Mould ², and Stuart MacGregor ³

ABSTRACT: Effective strata control, utilising fully encapsulated roof bolts is dependent on the installed quality of the reinforcement elements. One mechanism by which roof bolts may become less than fully efficient is by glove fingering (gloving) and un-mixing of the resin. Following a routine installed bolt quality audit and some small roof failures containing gloved bolts, a work programme was initiated to determine the extent of the gloving and un-mixing problem and to develop an understanding of mechanisms involved. Results have shown that gloving and un-mixing is a systematic and widespread phenomena, occurring across the range of resin and/or bolt manufacturers, and in a variety of roof types. Gloving was found in bolts installed using either hand held pneumatic or continuous miner mounted hydraulic bolting rigs, under run of mine (ROM) conditions by operators, and under controlled manufactures "best practice" conditions.

The mechanisms involved have been confirmed as being the development of a pressure front as the bolt encounters the resin cartridge and is spun up the hole, which in turn, leads to over-pressurisation and radial expansion of the resin cartridge. The result is an increase in the diameter of the plastic cartridge. Allowing the bolt to be spun up inside the cartridge without making sufficient contact to shred the cartridge or the hardener envelope, typically resulting in a portion of the cartridge enveloping the bolt and unmixed resin mastic and catalyst.

Once the mechanisms involved and extent of the problem became clear, further research was undertaken to assess alternative bolt profiles and modifications in an effort to minimise and/or eliminate the gloving and un-mixing phenomenon. Research has been undertaken using recovered bolts from various mine sites, as well as test bench trials and the quantification of the loading characteristics of gloved bolts using strain gauge roof bolts.

To understand the impacts of gloved and un-mixed bolts on roof control, failure pathways and reinforcement requirements a FLAC 2D numerical simulation was undertaken, with the results being incorporated into the strata management plan for a particular operation. Laboratory data has been collected and analysed to assess magnitudes of resin pressure as the bolt encounters the cartridge¹ and the effects of gloving and un-mixing on the load transfer characteristics of the resin bolt system.

INTRODUCTION

Gloving, in this context, refers to the plastic cartridge of a resin capsule partially, or completely, encasing a length of bolt, typically with a combination of mixed and unmixed resin filler and catalyst remaining within the cartridge. The gloved and unmixed portion reduces the effective anchor length and adversely affects the ability to reinforce the roof strata. Figures 1, 2a and 2b illustrate typical examples of gloved roof bolts and un-mixed resin/catalyst.

¹ SCT Australia Operations Pty Ltd

² Solid Energy NZ Ltd

³ SCT Australia Operations Pty Ltd

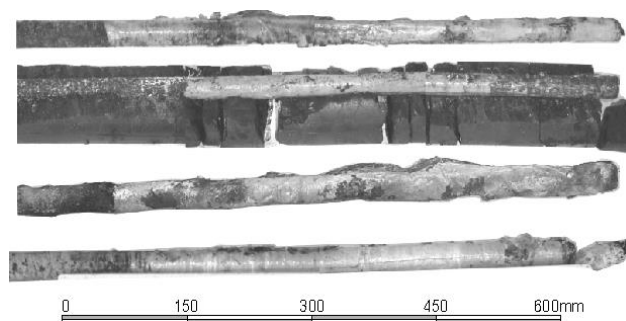


Fig 1 - Typical appearance of gloved and un-mixed bolts.

Gloving has been recognised in fully encapsulated roof bolts for many years (Pettibone, 1987) and has traditionally been attributed to poor installation methods, issues relating to drilling/installation equipment, poor handling and/or storage of consumables or geological issues resulting in abnormal ground conditions. Very little quantitative data is available in the public domain, although recent work by Campoli *et.al.* (2002) and Campoli, Mills, and Adams (2003), Fiscor (2002), Campbell and Mould (2003), and Pasters (2003), along with hazard alerts from the Queensland Mines Inspectorate (Qld Govt 2002) have brought some focus to the problem. Fiscor (2002) refers to gloving being identified as the cause of roof falls in American operations; in one of the few published references directly linking gloved bolts and falls of ground.



Fig 2(a) - Upper photo shows intact core in stone roof.

Lower photo illustrates how once the core is removed the gloving and un-mixing is revealed, in this case 550mm was found to be gloved and un-mixed.
 Note: Bolts are photographed with the top of the bolt on the left hand side and the nut and plate on the right hand side.

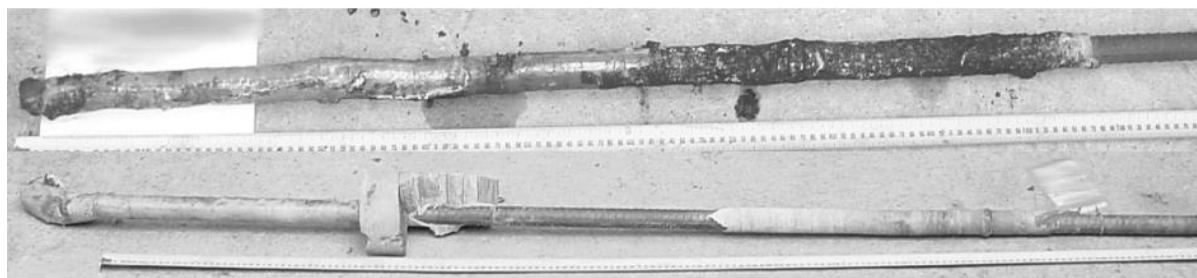


Fig 2(b) - Typical examples of gloved and un-mixed bolts.

Note: Bolts are photographed with the top of the bolt on the left hand side.

LOAD GENERATION AND TRANSFER IN RESIN GROUTED ROOF BOLTS

The performance of any reinforcement design is limited by the efficiency of load transfer of the reinforcing members. Load transfer is the mechanism by which force is generated and sustained in the roof bolt as a consequence of strata deformation (Fabjanczyk and Tarrant 1992).

In a fully grouted roof bolt the load transfer mechanism is dependant on the shear stress sustained on the bolt-resin and resin-rock interfaces. The peak shear stress capability of the interfaces and the rate of shear stress generation, determines the response of the bolts to the strata behaviour. This concept is illustrated in Figure 3.

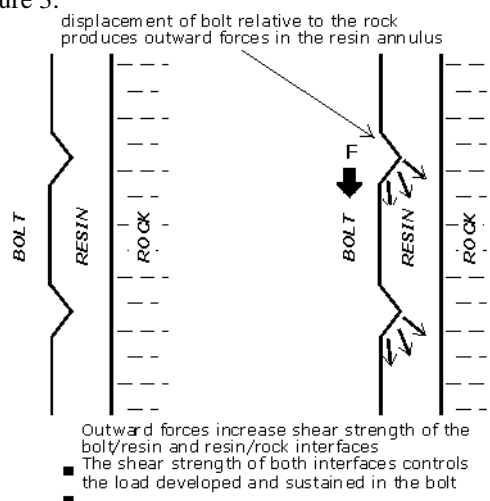


Fig 3 - Mechanism of load transfer (after Fabjanczyk and Tarrant 1992).

OVERVIEW OF INVESTIGATION PROGRAM

The work program reported here was undertaken over the past 2 years as part of an audit of the installed quality and effectiveness of roof bolts in various coal mining operations. The initial investigation program concentrated on defining the extent of the problem and developing an understanding of the mechanisms involved, this was then followed by a series of trials aimed at finding an acceptable solution. The latter involved trials of alternative bolt/resin suppliers and modifications made to bolts based on the understanding of the mechanisms involved.

The over coring method allowed the recovery of bolts installed ROM by operators at the face and in the back-bye districts of the mines, as well as to audit bolt installations under controlled manufacturers "best practice" conditions using either pneumatic (gopher/wombat) or hydraulic (miner mounted) drilling rigs. At each operation standard drilling consumables were used, and a strict measurement and recording protocol was defined to ensure the same parameters were recorded so that all results were comparable. Over coring was undertaken using specialist techniques and equipment, drill bits and barrels developed by SCT Operations, using either a Proram or Ramtrak drill rig. Stone roofs were typically drilled with a diamond bit and coal roofs with a tungsten-tipped bit. In addition to the over coring program, bolts of various type, age and manufacturer have been collected from falls of ground and goaf edges for comparison and to assist in gauging the extent and history of the issue. Further quantitative data has also been gathered using strain gauge bolts, installed to measure the loading characteristics of the gloved bolts.

EXTENT OF THE GLOVING PROBLEM

In excess of 80, 1800mm long roof bolts of four different types/manufacturers were over cored and recovered to date. The bolts have been recovered from five separate operations, in different geological and geotechnical settings, with the immediate bolted section comprising a range of lithologies from thick coals to more typical shale/mudstones and sandstone/laminites. The over cored bolts investigated had been installed for time spans, ranging from hours to up to 18 months.

Bolts and resins used in this program are from various manufacturers and suppliers in Australasia. The resin cartridges used were all two-part polyester (fast/slow setting times), 900-1000mm long and nominally 24mm in diameter. The bolts used were all flat topped, with a core thickness of 22mm and 1 to 2mm ridge profiles, with ridge spacing depending on manufacturer and bolt type.

The investigation program was undertaken in two phases, the first being the recovery of ROM installed bolts from throughout the five mine sites to assess if gloving was routinely occurring. The second phase was to undertake controlled "best practice" installations (often with a manufacturers representative present) to determine if an acceptable quality bolt (nil gloving and un-mixing) was achievable. Both data sets are presented here. The results of the investigation work are summarised in Figure 4 in terms of bolt and resin type/manufacturer, roof lithology and installation type, ie controlled or ROM.

As illustrated in Figure 4, regardless of bolt type, roof lithology or if the bolts were installed by miners ROM or under best practice controlled conditions, an average of 500mm of bolt length is typically affected by gloving, with 200mm typically gloved and mixed and 300mm typically gloved and un-mixed. Although it must be noted that there is a wide range in values in the data set, (from 30mm to 790mm), with the majority of the data set (70%) having in excess of 400mm un-mixed length.

The over coring program graphically illustrated that gloved and un-mixed bolts were common at all sites investigated regardless of roof lithology and occurs across the range of bolt and/or resin manufacturers. The results also indicate that there is little difference in ROM bolts and those installed under "best practice" controlled conditions. In fact, it is apparent that an acceptable quality bolt is typically not achievable using the current flat-topped bolt and cartridge systems on the market. As such it can be concluded that gloving and un-mixing is not the result of poor installation methods, issues relating to drilling/installation equipment, poor handling and/or storage of consumables or geological issues resulting in abnormal ground conditions.

In addition to the over cored bolts several areas of falls in back-bye regions and goaf edges of mines were inspected in an attempt to find bolts of differing ages and resin cartridge configuration to determine the time span over which gloving has been a problem. To date bolts ranging in age from hours to in-excess of 12 years have been recovered, all of which show a similar degree of gloving and/or un-mixing, indicating that it is an issue that has been prevalent in the industry for a considerable time.

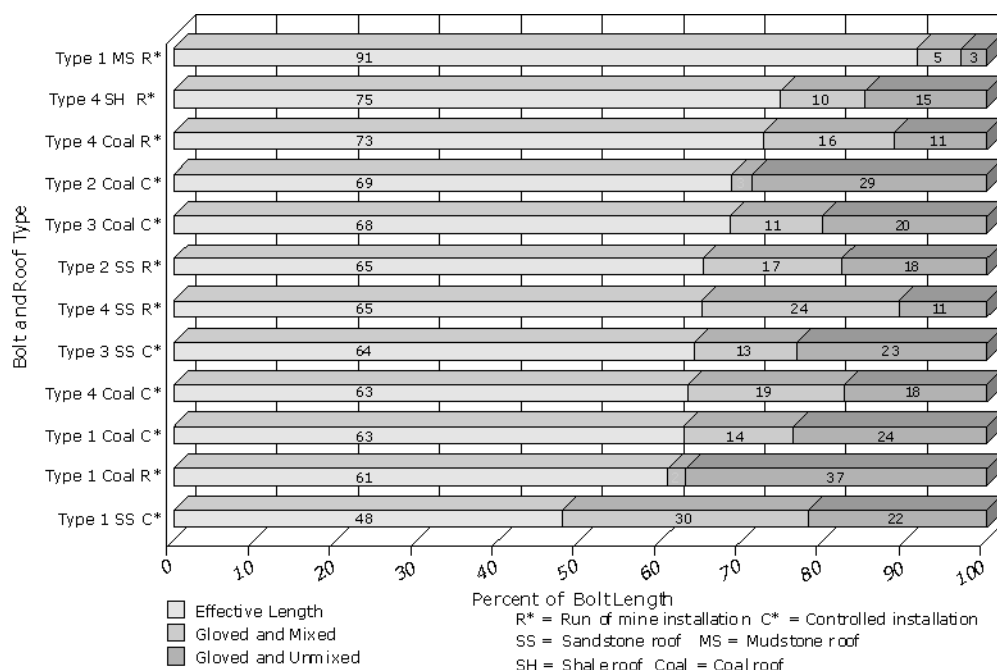


Fig 4 - Gloving investigation summary of results for standard bolts.

GLOVING MECHANISM

Determining the mechanism for gloving and the resultant un-mixing has been the focus of much of the preliminary investigation work. From observations of bolt installation, over cored bolts and from test bench trials and measurements, an understanding of the mechanisms has been developed and confirmed. This is discussed below.

A pressure front develops when the standard flat-topped bolt encounters the resin cartridge as it is spun and pushed up the hole. The bolt acts essentially as a piston hydraulically pressurising the resin and cartridge, which shortens and in turn undergoes radial expansion until it is confined by the sides and the back of the drill hole.

The result being an increase in the cartridge diameter from 24mm to the diameter of the drill hole, nominally 27 to 29 mm. In conjunction with the expansion against the drill hole wall, the catalyst envelope/tube also becomes flattened against the side of the drill hole.

The increased cartridge diameter allows the bolt (nominally 22mm core and 24mm rib diameter) to be spun up inside the cartridge without making sufficient contact to shred the cartridge or the flattened hardener tube. Culminating in a portion of bolt encased in intact cartridge and typically a combination of mixed and unmixed resin mastic and catalyst.

To confirm the causes of gloving, test bench simulations were undertaken in clear 28.5mm internal diameter polycarbonate tubing so that direct observation of the gloving mechanics could be made as the bolt was installed. Visual conformation of the pressure front development and of the bolt entering the cartridge (gloving) was possible.

The observations showed that the pressurisation of the cartridge reaches sufficient levels to expand the cartridge against the wall of the tube at 600 - 700mm from the end of the 1800mm tube, and it is at this point the bolt enters the resin cartridge. Once the bolt was spun up inside the expanded cartridge no further shredding of the cartridge took place, resulting in a 600-700mm gloved and un-mixed length of bolt, replicating the field observations, and confirming the mechanisms involved.

UP-HOLE DYNAMIC PRESSURE

Up hole dynamic resin pressures have previously been measured (Mills, 1999) using a pressure transducer through a hole in the top cap of a 27-27.5mm polycarbonate tube under test bench conditions designed to closely simulate field installation of roof bolts. The dynamic pressure was shown to rise to its peak of around 4MPa very rapidly as the bolt was spun up the hole with the maximum pressure developed after 4 seconds, which equates to the bolt being approximately 1100 to 1200mm up the hole. Figure 5 illustrates the results generated.

