Proceedings of the 2011 Coal Operators' Conference

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FOREWORD

On behalf of the organising committee I welcome you to the 11th Underground Coal Operators’ Conference (Coal 2011). This particular conference is held in the aftermath of difficult times of mine disaster and flooding.

Last year the tenth anniversary of the underground coal operators’ conference was duly celebrated with various high quality presentations, high delegate participation, high sponsorship and exhibitions. It was the year that the world became to be accustomed to online availability of the past conference papers. Over the past 10 conferences a total of 332 papers were published in the proceedings, and almost all the papers were placed online http://ro.uow.edu.au/coal/.

Since then, from 130 countries there has been more than 85 000 downloads of papers. This is a reflection of the popularity of the conference. It is clear that the conference has gained standing worldwide, which is demonstrated by the number of overseas contributors. This conference will enrich the list online by another 51 papers.

Sincere thanks go to all the authors of the papers, in particular the authors who are regular contributors, not an easy task to achieve; to the participants who keep the conference going; and to our sponsors and exhibitors who contribute significantly towards keeping the conference alive and active.

It was agreed by the conference organisers and the Coal Operators’ National Conference Committee that the conference title be changed to Coal Operators’ Conference to embrace surface mining as well. This may add extra time pressure on the two day duration of the conference and this may lead to holding more concurrent sessions or to extending the conference to three days.

Special thanks to the conference organisers, paper reviewers, proceedings formatting individuals, the University of Wollongong printery and others who assisted in making this conference another successful event.

Naj Aziz
Conference chairman
# Table of Contents

Organising Committee ........................................................................................................ ii
Reviewers ............................................................................................................................... ii
Sponsors ................................................................................................................................. iii
Foreword ................................................................................................................................ iv

## Technical Papers

The Duncan Method of Partial Pillar Extraction at Tasman Mine  
**K. McTyer and T. Sutherland** .......................................................................................... 8

Hydraulic Sluiced Longwall Mining without Supports  
**I. Gray** ............................................................................................................................... 16

Practice and Prospect of Fully Mechanised Mining Technology for Thin Coal Seams under Complex Conditions in China  
**S. Tu, F. Wang, Y. Lu, Q. Wu, Q. Bai** ............................................................................... 22

Geotechnical Considerations for Longwall Top Coal Caving at Austar Coal Mine  
**A. Moodie and J. Anderson** ........................................................................................... 29

Understanding the Causes of Roof Control Problems on A Longwall Face from Shield Monitoring Data - a Case Study  
**R. Trueman, R. Thomas and D. Hoyer** ........................................................................... 40

Development of A Cavity Prediction Model for Longwall Mining  
**B. Wiklund, M. S Kizil and I. Canbulat** ......................................................................... 48

Application of the Brittle Failure Criterion to the Design of Roof Support in the Soft Rocks of Coal Mines  
**R. Seedsman** ..................................................................................................................... 60

Why Dead Load Suspension Design for Roadway Roof Support is Fundamentally Flawed within a Pro-active Strata Management System  
**R. Frith** ............................................................................................................................ 73

Calculation of Subsidence for Room and Pillar and Longwall Panels  
**A. Sroka, K. Tajdus and A. Preusse** ............................................................................... 83

Simulation of Development in Longwall Coal Mines  
**M. S. Kizil, A. McAllister and R. Pascoe** ...................................................................... 91

Stability Assessment and Support Design for Water Deviation Binary Tunnels of Bakhtiyari Dam-Iran  
**S. M. F. Hossaini, F. Nezhadshahmohamad and M. Dadkhah** .................................... 99

Geotechnical Appraisal of the Thar Open Cut Mining Project  
**R. N. Singh, A. G. Pathan, D. D. J. Reddish and A. S. Atkins** ...................................... 105

Prediction of Rock Mass Rating using Fuzzy Logic with Special Attention to Discontinuities and Ground Water Conditions  
**H. Jalalifar, S. Mojeddifar, A. A. Sahebi** ....................................................................... 115

External Reinforcement of Concrete Columns  
**M. N. S. Hadi** .................................................................................................................. 121
Experimental Study on Quantitative Application of Electromagnetic Radiation Excited by Coal-rock Fracture  
W. Chen, X. He, B. Nie and H. Mitri ................................................................. 129

Evaluation of Rock Support Performance through Instrumentation and Monitoring of Bolt Axial Load  
H. Mitri ............................................................................................................. 136

Improvement of Rock Bolt Profiles using Analytical and Numerical Methods  
C. Cao, J. Nemcik and N. Aziz ......................................................................... 141

Bearing Capacity of a Glass Fibre Reinforced Polymer Liner  
J. Nemcik, I. Porter, E. Baafi and J. Towns ..................................................... 148

Determining the Ultimate Strength of 'Tough Skin', a Glass Fibre Reinforced Polymer Liner  
J. Nemcik, I. Porter, E. Baafi and J. Navin ..................................................... 154

Improved Techniques for Heading Drivage  
S. F. Johnson .................................................................................................. 159

A Major Step Forward in Continuous Miner Automation  
D. C. Reid, J. C. Ralston, M. T. Dunn and C. O. Hargrave .............................. 165

Remote Tele-assistance System for Maintenance Operators in Mines  
L. Alem, F. Tecchia and W. Huang .................................................................. 171

An Integrated Approach to Improving Safety and Efficiency through Communications, Tagging and Collision Avoidance Systems  
B. Nicholls and T. Napier .................................................................................. 178

Digital Networks and Applications in Underground Coal Mines  
D. Kent .............................................................................................................. 181

Optically-powered Underground Coal Mine Communications  
G. Einicke, D. Hainsworth, L. Munday and T. Haight ...................................... 189

Intrinsically Safe Communication and Tracking Technologies for Underground Coal Mines  
L. Munday, S. Addinell, J. Thompson and E. Widzyk-Capehart ........................ 197

The Nexsys™ Real-time Risk Management and Decision Support System: Redefining the Future of Mine Safety  
K. Haustein, E. Widzyk-Capehart, P. Wang, D. Kirkwood and R. Prout ................. 205

Ventilation Surveys and Modelling - Execution and Suggested Outputs  
J. A. Rowland .......................................................... ........................................... 214

The Efficiency Study of the Push-pull Ventilation System in Underground Mine  
X. Zhang, Y. Zhang and J. C. Tien ..................................................................... 225

Dust Monitoring and Control Efficiency Measurement in Longwall Mining  
B. Plush, T. Ren, K. Cram and N. Aziz .............................................................. 231

Development of a Water-mist Based Venturi System for Dust Control from Maingate Chocks and BSL  
T. Ren, G. Cooper and S. Yarlagadda ............................................................... 239

Proactive Strategies for Prevention and Control of Fires in Bord and Pillar Mines Working in Thick Coal Seams  
S. Yarlagadda, R. Balusu, T. Liu and Veera Reddy B ........................................ 249

Design of Water Holding Bulkheads for Coal Mines  
V. S Mutton and A. M. Remennikov .................................................................. 257
Influence of Temperature on the Gas Content of Coal and Sorption Modelling
L. Zhang, N. Aziz, T. Ren and Z. Wang .......................................................... 269

The Experimental Study of the Impact on Supercritical CO₂ from CH₄ Composition in Coal
D. Wu, Y. Cheng .................................................................................................. 277

Gas Content and Emissions from Coal Mining
A. Saghafi ............................................................................................................ 285

Gas Content Measurement and its Relevance to Outbursting
I. Gray .................................................................................................................. 291

Stresses in Sedimentary Strata, including Coals, and the Effects of Fluid Withdrawal on Effective Stress and Permeability
I. Gray .................................................................................................................. 297

Actions to Improve Coal Seam Gas Drainage Performance
D. Black and N. Aziz ............................................................................................ 307

Review of Gas Emission Prediction and Control Methods for Multi-seam Mining in Chinese Coal Mines
Z. Wang, T. Ren and L. Zhang .............................................................................. 315

Effect of Magma Intrusion on Coal Gas Outburst Indexes in China
L. Wang, Y. Cheng, J. Jiang, S. Kong, H. Jiang, X. Zhang .................................. 326

Regional Gas Drainage Techniques and Applications in Chinese Coal Mines
Y. Cheng, H. Wang and L. Wang ........................................................................... 335

Pressure Relief Gas Extraction based on Strata Movement of Mined upper Protective Seam
H. Wang, Y. Cheng and Q. Zhai ............................................................................. 343

Evaluation of Outburst Potential at Sihe Coal Mine, China
M. Liu, H. Mitri, J. Wei, W. Xiao and Z. Wen ....................................................... 348

CO₂ Storage in Abandoned Coal Mines
P. Jallil, S. Saydam, Y. Cinar ............................................................................... 355

Numerical Simulation of the De-stressed Deformation of Mining Coal Strata at Different Pore Pressures
H. Liu, Y. Cheng, H. Chen, S. Kong and Q. Zhai ................................................. 361

Goaf Inertisation and Sealing Utilising Methane from In-Seat Gas Drainage System
C. Claassen ......................................................................................................... 369

Analysis of Ethane Emission Trends from Active Goaf Seals at Mandalong Mine
M. Leal, B. Beamish and C. Claassen ................................................................. 375

Experience with Using a Moist Coal Adiabatic Oven Testing Method for Spontaneous Combustion Assessment
B. Beamish and R. Beamish ............................................................................... 380

Development of a Web-based Underground Coal Mining Information Management System
I. Porter, E. Baafi and R. Stace ............................................................................. 385

Improving Access to Underground Coal Operators’ Conference Papers
M. Organ, N. Aziz and J. NemciK ....................................................................... 390

Index to Authors .............................................................................................. 408
THE DUNCAN METHOD OF PARTIAL PILLAR EXTRACTION AT TASMAN Mine

Kent McTyer1 and Tony Sutherland2

ABSTRACT: Mining commenced at Tasman Mine in late 2006. The current method of mining is bord and pillar using continuous miner-bolters and shuttle cars for first workings and secondary extraction using breaker line supports. The two stage process was chosen to accommodate irregular shaped coal deposits, allowing adjustments to be made to extraction ratios for better management of subsidence and to maximise the efficiency of the operation. Following the completion of the first three full/partial extraction panels a change in mining method was undertaken due to variable caving. The adopted partial extraction method involves stripping the developed square pillars on four-sides on retreat to leave a load-bearing remnant coal pillar. The system of partial extraction has been successful in delivering safety, productivity and subsidence targets.

INTRODUCTION

Tasman Mine is an underground bord and pillar mine owned and operated by the Donaldson Coal Pty Ltd located 20 km west of Newcastle, NSW (Figure 1). The mine commenced in 2006 and secondary extraction began in the Fassifern Seam in 2008. The 975,000 t per annum consent limit ROM coal is transported by road 20 km to Bloomfield Coal preparation plant for processing and subsequent rail transport to the port of Newcastle for export.