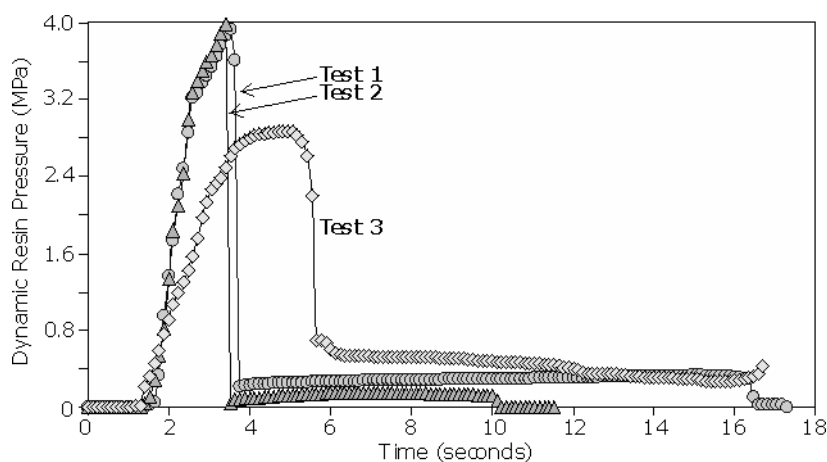


Fig 5 - Measured up-hole dynamic resin pressure as bolts

It is worth noting that the peak values recorded may not be the actual maximum pressure due to problems maintaining the transducer fittings, as the pressures were such that the end-cap detached and in some cases the polycarbonate tube split.

Up-hole resin pressures in excess of 4MPa are considered to be sufficient in magnitude to induce hydraulic-fracturing of strata and joints in situations where minor horizontal stress is of the same order of magnitude. Hydraulic fracturing causing the initiation or opening of joints or cleat, results in the injection of resin into the strata and can lead to significant loss of encapsulation length. Indeed resin injection into strata was commonly observed in over-cored bolts along with considerable loss of encapsulation. Typically resin could be observed in the drill core to radiate out from the bolt-hole to in excess of the diameter of the recovered core ($\text{Ø}100\text{mm}$) and be up to 2-3mm thick. Examples of resin injection as a result of hydro fracturing are illustrated in Figures 6a, 6b and 6c. The volume of resin loss is sufficient to significantly reduce the encapsulation length, which was found in extreme cases to be less than the length of the installed resin cartridge.

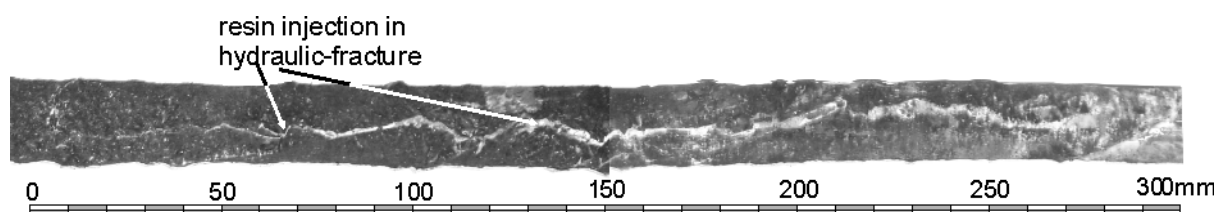


Fig 6(a) Example of resin injection into an induced fracture.

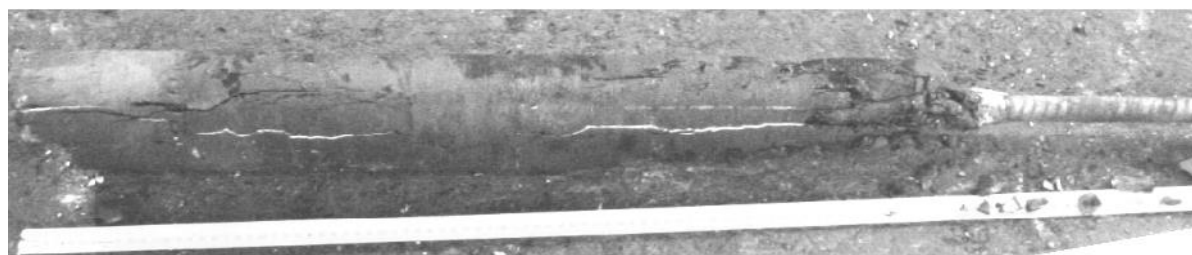


Fig 6(b) - Resin injection into an induced fracture in an all coal roof.

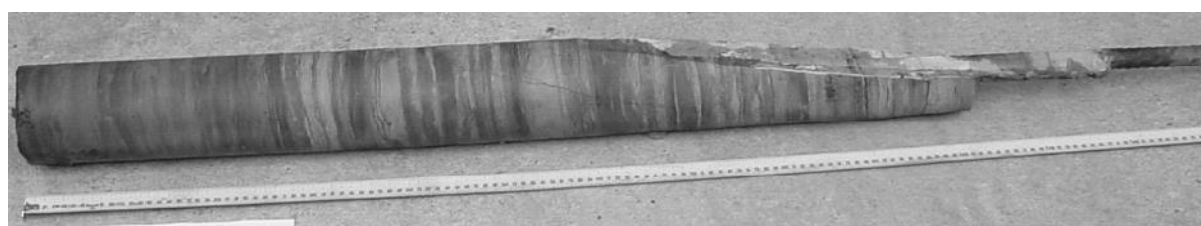




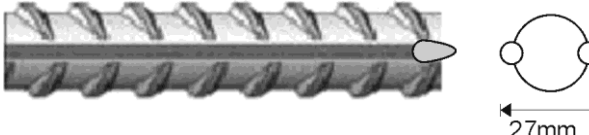

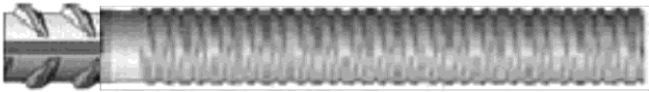
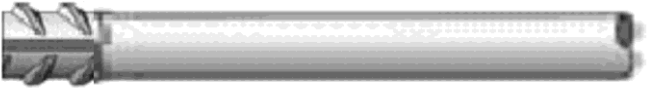

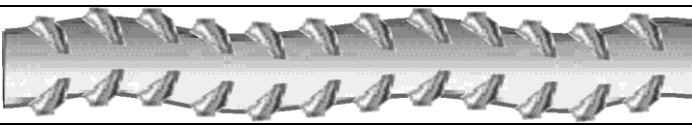
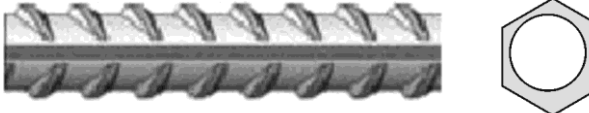
Fig 6(c) - Resin injection along a joint in interbedded sandstone and mudstone roof.

TRIALS OF ALTERNATIVE BOLT PROFILES AND TIP CONFIGURATIONS

Following the recognition of the extent of the problem and the understanding of the mechanisms involved, a considerable effort was made to develop a solution, using bolt tip modifications suggested by a range of sources (operators to consultants) and alternative profiles available on the market as recommended by various bolt manufactures.

Trials were undertaken across several of the mine sites to cover the same range of geotechnical conditions. In total a trial of 9 alternative profiles and tip modifications was undertaken, with details of bolts given below in Table 1. Some 95 bolts were recovered across the spectrum of modifications and mine sites. At all trial sites standard bolts were also installed and recovered as a control to ensure that the modifications could be directly compared to standard profile bolt types.

Table 1 - Details of Modified and Trial Bolt Profiles

Modification Type / Bolt Name	Description	Schematic Diagram
chamfered	Curved wedge of bolt removed from tip	
Welded	Bar of weld built up on opposite sides of bolt	
Horn	Two 'horns' built-up onto tip and sides of bolt	
Paddle	Tip flattened to form a paddle shape	
Peeled Threaded and	Ribs peeled off 160mm of bolt, and thread rolled on	
Peeled	Ribs peeled off 160mm of bolt	
Spiralled Wire	Spring wire attached to tip and wound along bolt	
200mm wiggled	Standard bolt with wiggle profile over upper 200mm of bolt	
Off-centre	Nut made off centre relative to the bolts long axis	

The results of the trial are summarised in Figure 7, which illustrates the average proportions of bolt length that were affected by gloving and un-mixing. Initial trials (June 2002) were carried out using the chamfered, horn, welded, paddle, peeled and peeled and threaded bolts. The spiralled wire, 200mm wiggled and off-centre type of profiles were only recently trialed (May 2003).

It can be clearly seen that of the modifications tested only the chamfered, wiggled and off-centre bolts offered any significant improvement, with each achieving in-excess of 90% effective bolt length. The remaining modifications showed only minor or very inconsistent improvements in both mixing and shredding of the cartridge. The result of the modification and alternative bolt profile trial indicates that significant improvements

in resin mixing and shredding of the cartridge was consistently achieved using the chamfered, wiggled and off-centre types of bolts.

The results of the initial trials of the chamfered bolt were considered so successful at the time that this modification was adopted as the standard bolt profile in a large scale trial at one of the mine sites involved. More recently the 200mm wiggled profile bolt has been introduced as the standard roof bolt at that mine.

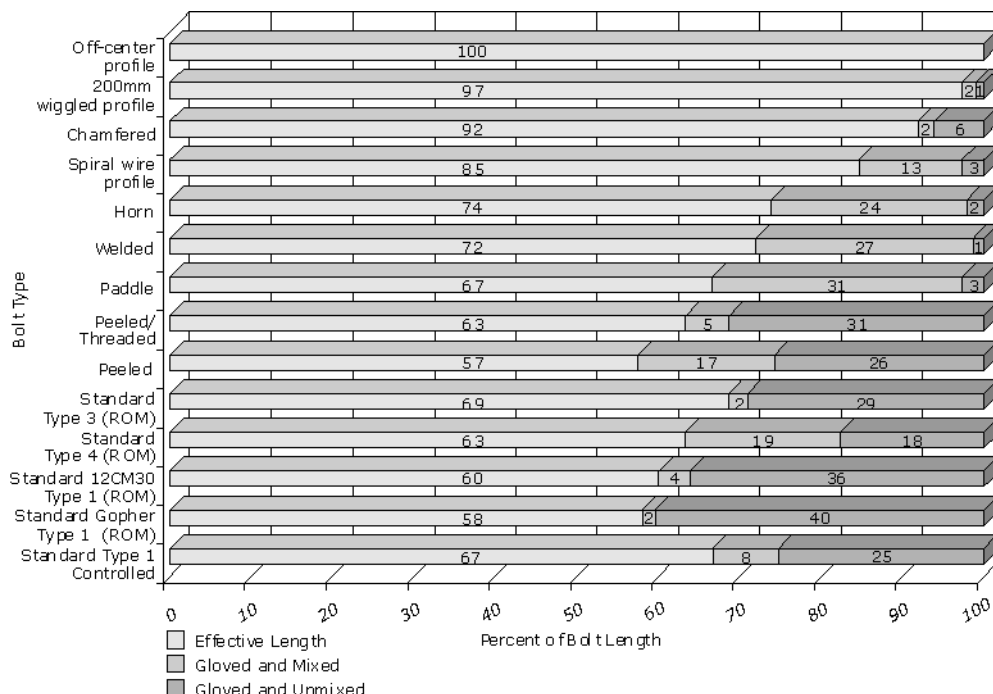


Fig 7 - Summary graph of average modified bolt performance.

LOADING CHARACTERISTICS OF GLOVED AND UN-MIXED RESIN

The use of instrumented roof bolts to quantify the loading characteristics of a gloved roof bolt was investigated. The results presented in Figure 8 show that the bolt (which was installed ROM by operators) was indeed gloved over the upper 400mm, as indicated by nil load transfer over that portion of the bolt length. The resultant loading profile illustrated that the bolt was able to generate significant load over the shorter length and was not approaching yield.

The use of a modified instrumented bolt as part of a ROM installed quality audit is currently being explored, as the results shown in Figure 8 illustrate that they are able to identify the extent of gloving and may be a cost effective, non-destructive alternative to over coring as a means of assessing if gloving and un-mixing is occurring in any operation.

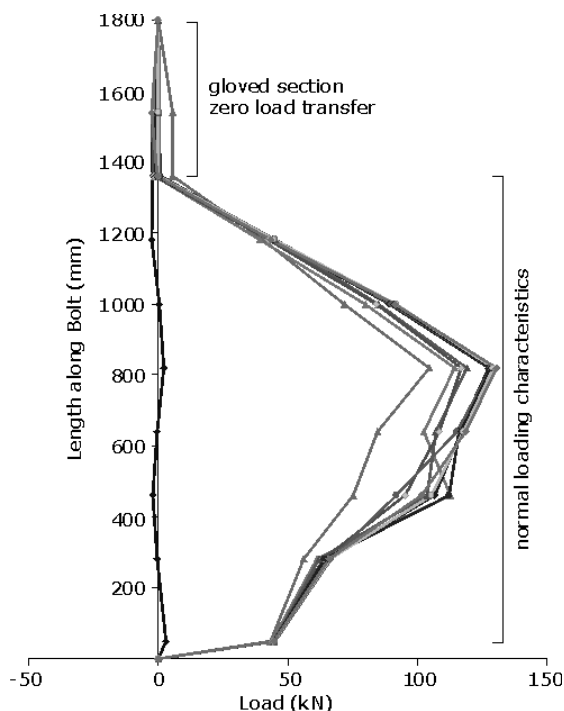


Fig 8 - Results of an instrumented roof bolt showing nil load transfer over gloved section.

LABORATORY PULL TESTS TO QUANTIFY THE EFFECTS OF GLOVED AND MIXED RESIN

Gloving of the bolt by the cartridge, where the resin has hardened, may adversely affect the load transfer characteristics of the bolt system by reducing the system stiffness and by providing a low friction interface reducing the magnitude of the shear stress sustainable on the bolt/resin/rock interfaces.

Short encapsulation pull tests undertaken by the United States Bureau of Mines (USBM) on gloved bolts indicated that there was no detrimental affect to the load transfer characteristics caused by mixed and hardened resin encased in the plastic resin cartridge (Pettibone, 1987). Pettibone also reported that the cartridge did not provide a surface, which would promote shear under heavy loads. The detrimental effect of gloving on short encapsulation pull tests has been discussed (Mazzoni et.al., 1996; Mark et.al., 2002).

In order to attempt to quantify the impact of gloved and mixed resin on load transfer characteristics laboratory short encapsulation pull tests (180mm) were undertaken at the University of New South Wales School of Mining using the rock bolt testing facility. Inspection and analysis of short encapsulation pull test specimens was undertaken to quantify relative proportions of gloving/unmixing and gloving/mixing. All of the test specimens exhibited some degree of gloving and un-mixing, with only 4 specimens having significant (greater than 10mm) gloving and mixing. The peak shear strengths (normalised for the actual effective resin length) were then correlated to the proportions of gloving and mixing.

The results, while not considered to be statistically representative, do indicate that the load transfer of the bolt/resin system is not significantly reduced by a gloved and mixed section of bolt, with the data scatter being within the same range of results as that of the shredded and mixed bolts.