Tasman Mine employs the Duncan Method of partial pillar extraction. The mine commenced pillar extraction employing a full-extraction modified Wongawilli system in March 2008. After completion of the first panel a system of secondary extraction was deemed more suitable for the poor caving conditions. A partial extraction design was sought to provide the best balance between safety, productivity and subsidence constraints. The Duncan Method of partial pillar extraction commenced in October 2008. Three Pillar extraction panels have since being extracted successfully using this technique.

Figure 1 - Location of Tasman Mine with reference to Newcastle

RESOURCE

Tasman mines Fassifern Seam coal beneath the Sugarloaf Range State Conservation Area. The Sugarloaf Range is characterised by steep topography and cliffs. The Fassifern Seam outcrops on three sides of the mining lease (Figure 2). As a result, horizontal stress is low by NSW industry standards.

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Mains development occurs beneath the centre of the Sugarloaf Range at depths up to 250 m. Production panels are developed from the mains toward the outcrop to depths of cover of about 40 m. Production panels are typically less than 800 m long and experience significant change in vertical stress within each panel. The proximity and visibility of the cliff-lines of the Sugarloaf Range State Conservation Area to Newcastle result in strict mine approval conditions regarding subsidence outcomes.

![Figure 2](image)

**Figure 2 - North-south section across the Tasman Mine lease showing the Fassifern Seam outcrop around the Sugarloaf Range (H:V = 20:1). Fassifern Seam is shown in black**

The Fassifern Seam working thickness is typically 2.2 m to 2.5 m. The seam floor consists of 4 m of coal interbedded with claystone. The potential for swelling of the reactive clays has caused isolated areas of floor heave. The immediate seam roof is 0.5 m to 1.4 m of shale overlain by two distinct roof types either side of a NW-SE trending transition zone across the centre of the deposit. On the northeast side of the transition zone the shale is overlain by 1.5 m to 2 m of coal, overlain by 6 m to 10 m of sandy claystone (referred to as the Awaba Tuff). While on the southwest side of the transition zone the shale is overlain by 0.1 m to 0.4 m of coal, overlain by 10 m to 15 m of sandstone and conglomerate. The upper roof is characterised by regular massively-bedded units including a consistent unit of conglomerate greater than 20 m in thickness (Teralba Conglomerate).

**INITIAL PILLAR EXTRACTION EXPERIENCE**

Pillar extraction commenced at Tasman Mine in March 2008. The panel was developed for full extraction using a modified Wongawilli layout. Full-height caving did not eventuate. During extraction the coal tops were observed to fall but the Awaba Tuff unit was routinely seen to span distances greater than 50 m. Caving of the roof strata may have been hindered by the relatively low indicated horizontal stress and the bridging effect of the overlying conglomerate units (Figure 3). The influence of coal left behind as stooks and webs may also have had a role in delaying the initial roof caving event. Underground observations indicated increasing weight being thrown outbye as the panel neared completion.

**PARTIAL PILLAR EXTRACTION REVIEW**

The criteria for the new mining layout centred on the balance of safety, reserve recovery and productivity. Subsidence control and the rapidly changing depth of cover were incorporated into the design. A review of other Australian coal mines’ experience with partial extraction was undertaken. As Tasman operates continuous miner-bolters in development there was flexibility available in design of pillar dimensions and partial extraction layouts.

A design was developed to maximize the number of extraction tonnes for each metre of panel development. Focusing on the extraction tonne to development metre ratio was seen to be of benefit to
the mine by reducing the development metres for any given panel area whilst still maintaining respectable resource recovery. In addition, the timing of development and extraction would better suit the mine schedule. With subsidence limitations around the cliff-zones and other surface infrastructure the partial extraction design was required to have flexibility and surety when designing remnant coal pillars. A key safety goal was to avoid pillar splitting during secondary extraction.

Figure 3 - Photograph taken six months after extraction in Panel 2 South. The span is 16.5 m wide by 34.5 m long.

TASMAN PARTIAL PILLAR EXTRACTION DESIGN

Layout of pillar extraction panels was undertaken by referencing the relevant legislation and mine design guidelines. The general principles outlined in MDG 1005 Manual on Pillar Extraction in NSW Underground Coal Mines (NSW Dept. Primary Industries, 1992) were followed as the mine layout and sequence was developed.

Panels are typically five headings with a central belt road and flanking returns. Roadways are nominally 2.3 m high, driven on 45 m by 45 m pillar centres with a 5.5 m bord width. Panels are 300 m to 1200 m long and between 155 m (four-headings) and 252 m wide (six-headings). Secondary extraction is undertaken using a continuous miner modified for pillar extraction, three mobile breaker line supports, and shuttle cars. Pillars within the panel are stripped on four sides during panel retreat and the barrier pillar is also stripped (Figure 4). Lifting left and right is performed on both sides of the roadway from every roadway driven in the panel. The dimension of Stook X is as large as possible – typically greater than 6 m from rib-line - providing it can be totally removed at the start of a subsequent lifting sequence. At the completion of secondary extraction one stook remains for each pillar.

Overburden depth to seam can change by up to 200 m over the length of a secondary extraction panel. To account for the variation in overburden thickness the lift length is changed. Operators use a lift length of up to 10.75 m (measured 90° from rib-line) at shallow depth, and shorter lift length (as short as 7.3 m) at greater depths (Table 1). The length of lift into the barrier pillar remains the same as the lift length into the panel pillar (Figure 5). As a result, a wider barrier pillar is formed as depths increase. By modifying the lift length the remnant panel and barrier pillar dimensions are smaller at shallow depth and larger at greater depth. The decision to change second-workings dimensions on retreat was taken to avoid making changes to pillar dimensions during first-workings. Maintaining routine and regular development dimensions has significant benefits in terms of planning and sequencing. The benefits of a routine development far outweigh the additional controls required during secondary extraction.
Figure 4 - A typical five-heading modified Duncan Method partial pillar extraction layout used at Tasman Mine

Load-bearing remnant coal pillars and inter-panel barrier pillars are designed using the UNSW Pillar Design Procedure (Galvin, et al., 1998). Pillars are designed subject to full tributary area load. The positive influence of the overlying conglomerate is not taken into account when designing the panel pillars. Remnant coal pillar factor of safety within the panel is greater than 1.6. Width to height ratio of the remnant pillars range from 7.2 to 11. Referencing Hebblewhite et al. (2005) and Zipf, (1999) the width to height ratio of the pillars will result in strain-hardening characteristics and limit subsidence to tolerable levels in the unlikely event of pillar overload conditions developing.

As depth increases the percent coal recovery reduces in order to maintain the same pillar stability figures. Theoretical coal recovery ranges from 82% to 67% when stooks and webs are taken into account. The ratio of extraction tonnes to development metres reduces with greater depth and ranges from 51 t/m to 36 t/m. The total tonnes from development and secondary extraction as a ratio of development metres in the 2.4 m Fassifern Seam at Tasman ranges from 70 t/m to 55 t/m.
Table 1 - Duncan Method remnant coal pillar dimensions, Factor of Safety, Length of Lift, theoretical extraction percentage, and extraction tonnes per development metre at Tasman Mine

<table>
<thead>
<tr>
<th>Fassifern Seam Height</th>
<th>Remnant Pillar Dimensions (rib-rib)</th>
<th>Remnant Pillar Width to Height Ratio</th>
<th>Remnant Pillar Factor of Safety</th>
<th>Length of Lift (90° from rib-line)</th>
<th>Length of Lift (60° from centre-line)</th>
<th>Percent extraction (within panel)</th>
<th>Extraction tonnes per development metre</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cover Depth &lt;80m</td>
<td>18 m x 18 m</td>
<td>7.5</td>
<td>&gt;1.61</td>
<td>10.8 m</td>
<td>15.6 m</td>
<td>82%</td>
<td>51</td>
</tr>
<tr>
<td>Cover Depth &lt;120m</td>
<td>20.5 m x 20.5 m</td>
<td>8.5</td>
<td>&gt;1.63</td>
<td>9.5 m</td>
<td>14.2 m</td>
<td>77%</td>
<td>46</td>
</tr>
<tr>
<td>Cover Depth &lt;160m</td>
<td>22.5 m x 22.5 m</td>
<td>9.4</td>
<td>&gt;1.68</td>
<td>8.5 m</td>
<td>13.0 m</td>
<td>73%</td>
<td>42</td>
</tr>
<tr>
<td>Cover Depth &lt;200m</td>
<td>24 m x 24 m</td>
<td>10.0</td>
<td>&gt;1.68</td>
<td>7.8 m</td>
<td>12.1 m</td>
<td>70%</td>
<td>39</td>
</tr>
<tr>
<td>Cover Depth &lt;240m</td>
<td>25 m x 25 m</td>
<td>10.4</td>
<td>&gt;1.62</td>
<td>7.3 m</td>
<td>11.6 m</td>
<td>67%</td>
<td>36</td>
</tr>
</tbody>
</table>

TASMAN PARTIAL PILLAR EXTRACTION IMPLEMENTATION

Tasman Mine Management adopted the 10 hurdles approach to implement the new partial extraction system. The 10 hurdles is designed to assist in applying the requisite Standard of Care called upon by the NSW Coal Mine Health and Safety Act 2002 and NSW Coal Mine Health and Safety Regulation 2006 (Nichols, 2009) and which also aligns with the framework of AS/NZS 4360. The 10 hurdles are:

- Consultation;
- Hazard Identification and Hierarchy of Controls;
- Risk Assessment;
- Risk Management- Health Safety Management System (HSMS), Safe work procedures (SWP);
- Information;
- Instructions and Training;
- Supervision;
- Monitor System of work;
- Review the operation;
- Revise as necessary.

Consultation commenced in July 2008 with the Tasman workforce. A consultative meeting was held in August 2008 with the NSW Industry and Investment Inspectorate outlining the method and design. Workforce representatives were involved in a risk assessment that was finalised in August 2008. Clause 88 Approval of the bord and pillar system was received from the Inspectorate in September 2008. A selection of the Tasman workforce undertook an underground visit at Duncan Colliery in October 2008 to understand the sequencing and other nuances of the system. Subsidence Management Plan Approval was received and was shortly followed by crew training and commencement of secondary extraction in October 2008.

MINING OBSERVATIONS

Pillar extraction using the Duncan Method commenced at Tasman Mine in October 2008. Tasman has a workforce experienced with pillar extraction; however the Duncan Method was new to the operators and supervisors. During crew training the sequencing and importance of adherence to the design were emphasized as were the strict supervision requirements developed by the mine. Introduction of the system has resulted in significant improvements in safety, greater confidence in subsidence outcomes and productivity benefits.