IMPLICATIONS FOR DESIGN

The wide spread nature of the gloving problem led to the realisation that the current roof support design assumptions were compromised along with the subsequent deformation trigger levels which form a major component of most strata management plans. The decision was made to case study a mine site where a calibrated numerical (FLAC 2D) model existed and a significant amount of field observation and roof deformation monitoring data was available to assess the impacts of the reduction in effective bolt length on the strata behaviour and stability of an opening.

The FLAC model was used to simulate two different support configurations using 1800 mm bolts. In the first case it was assumed that the bolts were fully encapsulated, and 100% effective. In the second the top 600mm of the bolt was gloved and encased in un-mixed resin, effectively making the bolts 1200mm long. The length of gloving and un-mixing was consistent with the recovered bolts for that operation.

All other parameters were kept constant in the models, these being the excavation size/geometry, the geology and geotechnical setting (depth of 200m, 4m coal roof) and load transfer characteristics of the effective bolt lengths.

A range of horizontal stress magnitudes (5 – 10MPa total stress, based on previous 3D over core stress measurements) were simulated to compare against the performance of 1.2m and 1.8m long bolts in the same environment. In this way the general stability curve (with respect to oversteering) for each bolt length is established.

Figure 9 details stability curves for the 1.2m and 1.8m long bolts showing total roof displacement versus horizontal stress. The modelling indicates that the gloved and un-mixed 1200mm bolts become overloaded earlier (at a lower stress level) than the 1800mm long, 100% effective bolts.

In the case of the 1200mm bolts, once over stressing and shear initiates in the roof, the failure path is rapid, with softening above the bolts and mobilisation of the contact at the top of the seam at lower displacement levels. For example softening above the bolts occurs at a displacement of 10mm for 1200mm bolts, compared with 15mm for 1800mm long bolts.

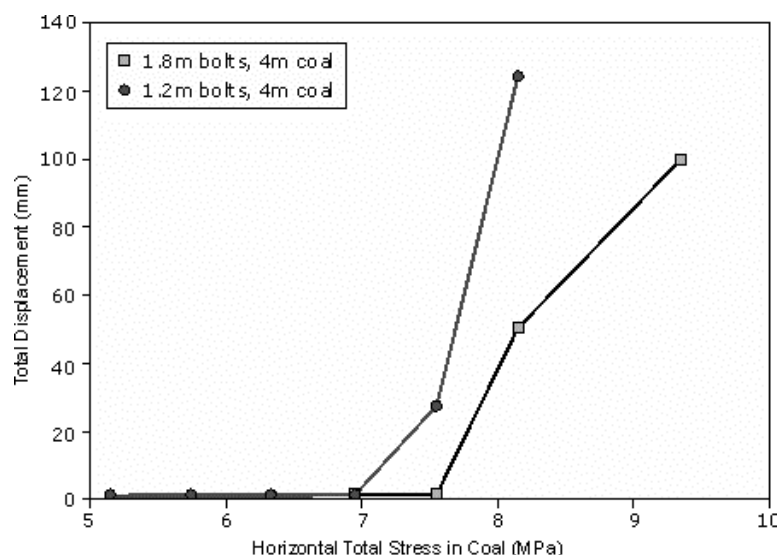


Fig 9 - Roof stability pathway with respect to applied horizontal stress.

Figures 10a and 10b detail FLAC model outputs of strata softening occurring for the same horizontal stress level. The magnitude of the horizontal stress modelled is 7.5MPa (total stress) in both situations, which was considered representative of the mining conditions at the time, based on observations of guttering and roof deformation levels. It can be seen that the longer bolts are better able to confine the immediate roof and act as a pattern to cope with a wider range of conditions while the gloved bolts act as isolated reinforcing members.

The findings of the modelling, and revised trigger levels have since been incorporated into an updated Strata Management Plan for the affected areas. Simultaneously a review and re-analysis of the existing roof deformation monitoring data (from wire extensometers) was undertaken to assess which areas of the mine were potentially requiring remedial secondary support at the revised lower magnitudes of deformation.

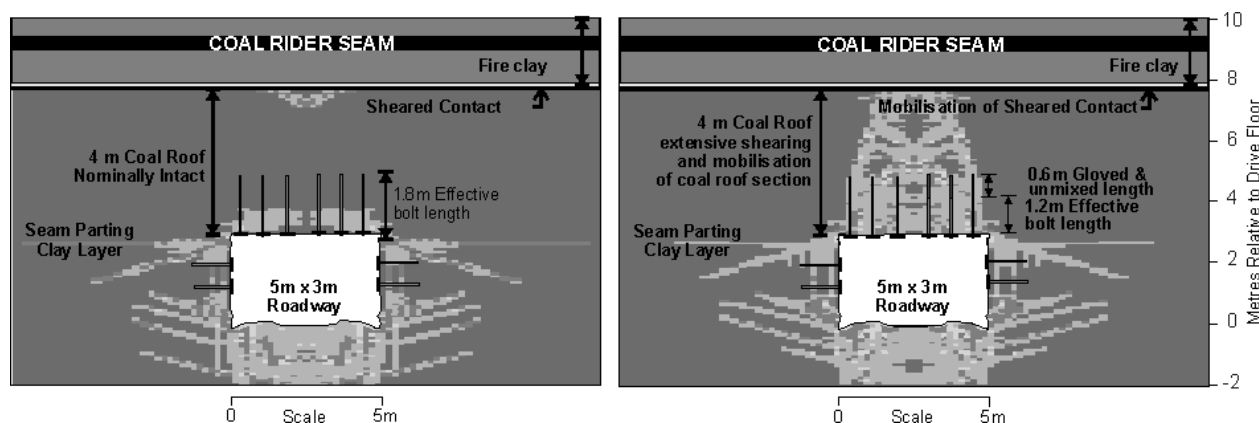


Fig 10(a)- Results of the FLAC modelling of 1800mm long 100% effective bolts.

Fig 10(b) - Results of the FLAC modelling of 1200mm effective length, gloved roof bolts.

SUMMARY AND CONCLUSIONS

Investigation work has identified gloving and un-mixing of resin-grouted roof bolts as a significant problem, which occurs across a range of geological/geotechnical settings, and across the range of bolt/resin manufacturers.

Gloving was found over a wide range of bolt lengths, with the results showing anywhere between 30mm to 790mm affected. Typically the gloving affected around 500mm of the up-hole end of the bolt. The length of bolt affected by un-mixing also varied, with up to 750mm of bolt encased in un-mixed resin.

While the affects of gloved but mixed resin may be minimal, the un-mixed portion affords no reinforcement to the roof. The impacts of this can be assessed in two fronts, the first being with respect to health and safety, and the second being the economic cost. In terms of primary reinforcement dollars, this equates to 10% to 30% of the reinforcement dollars being of no benefit.

A mechanism, which accounts for the gloving and un-mixing phenomenon has been described, and validated by field trials and test bench trials and measurements.

The over pressurisation of the resin column as the bolt is spun up the hole results in the radial expansion of the cartridge and flattening of the hardener tube against the borehole wall. The bolt enters the expanded cartridge and does not shred the hardener tube, often resulting in a gloved and/or un-mixed section of bolt.

Nine bolt modifications and alternatives were tested across the range of geotechnical conditions, with the best results being achieved by the off-centre, Chamfer and a 200mm wiggled bolt, with the latter two being introduced as the standard bolt profile at one operation.

Numerical modelling was used to assess the impacts of gloving and un-mixing and showed that the shorter effective bolt length does have a significant affect on the design assumptions and stability of mine openings. In the case presented, the shorter bolts could not interact as a pattern and the reinforcement afforded to the roof was reduced, compared to the design assumption of 1800mm long bolts. The result being: an increased height of softening at lower levels of deformation, leading to the isolation and over stressing of the immediate and secondary roof sections. Following the results from the modelling a review of the Strata Management Plan was required to incorporate the findings, and a review of all monitoring data was required to reassess the requirements for secondary support.

Strain gauge bolts have been used to assess the loading characteristics of the gloved and un-mixed bolts, and may provide a means of assessing gloving, ROM, on a regular basis as part of an audit process.

ACKNOWLEDGEMENTS

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AN UPDATE OF ROOF BOLT RESEARCH AT THE UNIVERSITY OF WOLLONGONG

Naj Aziz ¹

ABSTRACT: The influence of surface profile on load transfer mechanisms of bolts has been studied under both the constant normal stiffness (CNS) and constant normal load (CNL) conditions. Testing under CNS condition was conducted in a specially constructed constant normal stiffness shearing apparatus, whereby the flattened surface of a bolt section was pulled against the image of cast resin sample under constant stiffness conditions. Testing under CNL conditions included the conventional pull testing of an encapsulated section of bolt anchored in a borehole, and the short encapsulation push test.

The conventional pull testing involved pull testing of three different profiled bolts in three different diameter holes. The pull tests were carried out both insitu and in the laboratory. Parameters examined, in addition to bolt surface profile, were the resin annulus thickness and the effectiveness of resin mixing in the hole.

The credibility of push testing, in short steel cylinder sleeves, was examined by pulling the bolt out of the cylindrical sleeve instead of pushing. Also tested in short sleeve, was the possibility of changing the load transfer capability of a bolt by changing its surface profile.

A numerical simulation study has recently been incorporated to enhance the current programme of research carried out at the School of Civil, Mining and Environmental Engineering, The University of Wollongong. The numerical study included the modeling of both the short encapsulation push/ pull test as well as shear stress simulation across joints. The conclusions drawn were that the bolt surface profile is an important parameter affecting the load transfer capacity of the bolt/rock interface, that the anchorage strength of resin encapsulation is influenced by the effectiveness of resin mixing, and the annulus thickness, and the short encapsulation push test underestimates the peak shear load and peak load displacement.

INTRODUCTION

The following programme of research is currently ongoing at the School of Civil, Mining and Environmental Engineering, University of Wollongong:

- Examination of bolt resin shear failures under CNS conditions;
- Load transfer mechanism studies by conventional pull tests, to include pull out testing of bolt with encapsulation length up to 300 mm. This type of study is carried out both in the laboratory and in the field. Different bolt surface profiles were examined;
- Push /pull testing of the bolt encapsulated in a short steel sleeve;
- Double shear testing of bolts, with 1.20 m long bolts installed in a three-piece concrete block, subjected to shearing load;
- Modeling of bolt shear failure in both short encapsulation and double shearing tests; and
- Bolt corrosion with respect to stress corrosion cracking. A purpose-designed rig, that allows tests to be conducted on bolts under both tension and torsion conditions.

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The latest of the research findings are presented with different methods listed above. The study on shearing under CSN conditions reported by Aziz, (2002), Aziz and Dey and Indraratna (1999), was further extended to include testing under triaxial conditions. However, the study on short Encapsulation test, reported, by Aziz and Webb (2003a) is extended to include tests by pulling of the bolt out of steel cylinder. Additional studies include bolt technology appraisal by numerical modeling, laboratory, and fieldwork and are the subject of this presentation.

The modeling of the double shearing tests, though reported here, is dealt in a separate paper in this proceedings by Jalalifar, Aziz and Hadi (2004).

CONVENTIONAL SHORT ENCAPSULATION PULL TEST

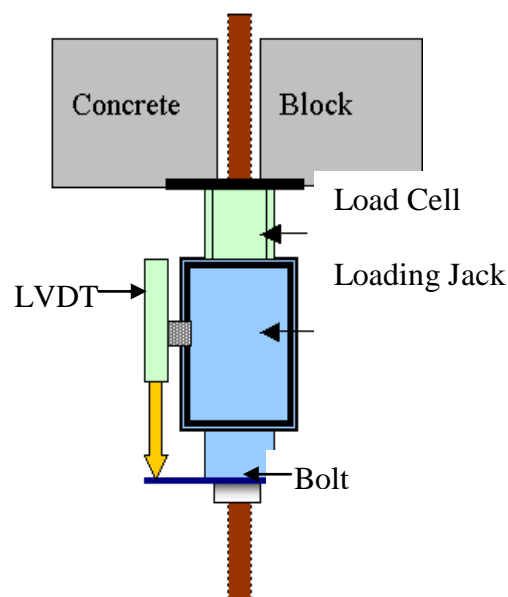
Laboratory Test

The laboratory experimental work was carried out in a purpose built testing rig facility pictured in Figure 1a. The rig consisted of a double deck steel frame structure. The upper deck carried a drilling medium of a block of rock or concrete and an overhead-lifting crane (not shown in the figure) used for lifting and placement of the drilled medium. A hydraulic drilling rig, positioned beneath the drilled concrete block, was adapted from a continuous miner.

The high strength concrete block had a 1.0m^2 base area that tapered to 0.9m^2 area at the top, and an overall height of 1.2m. Figure 1b shows the general arrangement for pull testing of the bolts. The hydraulic ram had a maximum capability of 30 tonnes and was powered with a two-stage 'Rodgers' hydraulic pump. The load applied to the bolt was measured using a hollow load cell. A Linear Variable Differential Transducer (LVDT) was used to measure the bolt axial displacement during the pulling process.



a) Drill Rig






b) Bolt Pulling System

Fig 1 – Laboratory drill rig and bolt pull test arrangement

The process of bolt installation in the concrete block consisted of, firstly drilling the desired borehole diameter (e.g. 27mm, 28mm or 35mm) to a pre-determined depth of 500 mm. The first 200 mm section of each hole was then reamed using a significantly larger drill bit. This was necessary to allow deeper anchoring of the bolt in the concrete block, thus avoiding premature cracking of the block during the loading process, as had occurred on several occasions previously. Also, the reamed section allowed any excess resin to fall out of the hole thus preventing over-encapsulation. All drill holes were checked for rifling to permit an effective concrete /resin bonding as shown in Figure 2.

Table 1 - General specifications of the bolts

	T1	T2	T3
Bolt core dia.(mm)	21.7	21.7	21.7
Profile centres	12.00	12.50	25
UTS (kN)	330	340	340
Yield Pt load (kN)	250	256	247
Profile height (mm)	0.65	1.40	1.25
Profile angle ($^{\circ}$)	21.5 $^{\circ}$	21.5 $^{\circ}$	21.50 $^{\circ}$
Profile top width (mm)	1.50	2.00	2.50
Profile base width	3.00	4.00	5.00
Bolt configuration			

**Fig 2 - Rifled drill hole**

Initially the encapsulation length was 300 mm, and this was later reduced to 260 mm as the pulling force exceeded the pulling capacity of the jack and was well above the elastic yield point of the bolt. All the bolt types used in the test had the ultimate tensile strength about 34 tonnes and yield strength around 25 tonnes. Other details of the bolts used in both investigations are shown in Table 1. For obvious reasons all the bolt types were given identification designations.

A total of 55 bolts were installed in three different borehole diameters of 27, 28 and 32 mm respectively (It should be noted that the third hole size diameter at the field study was 32 mm instead of 35 mm). The tested bolts were Bolt Types T1, T2 and T3. 45 bolts were installed in the concrete block using resin cartridge and the remaining 10 bolts were installed with PREMIX resin (known as Mix and Pore 'P1' Resin). Premixing involved mixing the resin in a container and pouring it into the hole around the bolt in the inverted concrete block.