SAFETY

No serious safety incidents have occurred to date at Tasman Mine since the introduction of the Duncan Method. Goaf edge load is minimal using this system as evidenced by rare low-level rib spall. Abutment load observations were backed up by low displacements measured on rib and roof extensometers. The development of abutment load at the goaf edge is slow due to the high capacity of the surrounding pillars.
and formation of load-bearing remnant pillars. Figure 6 illustrates the active extraction face zone is typically adjacent to coal pillars with a minimum pillar factor of safety of 2.2. Caving of the immediate roof does not present a serious windblast hazard due the small (less than 27 m wide) extraction voids formed. When the roof does cave it is typically the area from one intersection to the next intersection. Intersection to intersection is a relatively small area – approximately 50 m long by 27 m wide – and to date the impact of caving has been limited to minor interruption to ventilation.

![Figure 6 - Pillar Factor of Safety reduces from greater than 10 to 1.6 as the coal pillar is stripped on four sides](image)

For the given area of the panels there are a reduced number of intersections compared with other pillar extraction methods. As falls in the first or last lift adjacent to intersections are acknowledged to be the leading site of accidents during pillar extraction the reduction in the number of intersections is a significant safety improvement (Mark and Zelanko, 2001). In addition, stook X is far greater in size compared with other pillar extraction methods offering greater confidence in the support provided to intersections (Figure 7). Common to other partial extraction systems coal can be left behind for safety reasons with the only negative impact being reduced coal recovery. Tasman has found the secondary extraction of coal can be switched on or off for shutdowns or maintenance with rare and minor fender deterioration.

![Figure 7 - Looking across a four-way intersection at Stook X. The intersection has been extracted on three sides.](image)

The most significant improvements gained relate to the balance of panel development and secondary extraction. Development rates have recovered after a period of workforce acclimatisation and the time taken to complete development of a panel has reduced. Panels are more rapidly developed due largely to the reduction in panel metres. For a similar area of full-extraction the Duncan Method development metres are roughly 70% of the same area of full-extraction. Other dedicated partial extraction systems
Coal webs or stooks are rarely left behind to support the roof or stop goaf flushing. This may partly be the result of the low levels of vertical stress experienced at the goaf edge during extraction. The production figures from Panel 3 North suggest the estimated recovery following completion of the panel was within a few percent of the expected theoretical recovery. In areas without subsidence restrictions reduced coal recovery compared with full-extraction can be justified based on improvements to safety and productivity (Myors and Chastons, 2001).

**SUBSIDENCE**

The mine is subject to subsidence constraints in the form of natural features such as cliff-lines and steep-slopes and man-made features including a 330 kV power line, a fibre optic cable and three telecommunication towers. To date the expression of subsidence has been well below estimates. Tasman changes the remnant coal pillar dimensions to control subsidence beneath sensitive surface features. Sensitive features are assigned a Subsidence Control Zone (SCZ). SCZs are determined by the desired subsidence outcome and implemented by forming pillars of set dimensions with assigned pillar stability. Where the system has some advantages compared with other partial extraction systems is in the large width to height ratio of the coal pillars used to support the overburden. Where seam height reaches 2.5 m the w/h ratio exceeds 7.2. With the large width to height ratios the long-term confidence in the pillar stability is high. In addition, the immediate roof and floor material properties have less impact on pillar stability due to the large contact area of the pillar. Where the system must be used with caution is at shallow depth. To maintain elastic overburden behaviour at shallow depth the bord width must retain sub-critical dimensions.

Panel 3 North is the largest panel extracted to date. The maximum subsidence was measured across the widest part of the panel along the NE-SW survey line (Figure 8). Panel width is 252 m in the area with six headings. Recovery from within the panel is estimated to be close to 80% in the six-heading area beneath the NE-SW subsidence line. Maximum subsidence above Panel 3 North ranges from 51 mm to 101 mm, with maximum tilt of 1.2 mm/m (Figure 9). Back-analysis of Panel 3 North indicates pillars experience between 70% and 75% of the full-tributary area loading condition in the central portion of the panel.

**SUMMARY**

The Duncan Method of partial pillar extraction has been used successfully at two mines with generally similar geological conditions. Both Duncan and Tasman have an upper roof that can span large distances. In the case of Duncan the upper roof consists of dolerite, while at Tasman the upper roof consists of thick conglomerate and sandstone units. In both cases the immediate roof caves routinely if not readily. It is in these geological conditions that the Duncan Method has immense merit. Conversely,
site conditions where the control of surface or sub-surface subsidence is a critical mining constraint may also benefit from use of the Duncan Method.

Tasman Mine has taken an existing partial pillar extraction method and applied it to the site specific conditions and constraints. Since introduction targets have been met and the system continues to be refined and optimized to achieve objectives in the areas of safety, productivity and subsidence.

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HYDRAULIC SLUICED LONGWALL MINING WITHOUT SUPPORTS

Ian Gray

ABSTRACT: A concept for a mining method developed for a series of thin soft sloping seams between competent roof and floor strata is presented. The method proposed has the form of a retreating longwall where the mining method is by hydraulic sluicing. The face is essentially formed by drilling a directionally controlled borehole between gateroads. The directional drilling assembly is then removed from the drill-string and replaced by a jetting bit which is used to sluice the coal from between the borehole and the goaf. The process may then be repeated by drilling another hole parallel to the first and the process of mining operation is repeated.

BACKGROUND

Present longwall practice

Longwall mining is a capital intensive inflexible system which can yield very high production when things go right. It is generally suitable for operation in geologically uniform situations where the panel width can be maintained and where the panel length is adequate to justify the cost of moving the machinery between panels. It is limited in the range of height in which it can be used to those that permit miners to work safely. This means about 1.5 to 4.0 m, or higher if top level caving is used. Longwall mining has been used on slopes to about 25° but at these angles there is a constant battle to prevent the equipment walking down hill with the consequence that production is diminished.