Field Test

Field tests were carried out at the intake side of an underground local coalmine in the Illawarra Coalfield of NSW, Australia. All the holes were drilled in medium to coarse sandstone, which can be described as a competent formation. Three hole sizes were used with anchorage lengths being maintained at 300mm. The holes were initially drilled at 500 mm in length and the first 200 mm length was then reamed to 35 mm. A total of 36 bolts were installed in three different bolt diameters of 27, 28 and 32 mm respectively.

RESULTS AND DISCUSSIONS

Table 2 shows the average peak loads and peak displacement of all three bolts tested in the laboratory and in the field using different diameter holes. Figures 3 a, b and c show the laboratory results of the pull tests carried out on three different bolts and in three different diameter holes. Figures 3 d, e, and f show the field test results of the peak load and displacement values of similar type bolts in three different borehole diameter holes. Clearly the methodology of resin encapsulation application had some influence on bolt anchorage performance. The pull out anchorage loads for premix encapsulation far exceeded those obtained from cartridge types irrespective of bolt type and annulus thickness.

Table 2 - Summary of average results

Bolt	Hole dia	Average peak load		Displacement at peak load		Average shear stress		Shear Stiffness	
		(kN)		(mm)		(MPa)		KN/mm	
	(mm)	Lab	Field	Lab	Field	Lab	Field	Lab	Field
T1	27	246	190	8.05	25.1	13.9	9.30	50.0	35.0
T1	28	167	154	5.75	9.4	9.5	7.53	47.0	20.0
T1	32	>300*/ 66**	75**	3.54	8.9	2.8	3.67	-	-
T2	27	251.7	229	6.29	14.5	14.2	11.19	46.0	53.0
T2	28	235.8	155	7.04	8.0	13.3	7.58	38.0	22.0
T2	35/32	> 300*	(68)**	-	-	>16.9*	3.3	-	-
T3	27	>300	251	15.56	42	17	12.27	53.0	47.0
T3	28	252.8	179	12.63	12	14.3	8.75	46.0	17
T3	32	>300*	(16)**	-	3.0	>16.9	0.78	-	-

NB: * - Cartridge resin encapsulation , ** Premix resin encapsulation

Load –displacement

The laboratory results shown in Figure 3 were obtained from 260 mm long encapsulation, whereas the in situ field data were from 300 mm long resin encapsulation. The results from two different testing conditions have produced near similar trends. As expected, the peak pull force for widely spaced profiled Bolt Type T3 occurred at greater displacement than Bolt Types T1 and T2 as indicated also in Table 2. Such behaviour is similar to that obtained from both the CNS test (Aziz and Dey 1999) and short encapsulation test (Aziz and Webb 2003 a, and 2003b). In particular, the displacement at peak load was greatest for Bolt Type T3 bolt. Bolt Type T2 followed this, in most cases, and the least displacement was for Bolt Type T1.

The following can be deduced from both the laboratory and field test results as listed in Table 2 above:

- 1) The peak load displacement varied according to the bolt profile configuration. There was very little difference in displacement at peak load between two equally spaced Bolt Types T1 and T2 profiles, however the displacement was greater in widely spaced Bolt Type T3.
- 2) This finding was in agreement with previous reporting by both Aziz (2002) Aziz and Webb (2003a), and Aziz and Webb (2003 b).
- 3) For all three bolt Types, the average peak pulling force values and displacement at peak load was highest in the 27 mm diameter holes. This was followed by the 28 mm holes and with the least values being obtained in 35 mm holes. However, the variation in peak loads with respect to borehole diameter did not hold when the bolts were anchored with pre-mix resins encapsulation.
- 4) Premix resin encapsulation was found to be superior in performance to the cartridge type. This is obviously clear from the results of the tests in the laboratory for all three bolts and as evident Figure 3c.
- 5) The reduced performance of pull out force with increased annulus thickness was considered to be attributed to insufficient resin mixing leading to excessive gloving, which is discussed later
- 6) Rifling of the hole (Fig 1) prevented the failure along the resin /concrete interface.

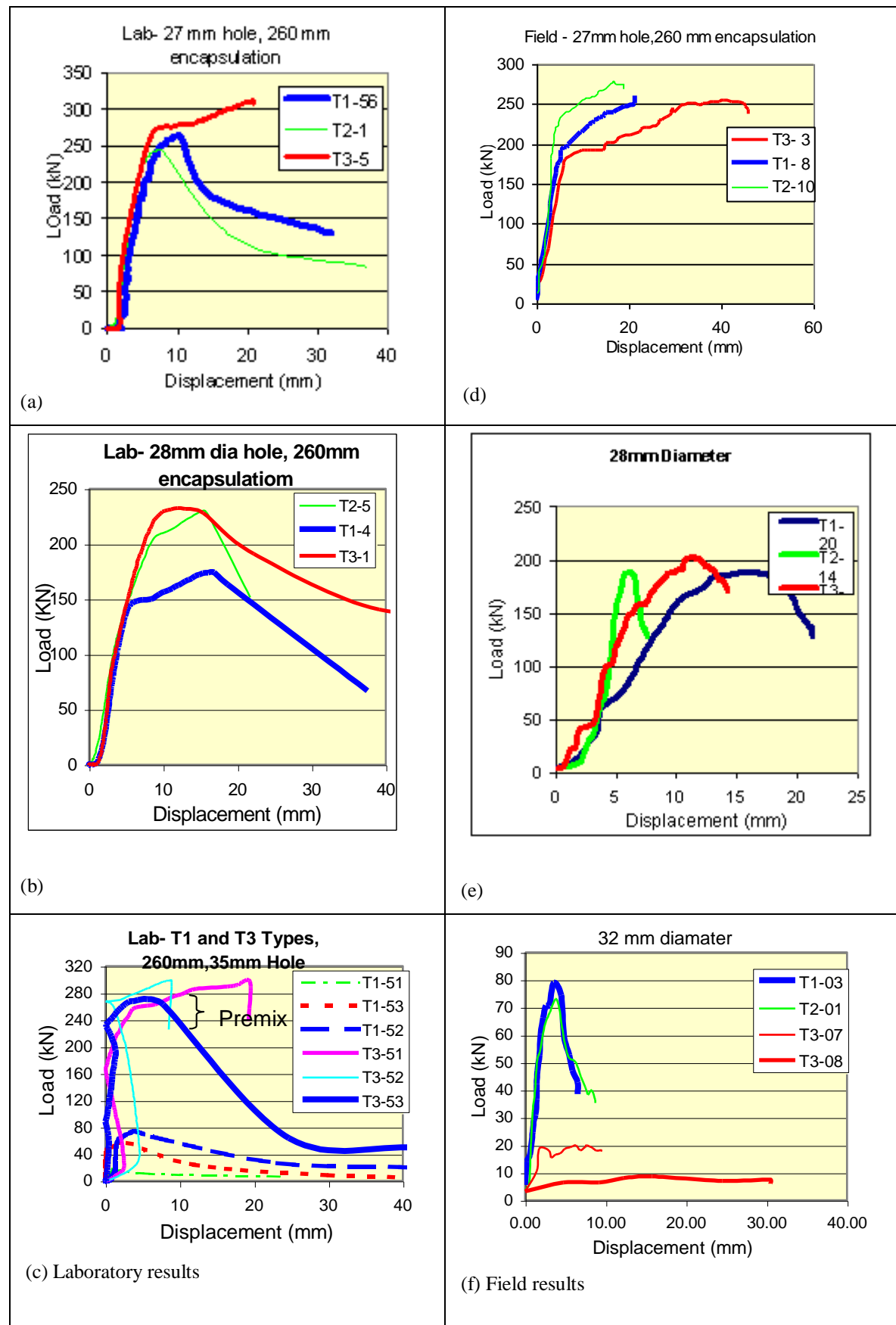


Fig 3 - Laboratory and field test results for different bolts

Encapsulation Annulus Thickness

Figure 4 shows the load displacement profiles of Bolt Type T1 in different diameter holes obtained from both the laboratory and field tests. Both tests clearly demonstrated that the increased resin annulus thickness had an adverse influence on the bolt performance. A closer examination of the results in Figures 3 c and 3 f, revealed the same pattern of results for bolt Type T1 and T3 respectively, but although at different rates. No tests were made on Bolt Type T2 in 35 mm holes.

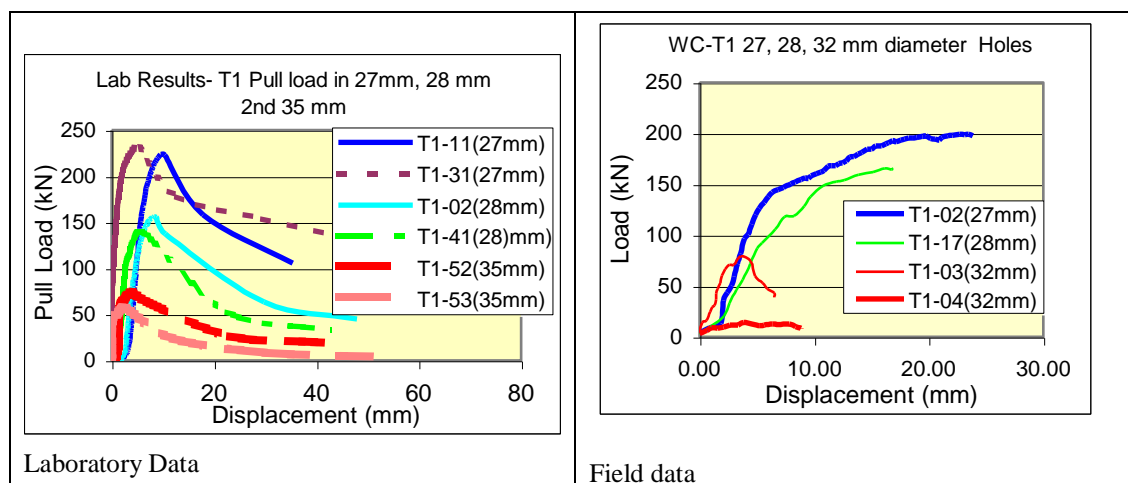


Fig 4 - Load-displacement of Type T1 Bolt in different annulus thickness encapsulation

No differences in performances were observed in the laboratory tests when all three-bolt types were installed in different diameter holes using premix resin encapsulation. As can be seen in Fig. 3 c, the peak pulling load of premix resin installed bolts were around 300 kN. The results demonstrated that increasing annulus thickness of the encapsulation was due to the quality of resin mixing and the degree of gloving formation and variations in resin encapsulation thickness. Bolts with higher and closer spaced profiles are likely to provide better mixing capability of the resin encapsulation than the bolts with lower and wider spaced profiles. This is because the high and closer spaced profiles (eg. Bolt Type T2) may generate more effective spinning force, allowing better shredding of the resin cartridge sleeve than the bolts with wider and lower profiles.

In an endeavour to examine the role of increased annulus encapsulation thickness on resin anchorage strength, a comparative push test was made using two different encapsulation thicknesses. As can be seen in Figure 5, that there was a dramatic reduction in pulling force between the two-encapsulation thicknesses. In both cases the same profile type of bolt was used.

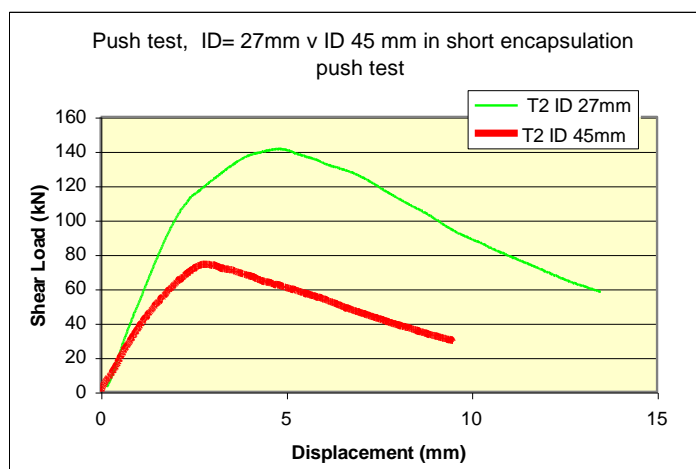


Fig 5 - Comparison of push tests in different internal diameter steel sleeve

PUSH/PULL TESTING IN STEEL SLEEVES

Concern has been expressed about the methodology of testing, in which the bolt is pushed out of the steel tube rather than being pulled. In reality the installation and subsequent performance of bolts in-situ results in the bolt being in tension and sometimes in tension and shear. There will be a general reduction in bolt cross section as a result of bolt tensioning, causing premature bolt resin surface contact failure and loss of grip. Scepticism has been expressed on the role of bolt profile configuration and its influence on load transfer capacity of bolt /resin interface. Accordingly the following two sets of tests were undertaken;

1. Increasing the rib profile spacing of Bolt Type T2 by filing away the alternative ribs
2. Pull testing of the bolt from the steel sleeves instead of the conventional push test

Profile spacing: Figure 6 shows the results of profiles of the Bolt Type T3 and Bolt Type T2 with its alternate ribs being filed away. As can be seen, the removal of alternative ribs from the bolts has increased peak load displacement, which is as close to that of Bolt Type T3. It is unlikely the profile configurations would fit to each other because of different profile spacing and profile height as shown in Table 1. However the results clearly demonstrate the influence of the profile configuration on load transfer mechanism relationships. This finding should be taken into consideration for future designs in different ground conditions, see Aziz and Webb (2003a, and 2003 b), and Aziz, Dey and Indraratna (1999).

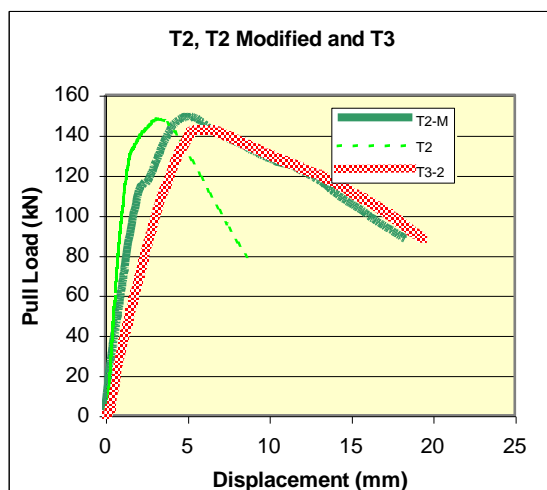


Fig 6 - Load displacement profiles of Bolt Types T2 and T3 and a modified Bolt Type T2 with alternative profile removed.



Fig 7 – Pull tests arrangement

Bolt pulling through the steel sleeve: Figure 7 shows the test set-up for pulling a bolt out of 75mm steel sleeve, and Figure 8 shows the pull test load /displacement results of bolt Type T2.

Table 3 shows the comparative test results of pull and push tests. Both push and pull test samples were prepared from the same premix resin batch. Also included in the table are the average values of push tests carried out by Webb (2001).