Hydraulic mining

Hydraulic mining has been in existence for many years. One of the most notable examples of its use was Mitsui Sunagawa mine in Hokkaido, Japan in an approximately 6 m thick weak (3 MPa uniaxial compressive strength) seam at a dip of approximately 70°. There the coal was sluiced by the use of a monitor via adits driven in rock, which intersected the seam to form monitor bays and draw points. The water used for this operation was delivered to the 22 mm diameter monitor nozzle at pressures of 10 to 15 MPa and a flow rate of 3 m³/min, (Gray, 1980). This corresponds to a nozzle velocity of 132 m/s. This was a gassy mine was a gassy mine which had a very strict regime of gas drainage to avoid outbursts.

Following the closure of Sunagawa mine the pumps and monitor were transferred to the Strongman mine in New Zealand (Duncan, 1998). The use of the monitor was to be from a series of sub horizontal roadways driven in the sloping coal seam. A similar plan was prepared for the underground part of the Burton Mine in North Queensland. However this was never put into operation. The ill fated Pike River mine also planned to use hydraulic extraction (Whittall, 2006).

The fundamental limitation of all the hydraulic mining systems is the amount of roadway that must be driven to the volume of coal that is extracted. The reason for this is the limited effective range, approximately 30 m, of the water jet produced by the monitor. This limitation is brought about by dispersal of the jet and by limited visibility for the operator.

Directional drilling

Over the past 25 years, directionally controlled drilling has been used in underground mining so that holes of 600 m length can readily be drilled in seam. These can be extended to about 1500 m with more effort and a drop in drilling rate (Gray, 1994). The drilling system that has been almost universally adopted for this purpose comprises one that uses a poly crystalline cutter drag bit rotated by a down-hole motor. This bottom hole assembly has as part of it a bent sub which can be orientated so that the drill may drill in a preferred direction chosen by the operator. This orientation is achieved by rotating...
the drill string to a chosen tool face position under the real time guidance of an electronic survey system, which contains triaxial magnetometers and accelerometers. Thus it is possible to drill around rolls in the seam or to re-join the seam if it is disrupted by a fault. The angular build rate that may be achieved by such a system commonly lies in the range of 0.5 to 1.0° per metre. Rotation of the drill string may be used to make the bottom hole assembly drill a relatively straight hole.

The drill pipe used for this process has, in the Australian context, been almost universally NQ or more recently NRQ type manufactured by Boart Longyear. This high tensile steel pipe (620 MPa yield strength) with heat treated tool joints has an outside diameter of 69.9 mm (2 3/4") and an inside diameter of 60.3 mm (2 3/8"). It has been generally used to drill boreholes using a 96 mm drill bit. This drill pipe has been found to be particularly useful because it only weighs 23 kg per 3 m length and is readily handled by underground drill operators.

Water jet drilling

Over many years and several research projects water jet drilling in coals has been shown to be feasible. Some of the most important work in this area was conducted by Kennerley (1990) and others that followed at the University of Queensland culminating in the tight radius drilling system that permits drilling at the end of a hydraulic hose. The drilling pressures used have varied between about 50 and 120 MPa with flow rates of approximately 0.2 m³/min. Also it has been shown that it is possible to cut slots with the water jet using lateral jetting nozzles at the drill bit.

There is a significant difference in the flow rates and pressures used between water jet drilling and hydraulic sluice mining. One is a cutting process which uses high pressure water while the other is more dependent on mass flow and momentum to deliver a force to the coal while pressurising cleats to assist the disintegration of the coal. The former would appear to rely more on the effects of cavitation to cut the coal.

THE PROPOSED MINING METHOD

The concept

The mining method proposed uses a combination of the technologies described above. It works as a form of longwall between gate roads which are driven on the across dip with sufficient gradient to allow flow coal and water and have a ditch formed in them to carry the slurry produced by mining. The difference is that the face does not have the conventional powered hydraulic supports nor a shearer and armoured conveyor. Rather the face is formed by drilling from one gateroad across the panel to the next gateroad. This drilling would be accomplished by using directional drilling techniques to cover a panel that might be between 100 and 500 m in width. Once the drill has reached the far gateroad the directional drilling assembly is removed and replaced by a lateral jetting assembly. Water is then pumped through the drill string and out of the lateral jet to erode the coal which flows under gravity back to the lower gateroad. The drill string is withdrawn during this process as the coal is eroded. Once the string is fully withdrawn it is moved with the drill down dip in the gateroad and the process is repeated. It is envisaged that adjacent boreholes would be placed 5 to 25 m apart, depending on local conditions. The erosion is accomplished either completely between the boreholes or leaving some form of narrow pillar of material so as to control roof failure.

Drilling may be accomplished from the upper gateroad to the lower one with erosion taking place on withdrawal of the drill string uphill. In this case the slurry formed by mining would flow down through what could potentially be goaf. By angling the face line by choice of drilling direction a face may be maintained so that the slurry will flow down the intersection of the retreating face and the floor. Alternatively a narrow pillar can be left so that the slurry flows down an enlarged, eroded hole that does not collapse immediately.

Alternatively the drilling may be undertaken from the lower gateroad to the upper one to ease the return of cuttings. In this case the hole would have to be of reasonably large diameter (0.3 m) and may need to be enlarged by reaming to this size. Such reaming could be undertaken by either the use of water jets or a mechanical rotary reamer. The jetting process for mining would then be undertaken. This would involve jetting from the top down thus eroding coal that flows back down the enlarged borehole. Once the slurry has reached the lower gateroad it can be in the ditch or flume to a sump for pumping to surface as slurry or for local separation into wetter and drier components which can then be transported
to surface. The version of the operation that uses drilling from the upper gate road is shown Figures 1 and 2.