There were some variations in the values from different bolts. The difference between the average push and pull test results for both Bolt Types T1 and T2 were in the range of between 8 and 11 % respectively. Further research is continuing to examine other profiled bolts

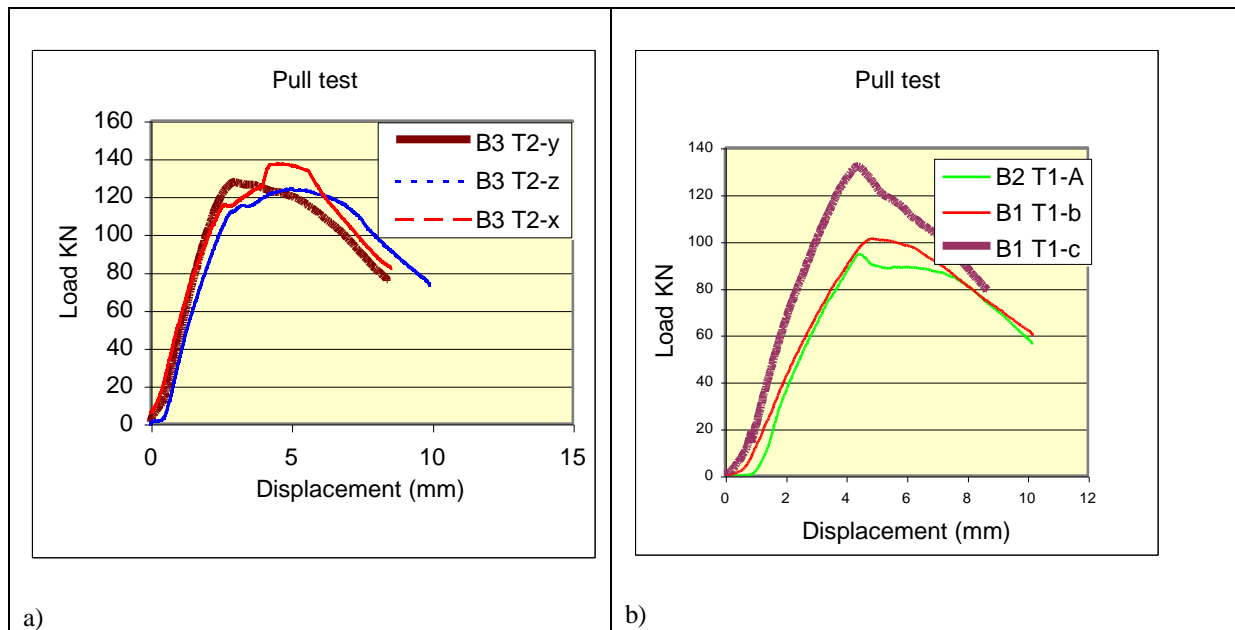


Fig 8 - Pull test results for Bolt Types T1 and T2

Table 3 Data from push and pull tests

Bolt Type	Av peak push load and SD (kN)	Av Peak pull load and SD (kN)	Diff (%)	Ave peak Load displacement Push and SD (mm)	Ave peak load displacement and (SD) (mm)	Diff (%)
T1	130.7 (± 10.85)	120.06 (± 16.4)	8	3.55 (± 0.63)	4.60	13
T2	140.31 (± 6.0)	129.4 (± 6.95)	9	4.85 (± 0.82)	7.92	39

CONCLUSIONS

Both programmes of the experimental study have led to the following conclusions:

- 1) Bolt surface profile configuration play a dominant influence on the load transfer capacity of the bolt. The hight and profile spacing affect the level and sustainability of the transfer mechanism. Wider spaced profiled bolts maintains peak pull load at greater displacement than the closely spaced bolts. Also post peak load tapers off gradually as compared to narrow spaced and low height profile bolts.
- 2) Changing the profile configuration of the bolt caused a change in load transfer capacity of the bolt
- 3) The strength of resin encapsulation is influenced by the annulus thickness of encapsulation. Also affecting the strength is the quality of resin mixing and degree of gloving formation.
- 4) Premix resin encapsulation was found to be superior in performance to the cartridge type resin.
- 5) The methodology of removing the bolt in short encapsulation tests has an influence on the pulling results. There was a difference of 10% between the bolts pushed and pulled out of short encapsulation tests.

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MODELLING OF SHEARED BEHAVIOUR BOLTS ACROSS JOINTS

Hossein Jalalifar¹, Naj Aziz¹, and Mohammad Hadi¹

ABSTRACT: A three dimensional numerical model was developed to simulate the shearing of reinforced joints. Reinforcement of the shearing surfaces is effected with pretensioned bolts installed perpendicular to the sheared joint surface. The influence of bolt pretension forces examined included 20 kN, 50 kN and 80 kN respectively and aimed to complement the experimental work on double shearing of bolts installed in two different strength concrete blocks. Post shear stresses were analysed for both linear and nonlinear regions of the load - deflection curve. Simulation of several models in varying conditions provided a better understanding of the role of bolt pretensioning in sheared joint and bedding plane reinforcement. There was a clear relationship between the level of bolt pretensioning and the shear load applied. It was shown that the strength of the sheared composite medium was influenced by the applied shear load. The modeling study is part of a comprehensive programme of research work aimed at providing a better understanding of load transfer mechanisms in bolt /resin /rock for effective strata reinforcement.

INTRODUCTION

Significant research have, in the past, been undertaken to study the mechanical behaviour of bolted rock joints, Spang and Egger (1991), Pellet and Boulon(1993), Ferrero (1995). Bjurstrom (1974) was the first to report on the systematic research work on fully grouted rock bolts. His shear tests were conducted on fully cement grout bonded rock bolts embedded in blocks of granite. According to Bjurstrom, inclining the bolt resulted in stiffening the shearing surface by increasing the shear strength at small displacement. Dight (1982) carried out a series of laboratory tests, to evaluate the shear resistance of bolted joints using various materials and he found that the normal stress acting on the joint surface had no influence on the shear resistance and joints with inclined bolts were stiffer than the perpendicular ones. Dight (1982) proposed an expression to predict the maximum force mobilized in the bolt. He found that the failure of the bolt was caused by the combination of axial and shear forces. Ferrero (1995) proposed a shear strength model for reinforced rock joints based on numerical modeling and laboratory tests. The overall strength of the reinforced joint was considered to be the combination of both the dowel effect and the incremental axial force increase due to the bar deformation. Also, Ferrero proposed a modified analytical model for bolts installed perpendicular to the joint plane in stratified bedding plane. Aziz, Pratt and Williams (2003) conducted laboratory studies of double shearing of bolts in concrete and found that the medium strength and the axial tensional load influenced the level of shear load.

As a continuation of the research on bolting, a programme of numerical modeling is currently been undertaken to simulate the role of profile configuration on the load transfer mechanism bolt pull/push testing and the influence of bolt pretensioning across the sheared surfaces to complement the laboratory studies by Aziz, Pratt and Williams (2003). The 3D FEM modeling was carried out with ANSYS 3D.

EXPERIMENTAL STUDY

Figure 1 shows the general set up of the assembled double shear box unit in a testing machine and sketch of a deformed bolt together with a post testing deformed bolt. Details of the experimental study on the shear behaviour of bolts in jointed concrete blocks were reported by Aziz, Pratt, and Williams (2003). Three bolt

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types were used for the study, they were known as Bolt Types T1, T2 and T3 Respectively. The properties of various bolts are reported by Aziz (2004).

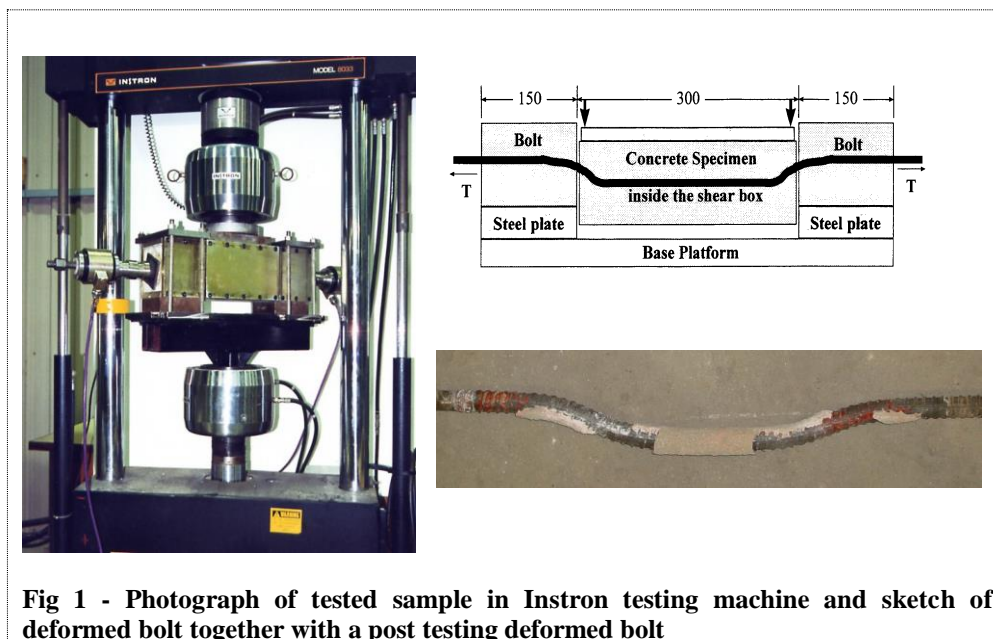


Fig 1 - Photograph of tested sample in Instron testing machine and sketch of deformed bolt together with a post testing deformed bolt

Tests were made with axial confining loads of 20, 50 and 80 kN respectively. The development of shearing loads was examined with respect to the surface profile configuration of the bolt used. Shearing tests were conducted in two different concrete blocks of 20 MPa and 40 MPa strength. Figure 2 shows a typical Shear load and displacement profiles of three different bolts cast in 40 MPa concrete.

The following points were noted from shear load and deflection graphs (Aziz, Pratt and Williams, 2003):

- 1) The shear load of the bolt increased with increasing bolt tension. This behaviour was obvious in bolts with low profiled and widely spaced profiled bolts.
- 2) The strength of the medium has influenced the shear load level but not the trend. Shear load values for all bolts were generally less in 20 MPa strength concrete medium in comparison to the shear load values of bolts tested in 40 MPa concrete.

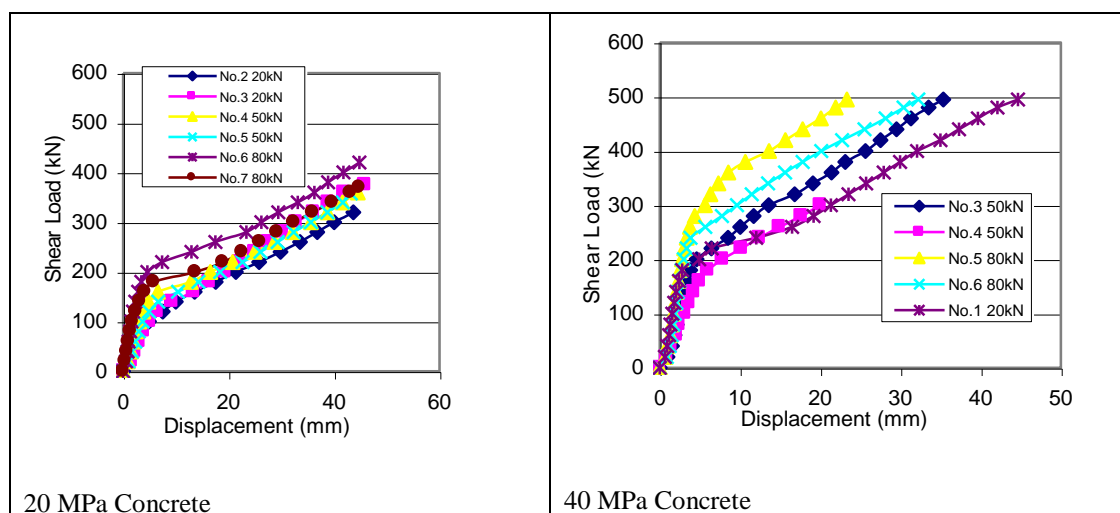


Fig 2 - Shear load and shear displacement of a bolt tested in both 20 and 40 MPa strength concrete and under different tensile loading conditions

- 3) The shear displacement at elastic yield point was not consistent irrespective of the concrete type and the axial load. This was the same for all three bolt types tested.
- 4) High profiled and closely spaced bolts such as Bolt Type T2 displayed constant shear load at all three levels of bolt tension loads in both 20 and 40 MPa concrete mediums. The consistency of shear loads at bolt type elastic yield point was more pronounced than the other bolt types.
- 5) Deflected bolt sections experienced regions of tension and compression. Resin columns remained adhered to the sides of the bolt region that experienced compression, but had broken off the sides that were in tension (Figure 1)

3D NUMERICAL ANALYSIS

A three dimensional finite element model of the reinforced structure subjected to the shear loading was used to examine the behavior of bolted rock joints and compare it with experimental results. Three governing materials (steel, grout, rock) with two interfaces (bolt-grout and grout-rocks) were considered for the 3D numerical simulation.

A general purpose finite element program (ANSYS, Version 7), specifically designed for advanced structural analysis, was used for 3D simulation of elasto-plastic materials and contact interfaces behaviour. The model bolt core diameter (D_b) of 22 mm and the grouted cylinder (D_h) of 27 mm had the same dimensions as those used in the laboratory test. Due to the symmetry of the problem, only one fourth of the system was considered here. Figure 3 shows the three-dimensional model, which consists of 9964 nodes and 6460 elements, and Figure 4 shows the model cross section.

The elastic behavior of the elements was defined by Young's Modulus and Poisson's ratio of various materials as per Table 1.

Table 1 - Elastic material properties

Material	E (GPa)	Poisson's Ratio
Concrete (20 MPa)	20	0.25
Concrete (40 MPa)	32	0.25
Grout	12	0.2
Steel	200	0.3

The interface behaviour of grout-concrete was considered as a perfect contact, and was determined from the test results. However, the low value of cohesion (150 KPa) was adopted for grout-steel contact.

3D solid elements (Solid 65 and solid95) that have 8 nodes and 20 nodes were used for concrete, grout and steel respectively, with each node having three translation degrees of freedom, that tolerated irregular shapes without significant loss in accuracy. 3D surface-to-surface contact elements (contact 174) were used to represent the contact between 3D target surfaces (steel-grout and rock-grout). This element is applicable to 3D structural contact analysis and is located on the surfaces of 3D solid elements with midside nodes. The numerical modeling was carried out at several sub steps and the middle block of the model was gradually loaded in the direction of shear. Figure 5 shows deformed shape of elements.

Simulation of several models in varying conditions (a range of bolt pretension load and concrete strength) were carried under a vertical load and results were analysed for both linear and non-linear regions of the load - deflection curve.

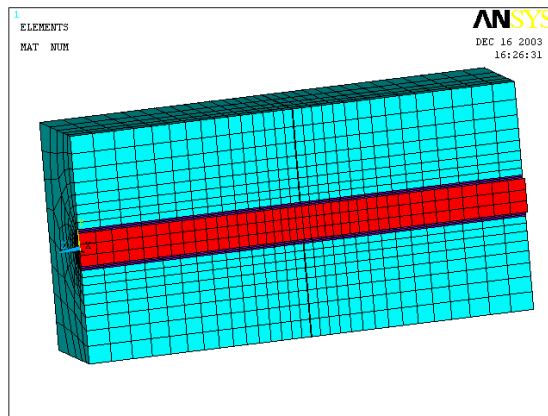


Fig 3 - Three-dimensional image of the numerical model

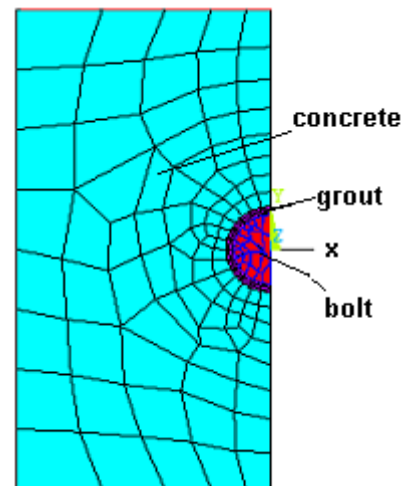


Fig 4 - Cross-section of the numerical model

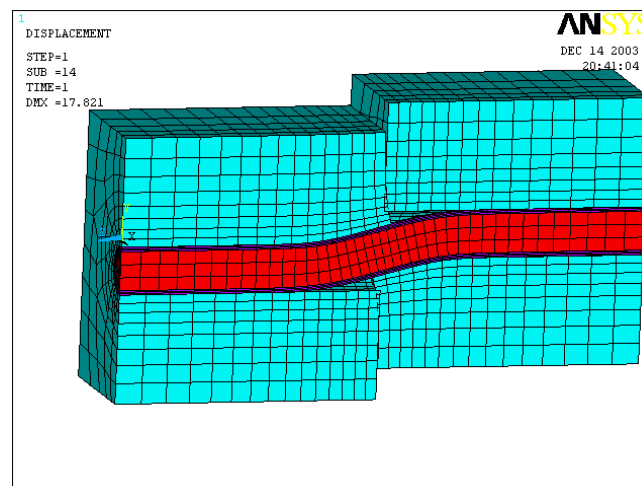
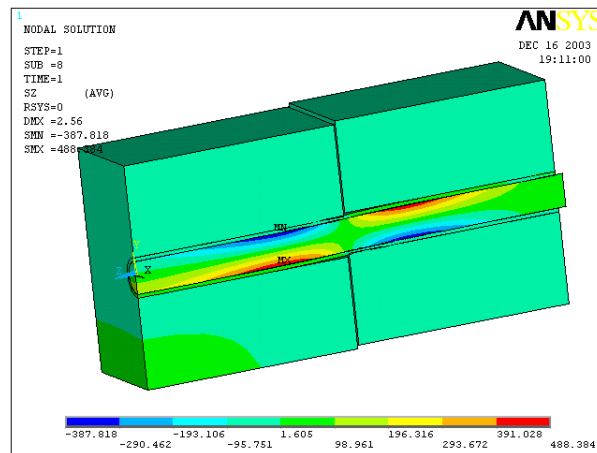


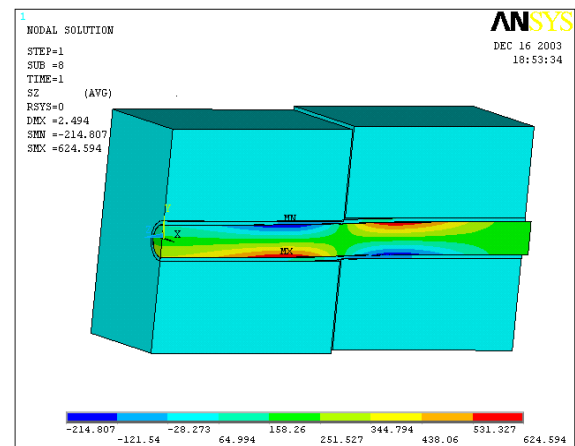
Fig 5 - Deformed shape of elements

NUMERICAL RESULTS

Comparison of the stresses developed in both 20 and 40 MPa concrete, with the bolt pretension loads of 20 and 80 kN at both pre-failure and post failure were examined. In both concrete strengths (20 MPa and 40 MPa), the increase in bolt pretension led to the increase in the tensile stresses in the axial direction of the bolt (Figures 6 and 7), and the compressive stresses reduced, this trend is more dominant in the linear region than post failure region.

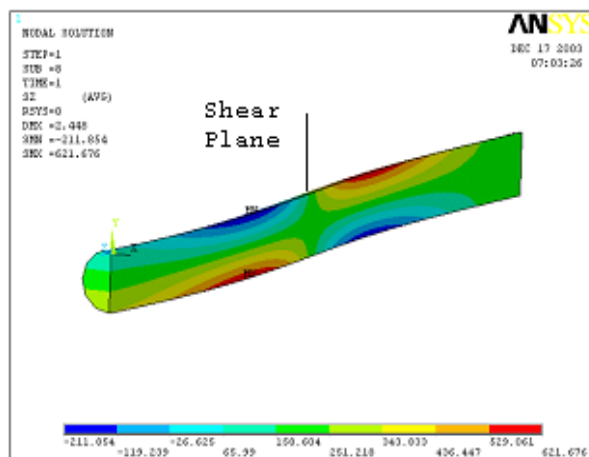


a - 20 kN initial tensile load

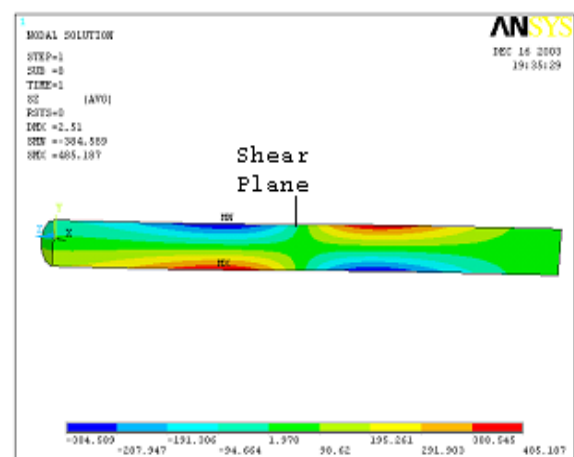


b - 80 kN initial tensile load

Fig 6 - Stress contours along the bolt length in 20 MPa concrete and in different pretension loads (a: 20 kN, b: 80 kN) in the pre-failure region.



a 20 kN



b 80 kN

Fig 7 - The stress contours along the bolt in 40 MPa concrete and in different pretension loads (a 20 kN and b 80 kN) in pre-failure region

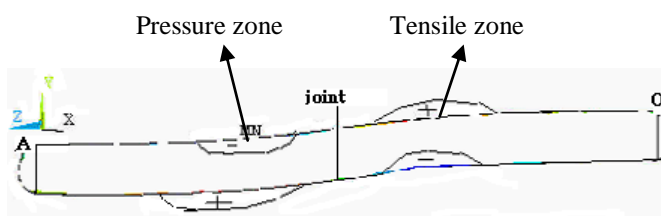


Fig 8 - Tensile and pressure stresses zones around the bolt

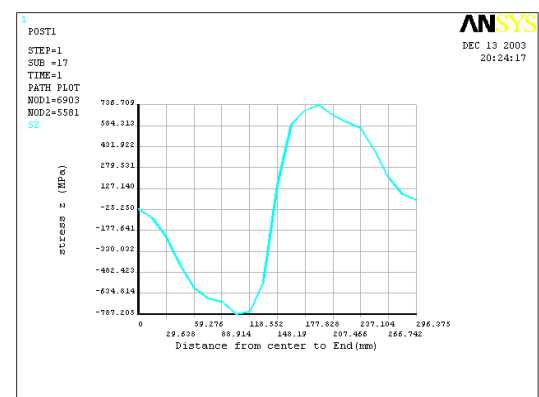


Fig 9 - Stress changes along the bolt

Increasing the confining pressure causes a reduction in bolt deflection but this reduction, if it occurs prior to yield point, is not significant as demonstrated in both the experimental and numerical results. However, the effect of pretension in post failure has significantly affected bolt deflection and that is demonstrated in the numerical and experimental results. As can be seen from Figure 8 the stresses in the upper half of the bolt and towards the perimeter are tensile while it is compressive at the centre. However the stress conditions at the lower half section of the bolt is reversed. This can also be observed from the experimental results as shown in Figure 1. The degree of the changes in the post failure region is plotted in Figure 9. It can be seen that stresses in these zones are high and the bolt appears to be in a yield situation. The location of these stresses is shown in Figure 10. Figures 11 and 12 show the shear stress contours along the length of the bolt at different pretensions and different concrete strengths. The maximum shear stress is concentrated in the vicinity of the joint plane. These stresses slowly increased after beginning at the plastic deformation level and finally ending at a stable situation. By increasing bolt pretension load, the shear stresses were decreased in both concrete types (20 MPa and 40 MPa), causing an increase in the bolt resistance to shear. With increasing \shear loads, deflection in the bolt increases and plastic strain is created in critical locations in all materials. The situation of these strains and the rate of strain changes along the model are shown in Figures 13a and 13b respectively. Figure 13 b shows the rate of changes in plastic strain along the bolt. Contact pressure contours were found to decrease with increased the confining pressures and visa versa.

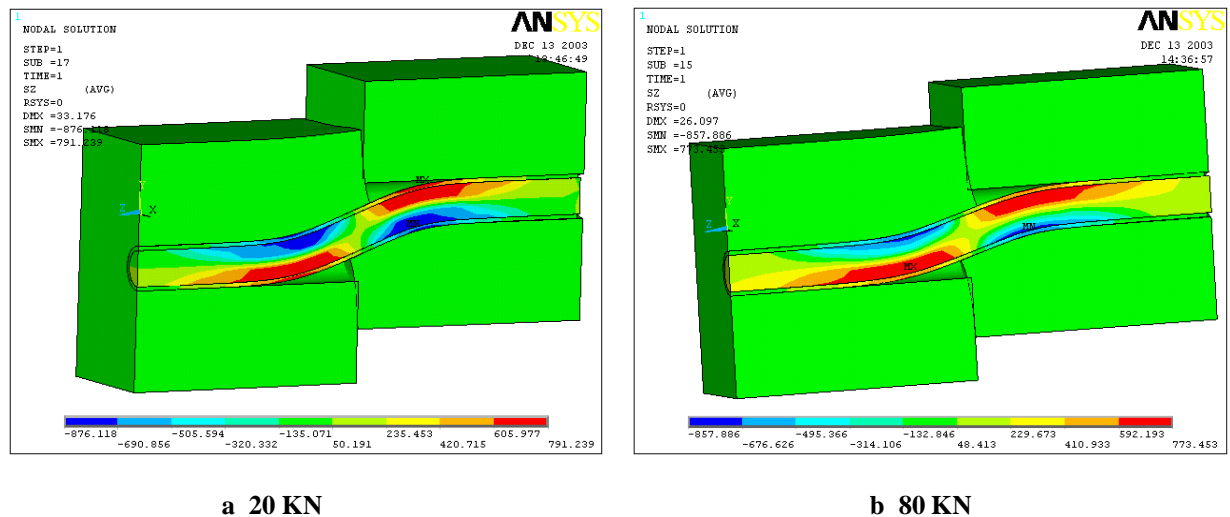


Fig 10 - The stress contours along the bolt in 20 MPa concrete strength with different pretension I

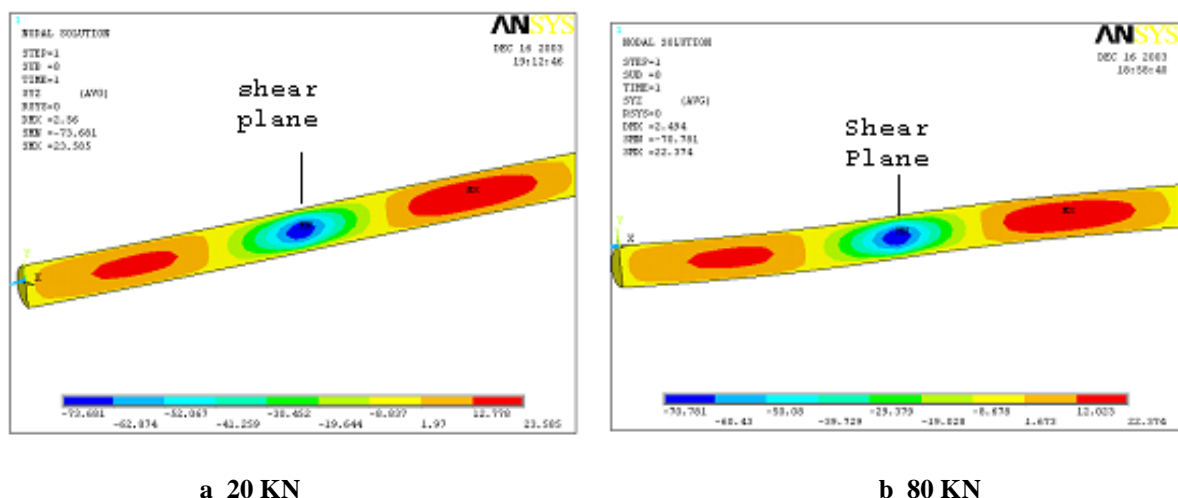
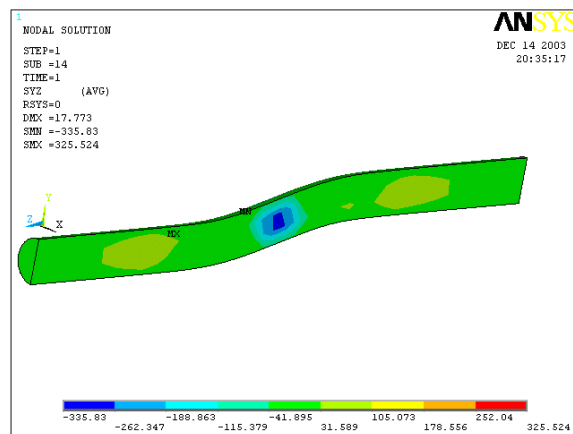
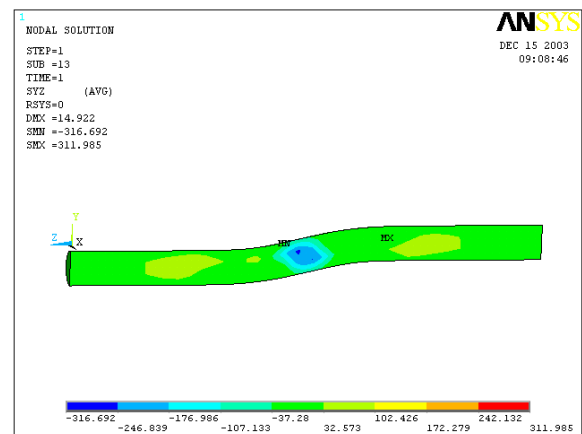


Fig 11 - Shear stress contours along the bolt in different pre-tension (a: 20 KN and b: 80 KN) with 20 MPa concrete strength in linear region. loads (a: 20 KN and b: 80 KN) in post-failure region

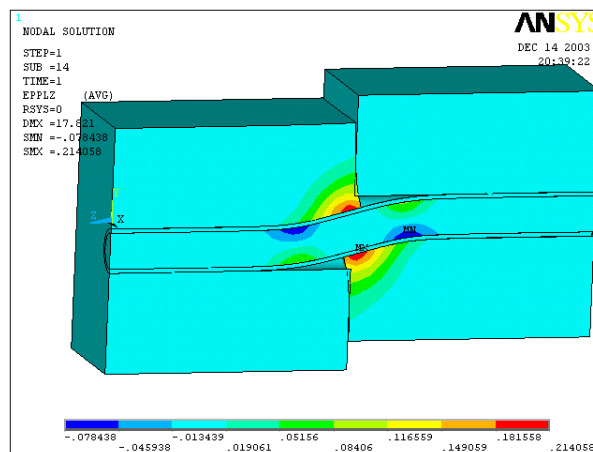


a 20 kN

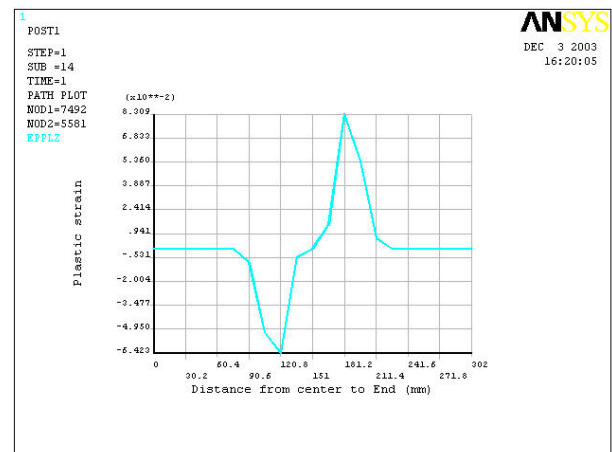


b 80 kN

Figure 12 - Shear stress contours along the bolt in different pre-tension (a: 20 kN and b: 80 kN) with 40 MPa concrete strength in the post failure region



a Plastic strain contour



b Plastic strain changes

Fig 13 - Plastic strain contours and the rate of changes in concrete 40 Mpa strength with 20 kN pretension load

Fig 14 displays the contact pressure contour between the bolt-grout interfaces. With increasing shear load contact pressure was found to increase. However, the increase of pretension has reduced contact pressure. In addition induced stresses are produced in concrete blocks causing it to fracture and fail. Minimum main stresses in the corners of concrete blocks that are affected by the steel pressure are shown in Fig 15. As it can be recognized, the stresses around the edges in this area are high and higher pressure can induce longitudinal fractures in concrete blocks. This was also recorded in the experimental results.