Figure 1 shows a panel in a view that is perpendicular to the seam being mined. The coal to be mined is to the right between the face and the main headings. The goaf is to the left of the face. The face lies between the upper (top of drawing) and lower gateroads. Drilling takes place from a drill site of the lower of the upper gateroads. Drilling is accomplished to the upper of the lower gateroads. The directional drilling assembly is replaced by a lateral jetting nozzle and mining is accomplished by jetting to form jet cut mining face (3-3) and flow is down the advancing face edge to the gateroad where it flows in a ditch to the sump at the bottom right hand corner of the figure. The previously mined panel lies to the top of the figure. An un-eroded pillar of coal is left below the upper gateroad to protect the drill and crew.

**Figure 1 - View of hydraulic mining longwall panel perpendicular to the seam**

Figure 2 shows an almost complete section (2-2) along the face of Figure 1. The goaf of the previously mined panel is shown in the upper right hand corner above the upper of the upper gateroads. The drill is in the lower of the upper gateroads. It has been used to drill to the upper of the lower gateroads where it will be fitted with a lateral jetting bit. The drill string has then been withdrawn while jetting the section marked as the eroded zone. The slurry produced from this has flowed to the ditch in the lower gateroad for transport to the sump.

**Figure 2 - Section of hydraulic mining system roughly along section 2-2 from Figure 1**

Figure 3 shows section 3-3 from Figure 1. It shows the jetting bit which is protruding from the borehole with a lateral jet which cuts a mining face roughly perpendicular to the advancing face. A goaf is shown being formed to the left of the drawing. The slurry runs down the intersection of the floor and advancing face (out of the page) toward the lower gate road.
Drilling

Rather than drill with the standard NQ or NRQ drill pipe currently used in directional drilling operations it is proposed to move to a large diameter pipe. This size increase is to allow the pipe to have a larger fluid flow rate capacity, so that it will be more robust to withstand greater torque to turn it. Such a pipe is readily sourced by simply increasing by one size to HRQ drill pipe also manufactured by Boart Longyear. The outside diameter of this is 88.9 mm (3 ½") with an inside diameter of 77.8 mm. This is comparatively light at 35 kg per 3 m length or could be reduced to 23 kg per 2 m length. The drill hole would be drilled using a 3 ½" down-hole motor to about 150 mm. The survey system would be similar to that used currently for directional drilling. Heavier drill pipe could be used for greater robustness but the manual handling of such pipe would be a burden. Standard oilfield tool joints will normally be externally upset. Such upset tool joints would pose a problem for extraction if the hole collapsed. Flush external, internally upset oilfield type tool joints would lead to excessive pressure loss at each tool joint for the flows being considered.

Sluicing hydraulics

The HRQ drill pipe proposed has been tested and found satisfactory to 35 MPa pressure using some specialist tool joint seals. The volumetric capacity over a length of 300 m of such drill pipe is 2.9 m³/min for a 2.5 MPa pressure drop within the pipe. Such a pressure drop is quite acceptable and delivers a flow very similar to that used at Sunagawa. To deliver 12 MPa at the nozzle will require a total of about 14.5 MPa to be available at the top of the drill string. This pressure and volumetric flow can be delivered by larger oilfield triplex mud pumps that are available [13.8 MPa (2000 psi), 750 USGPM] or possibly more readily from pairs of smaller mud pumps. Such flows could be readily reticulated around a mine using 200 mm pipes with little pressure loss.

The nozzle could possibly be better designed than those used at Sunagawa which had a very high pressure loss coefficient. There is a desire to manufacture the jetting bit in a slick form so that it does not become trapped in the event of hole collapse. The jet proposed would be formed out of a gradual taper and bend housed within the same diameter section as the drill pipe. This can be achieved using spark erosion manufacturing techniques in very hard material to resist wear.

The flow of the slurry along the advancing face of the goaf edge or the hole which is being eroded will be one of the controlling factors in the success of the mining method. The flow here will depend primarily on the slope available for the fluid to run down and whether the goaf collapses right up to the retreating face edge. In the latter case the process of leaving an un-eroded pillar of coal between the current mining zone and the goaf may be adopted to temporarily hold the area open. The prime requirement is to be able to break the coal into sufficiently small pieces that it is carried down slope in a turbulent flow.
Experience would suggest that the slope must be higher than 5° for this to occur. Slopes closer to 10° would provide better security against a blockage.

The movement of the slurry in the ditches of the gateroads is essentially a problem of channel flow. Fortunately coal has a comparatively low density and is therefore more easily transported in water than denser rock which may need periodic manual removal. If required the use of a simple lining of the ditch using sheets of polyethylene will reduce surface roughness and erosion thus speeding flow and reducing maintenance requirements.

**Strata control**

One of the keys to the successful implementation of the system is that of strata control. The roof must stay up for a while before the goaf forms so that flow from the mining face can take place and the floor not be excessively eroded. The general requirement is therefore for a soft easily erodible coal seam between relatively strong roof and floor. This is a requirement that is met in some geological settings.