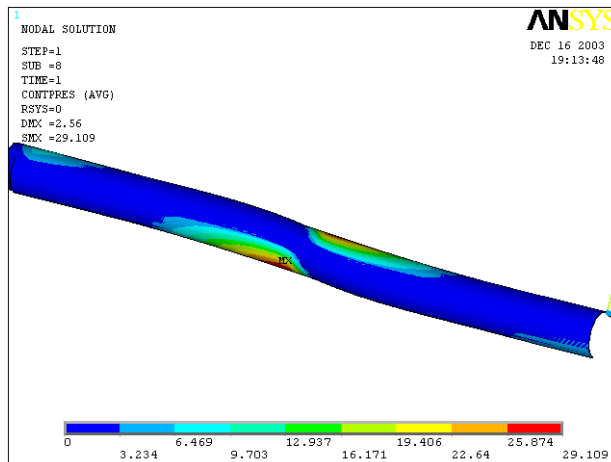


Fig 14 - Contact pressure contours in concrete 20 MPa strength with 20 KN tensile load in the pre-failure region

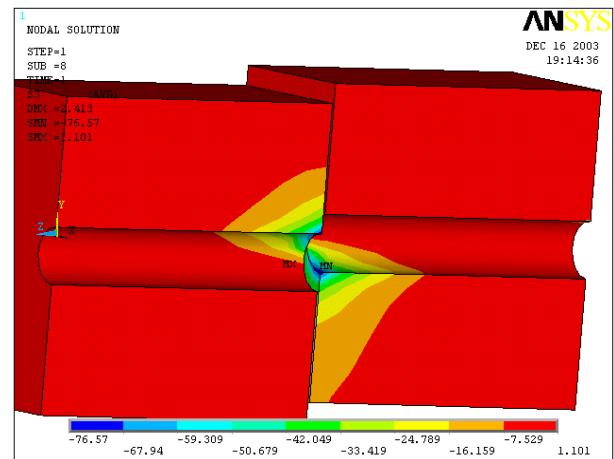


Fig 15 - Min stress contours in pretension load in the pre-failure region in concrete of 20 MPa strength and 20 KN

CONCLUSIONS

The numerical simulation of shearing associated with the reinforced joints and bedding planes provides a unique insight into the build up and distribution of stress when shearing occurred. The build up of stresses at both the bolt's elastic and plastic ranges were clearly demonstrated. Stresses generated as the shearing occurred on the reinforced block was quantified, which permitted a better understanding of the reinforcement applications and the role of bolt pretensioning. A number of conclusions drawn which were based on the numerical solutions were found to be in agreement with the experimental results and in particular:

- The maximum deflection at a particular applied vertical load was observed in softer medium and lesser initial tensile load.
- The tensile and compressive stresses were successfully simulated on each side of the shear plane. The level of stresses increased with increasing shearing load. However, post yield point there was no significant changes in their locations (Figures 1, 8, 10).
- Higher values of shear stresses were concentrated at the bolt sections near the shear plane (Figures 11,12) and by increasing pretension load causes led to reduction in shear load.

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SUCCESSFUL USE OF A STRESS RELIEF ROADWAY AT APPIN COLLIERY.

Rod Doyle¹ and Winton Gale²

ABSTRACT: High horizontal stress levels can lead to extensive roadway deformation requiring expensive secondary support to ensure stability; this is particularly the case with longwall installation faces. Longwall installation roadways are a critical construction within coal mines. The use of a purpose built 'Stress Relief Roadway' to minimise roof deformation in the nearby longwall installation roadway, by reducing stress impacts has been undertaken at Appin Colliery – BHP Billiton Illawarra Coal. Its use led to significant cost and operational benefits. This paper outlines the process used; from identifying horizontal stress as an issue, as well as generating computer models through the various options and culminating in ground monitoring of the constructed roadways to the successful start of the longwall panel.

INTRODUCTION

Appin Colliery is located in the Southern Coalfield of New South Wales some 30km from the city of Wollongong and around 40kms from the port of Port Kembla. The Bulli seam is coking coal and is utilised in the local and international steel making industries. Appin Colliery has been operating for some 40 years and during that time has produced in excess of 40 Million tonnes of coal. The mine has been the backbone of coal production for the BHP Billiton Illawarra Coal Group of collieries.

Overburden at the mine consists of sedimentary rocks with a depth of cover at about 500m and as a result the vertical loading is of the order of 12-14MPa. Horizontal stress is of greater significance and measurements have been taken from near the area of interest. Stress levels range in the order of 20-30MPa and are significant enough to cause serious roof deformation. The previous two longwall panels had stress effects that caused operational issues with both longwall installation faces. This was the primary cause to seek a viable alternative – the stress relief roadway.

PURPOSE

The purpose of a stress relief roadway is essentially to improve roadway conditions in the nearby wider longwall installation roadway at the expense of the pre-driven stress relief roadway. The logic behind such a development is that the initial driveage suffers deformation as a result of the high horizontal stress. The stress causes significant roof guttering and or slabbing. This process in turn forces the stress field to readjust itself and the later formed longwall installation roadway is driven in an environment protected from the higher stress.

HISTORY

Stress Relief Roadways have a long and successful history of operation at Appin Colliery. Figure 1 shows the mine layouts and identifies mining areas and the region of the stress relief roadway discussed in this paper. Figure 2 highlights the area this paper deals with. In some instances lifting off of the relief roadway was undertaken to ensure that significant deformation took place, thereby ensuring stress relief for the wider installation face.

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² Strata Control Technology Operations

For some time the mine stopped providing stress relief roadways as it was considered that they interfered with the critical path development rates. But in practise experience has shown that a smoother longwall installation process offsets the economics of slower development rates.

GEOLOGICAL MAPPING

Geological mapping of all roadways in the underground environment is undertaken to monitor geological structures as well as stress deformation. Hazard plans for the mine are prepared and indicate the nature of hazards, such as faults, dykes, stress and gas as well as cross-sections indicating the proximity of adjacent coal seams.

While the immediate roof strata varies it is predominantly a sequence of interbedded fine-grained sediments typically consisting of massive siltstones or fine-grained sandstones ranging in strength from 30-80MPa (UCS). Estimates of Coal Mine Roof Rating (CMRR) would be in the vicinity of 40 to 50 (weak to moderate).

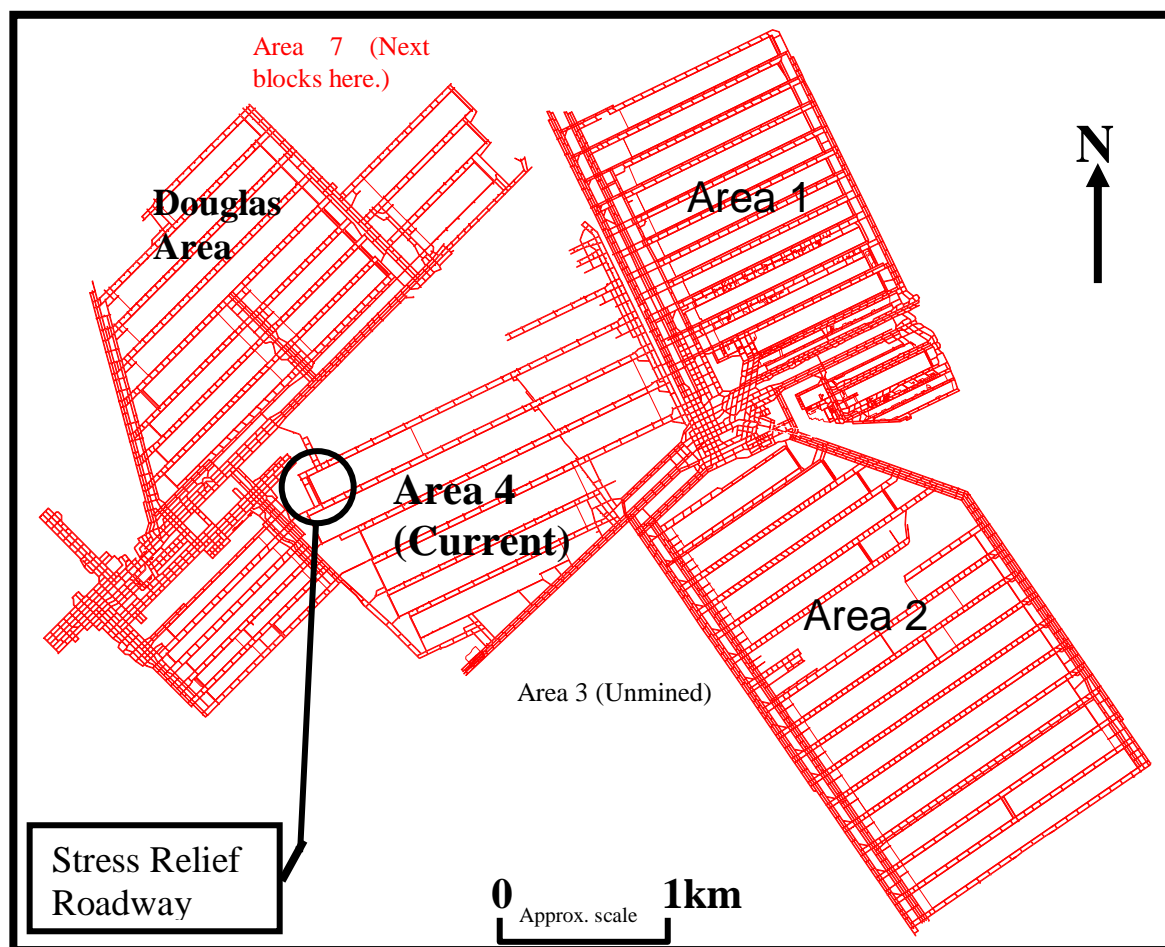


Fig 1 - Appin Mine layout – goafs not blacked out.

STRATA CHARACTERISATION & COMPUTER MODELLING

To ensure the correct geometry for the stress relief roadway and longwall installation face separation it was considered advantageous to conduct computer modelling. Strata Control Technology Operations (SCT) undertook extensive computer modelling to simulate the behaviour of the roof around the roadways in the area of the longwall 406 installation face.

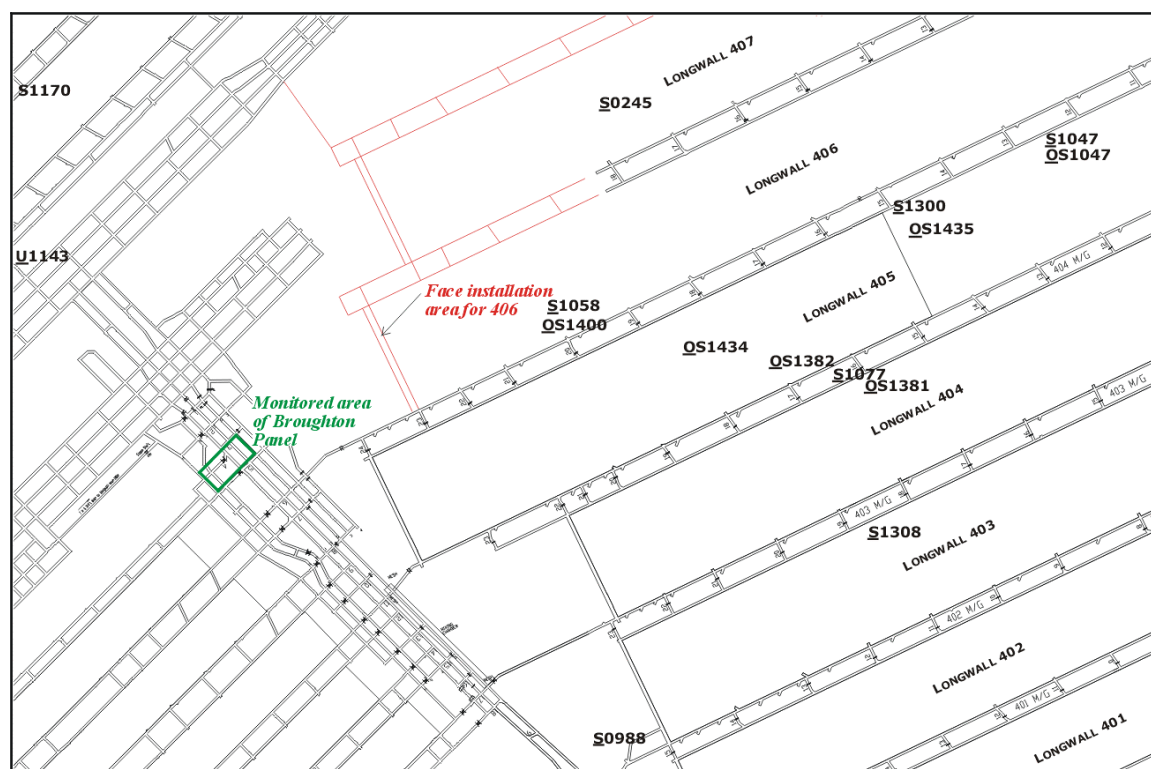


Fig 2 – Mine layout and borehole location

Previous studies undertaken by SCT focussed on the nearby Broughton Panel of the previous Tower Mine – now incorporated as part of Appin Colliery (See Figure 2.). Information available to develop a model of the strata included bore logs and core photographs, geophysical logs and underground monitoring together with extensometer data. Some UCS data was available and estimates were also made from geophysical logs, the latter combined with core assisted in developing bedding plane cohesion charts.

In the roof fine-grained sediments make way for sandstones and generally range from 40-90MPa. The magnitude of the horizontal stress is estimated in the range of 25-28MPa. The Bulli seam is some 3.30m in thickness and has an estimated in-situ strength of 6MPa and the floor is typically shale about 1.5m thick before grading into sandstone. This sandstone overlies the Balgownie Seam which is the latter being about 1.0m in thickness and about 8m below the Bulli Seam.

It should be recognised that computer models are based on the characterisation of the strata and in this instance have been used to simulate the likely strata behaviour during drivage. Underground mapping and observations were also conducted prior to running the model to improve confidence in the outcome. The model utilised the FLAC Version 3.4 with rock failure and stress input routines that have been developed by SCT Operations.

The model has the capacity to review the drivage of individual roadways or a sequence of drivages to determine their impact upon one another. Thereby assessing the impact of stress relief on the installation road. Figures 3, 4 and 5 highlights various aspects of the modelling. Figure 3 highlights the construction of the model and the breakdown of the strata horizons that have been characterised – note their UCS strengths. Figure 3 shows the final layout of the stress relief roadway (Right Hand Side) and the longwall installation roadway (LHS).

Figure 4 examines the case of the single roadway. For a single roadway the model indicates that the height of fracturing will extend to 5m above the roof of the opening when the stability level of stress is exceeded. The stability limit is estimated at about 17MPa for the existing ground conditions. The nature of the fracturing is clearly evident in the model.

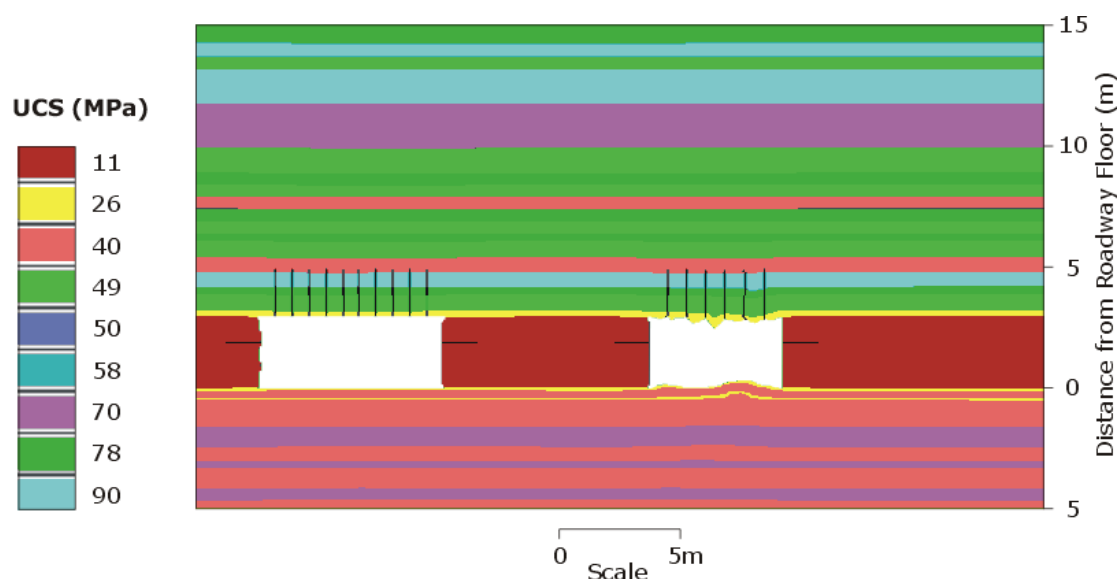


Fig 3 – UCS model of roadway and face heading

The benefit of a stress relief roadway is that it deforms significantly enough that it causes the stress field to readjusts so that it doesn't act to the same extent upon a nearby roadway i.e. the longwall installation roadway. The critical factor is to ensure that there has been sufficient stress relief to achieve the desired benefit. In some older examples of stress relief roadways lifting off coal (fendering) ensured that the benefits of the extra wide stress relief roadways was achieved and that the stress field was actively interfered with. However in this instance fendering was postponed to observe the nature of the monitoring and when results were satisfactory fendering was not considered essential. While SCT Operations made a recommendation that fendering of the stress relief roadway be undertaken – in this instance this was not undertaken. The purpose of this was to induce more failure at a greater height into the roof. Thereby ensuring that the stress was driven well above the influence levels likely to impact upon the installation roadway.

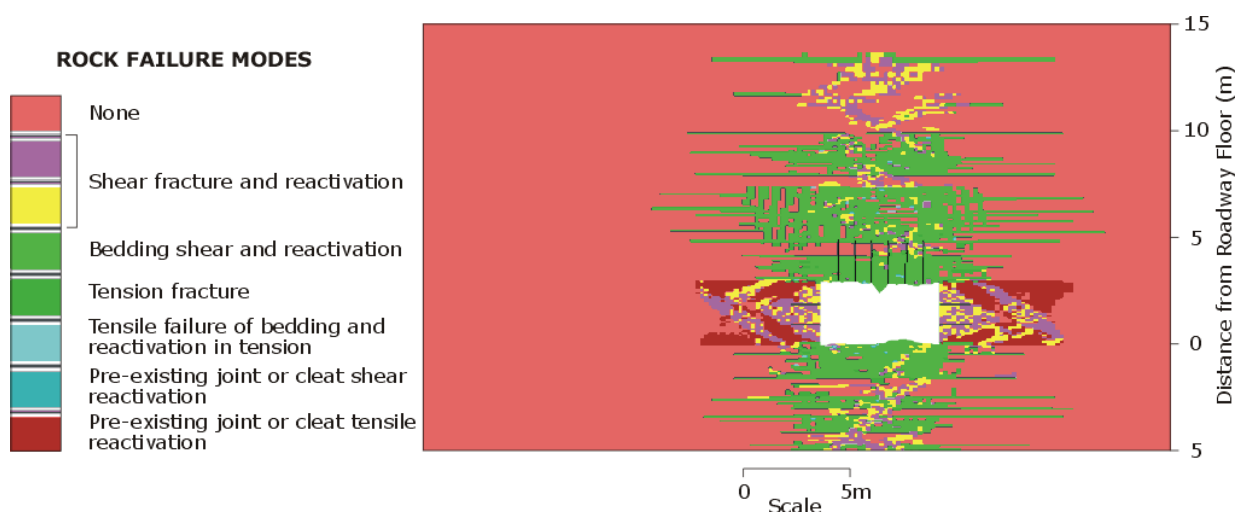


Fig 4 – Rock fracture mode and distribution for a single roadway at 20MPa

Running the various models (Figure 5) allowed a recommendation of a distance between 8 to 15 metres of solid coal between the stress relief roadway and the face installation roadway. It was considered that this would allow the stress field to be sufficiently modified to allow a successful longwall installation roadway to be opened up (first pass) and then widened out. Figure 5 shows the level of deformation around the roadways. The roof displacement anticipated in the face installation roadway was less than 10mm.

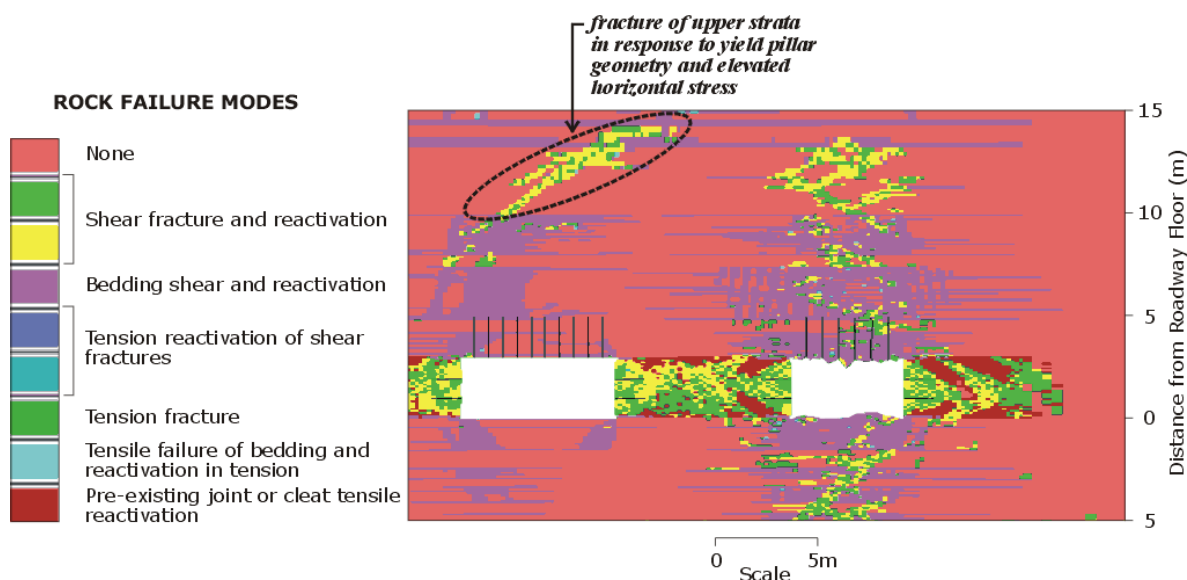


Fig 5 – Rock fracture distribution for stress relief installation roadway – 8m pillar

COST BENEFITS

In the past stress relief roadways were utilised quite successfully at this mine. However with the drive to ensure longwall continuity and reduce costs the stress relief roadway process was abandoned. However on longwall 405 the face installation roadway suffered considerably due to the impact of high horizontal stress to such extents that it resulted in several episodes of PUR injection and considerable installation of megabolts to lock the roof up. These secondary support measures were successful and were a credit to their use. Nevertheless the cost of secondary support for this installation heading was excessive. Alternative measures were sought for longwall 406 panel.

The additional cost of drivage – in this instance some 255m of drivage at a nominal cost of \$1500 p.m. (\$380K) was offset by the revenue of the coal mined ($5\text{m} \times 3.2\text{m} \times \text{SG}1.4 \times \$\text{A}67 = \$1500$ – Example Only). This essentially makes the cost of the heading cost neutral in terms of economics. However the real cost is therefore in the use of the mining equipment and human resources to cut the stress relief roadway when it could be producing additional development metres in a different area in other words - opportunity. Which can be an important factor in its own right. However, when considering the potential for difficulty in a face installation roadway and the additional cost of secondary support together against the cost of driving the stress relief roadway it is considered to be a minor investment which pays huge dividends in terms of having a face installation roadway of excellent attributes. This presents a greater opportunity to increase and improve longwall productivity, which is where the real benefits of a smooth transition flow.

OBSERVATIONS & MONITORING

As part of the ventilation requirements a guaranteed back heading was required to remain standing during longwall retreat and as such a twin heading panel was driven to create the 'bleeder' (ventilation roadway) and the stress relief roadway. The drivage experience of these roadways was as expected with both roadways

suffering from some significant stress deformation. Tell tales were installed in the bleeder roadway at 50m spacings and ranged from 80 to 100m of total convergence. Six bolt patterns spaced at one-metre intervals controlled the roof and full mesh was utilised. Full mesh modules were used to minimise any slabbing of the roof and thereby ensure the safety of mining personnel during drivage.

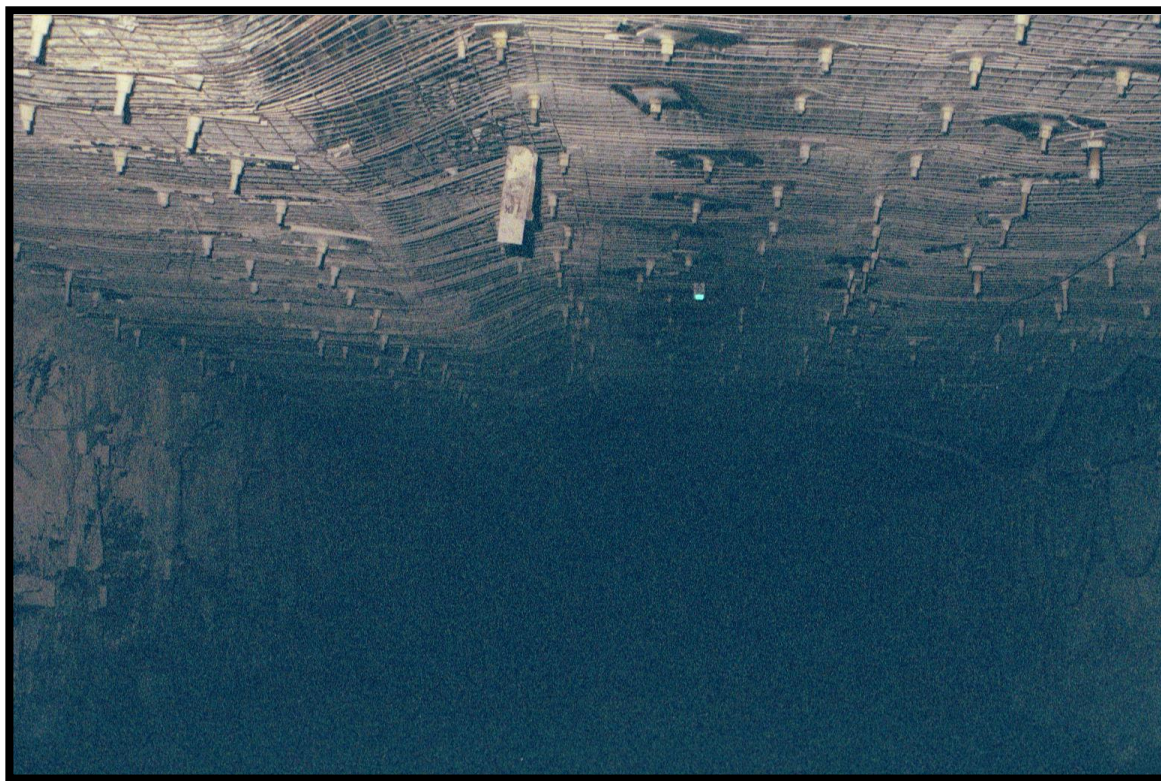


Fig 6 - Photograph of the LW406 Installation Roadway after widening. First Pass on RHS

In the installation face as the first pass progressed tell tales were installed offset from the centre of the roadway to be in the centre of the heading when widened out. These tell-tales were placed at 25m spacings. In 3 locations sonic extensometers were used to replace tell-tales with the aim of providing additional information of roof deformation. In all cases the total level of movement was less than 10 mm up to the time of longwall support placement i.e. when the tell-tales could no longer be measured. In general the roof was visually in a very good condition as Figure 6 shows. The placement of some megabolts was used around the intersections and around the tailgate intersection in particular, which did showed signs of deterioration.

CONCLUSIONS

Appin Colliery has a mixed history of using stress relief roadways. This recent method utilised a scientific approach with geological mapping, stress mapping and measurements together with monitoring of strata as well as computer modelling to assist in the development of the installation face and to ensure its success. The process worked well and is believed to be of both economic and operational benefits.

The success of this work has given the confidence to go ahead with stress relief roadways to assist in the protection of the remaining two longwall panels in this current mining area.

ACKNOWLEDGEMENTS

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