**Ventilation**

The need to ventilate the face poses significant limitations on the system. If the goaf collapses right up to the face then the ventilation along the face will be lost. It is therefore necessary to be able to arrange the ventilation so that it can be maintained along the top and bottom gate roads if this situation occurs. This will permit the drilling and starting of another face adjacent to that is being lost. The question of how the water jet mining face is regarded from a ventilation viewpoint needs to be considered. Is it a face that must be kept ventilated at all times, or is it a part of the goaf where there will be always at some location an explosive mixture of gas and air? The latter means that the mining method is simpler to use. The former would require the formation of the goaf proper to be delayed by leaving narrow pillars behind that maintain an opening for ventilation. The actual face where the water jet would be applied would not however be ventilated. This is not an issue if it is regarded as a wet borehole with no risk of ignition, nor if the coal is not gassy.

**Jet control**

The water jet control technology need not be initially in excess of anything that is used for current directional drilling. All that is required is an indication of radial position, or tool face angle in drilling terms, of the water jet. The operator can then play this jet in the required direction for a suitable period, or until coal ceases to be produced, and then move the drill string longitudinally to commence the sluicing of the next section of coal. If a hard roof and floor exist then they will limit the extent of erosion. Refinements could be made by the introduction of improved monitoring systems to determine whether the coal had been sluiced away. Such systems might use visual imaging, acoustic imaging or variants involving listening to transmitted sound of the jet hitting the side of the opening. These are not however considered to be necessary for the system to be trialled.

**CONCLUSIONS**

The system proposed is a simple low cost method that is designed to permit the establishment of a form of continuous panel mining, which will achieve significant extraction ratios with respectable production for an order of magnitude or less cost than a conventional longwall. It has the requirement that the seam to be mined slopes adequately for hydraulic mining practises to be used. However it has the advantage that it can be used around faults as there is no restriction that requires the panel width to remain constant. Neither is there a concern with a fault transecting a panel. The panel will simply be drilled and sluiced up to the fault and then recommenced on its other side. There is no need to move a major face line incorporating powered hydraulic supports, conveyor and shearer. It is also suited to seams that are too narrow to be mined by conventional longwalls.

The possibility exists that the face may collapse but this is not a disaster as it can be readily re-established by drilling a new hole. Even the loss of all the mining equipment that exists along the face occurred it is unlikely to cost more than $0.5 million. Compare this with the cost of an iron bound longwall!

The system is simple and could be applied using existing technology now. All that is needed is a seam of greater than 10° slope and preferably with a moderately strong roof and floor. For a trial the seam would
preferably not be gassy but the potential exists within the system to re-enter gas drainage holes drilled within the seam and use them as the basis for the mining method.

Patents have been applied for to cover the methods described.

REFERENCES


PRACTICE AND PROSPECT OF FULLY MECHANISED MINING TECHNOLOGY FOR THIN COAL SEAMS UNDER COMPLEX CONDITIONS IN CHINA

Shihao Tu¹, Fangtian Wang¹,², Yan Lu¹, Qi Wu¹, Qingsheng Bai¹

ABSTRACT: In China, thin coal seam are rich in resources and complex in conditions, however, the characteristics such as narrow mining space, the low level of mechanised technology, bad working environment and the high cost of mining, directly restrict the development of mining safety and high-efficiency. In thin coal seams with hard gangue which contains concretions of pyrite, LS-DYNA is applied to calculate the rational blasting parameters and carry out the deep-hole pre-splitting blasting technology, the hard gangue is fractured effectively, hence advancing the productivity of thin coal seam mining. In addition, the mining rate is speeded up in thin protective layers in extreme close coal seams by enhancing the level of fully mechanised equipment and other effective measures. Safety and high-efficiency mining can be realised in the outburst coal seam. Thin coal seam mining technology faces many problems presently, i.e. the low level of equipment automation, the low advance rate of mixed coal-rock drift, and the big intensity of worker labour. By lowering the labour intensity, improving the efficiency by means of advancing mining automatic equipment and other measures, respectively, thus manless working faces can be successfully realised in thin coal seam mining.

INTRODUCTION

Thin coal seams, traditionally identified as less than 1.3 m of thickness, are rich in resources and widely ranging in China. The mineable reserve is 6 150 Mt of thin coal, accounting for 20.4% of the total coal resources (Chen, 2007). Simultaneously, the mineable reserve of 2 529 Mt and accounts for more than 85% of Chinese Key Coal Mines. Particularly, the thin coal mineable reserve is 1 800 Mt in the provinces and cities include Shanxi, Hebei, Sichuan, Inner Mongolia, Guizhou and Chongqing, furthermore, the mining bureaus of Xuzhou, Datong, Kailuan, and Pingdingshan, have 501 Mt of thin coal mineable reserves, which takes up 20.9% of these bureaus’ total reserves (Liu and Liu, 2002; Yan, 2004). The mineable reserves of thin coal seam in provinces are listed in Table 1.

Table 1 - Statistics of thin coal seam mineable reserves in districts in China (Liu and Liu, 2002)

<table>
<thead>
<tr>
<th>District</th>
<th>Hebei</th>
<th>Shanxi</th>
<th>Inner Mongolia</th>
<th>Liaoning</th>
<th>Jilin</th>
<th>Helongjiang</th>
<th>Guizhou</th>
<th>Henan</th>
<th>Sichuan</th>
</tr>
</thead>
<tbody>
<tr>
<td>Reserves /Mt</td>
<td>327</td>
<td>1380</td>
<td>197</td>
<td>198</td>
<td>65</td>
<td>44</td>
<td>464</td>
<td>524</td>
<td>1480</td>
</tr>
<tr>
<td>Proportion %</td>
<td>16.8</td>
<td>17.6</td>
<td>15.1</td>
<td>12.9</td>
<td>18.3</td>
<td>1.35</td>
<td>37.2</td>
<td>12.3</td>
<td>51.8</td>
</tr>
</tbody>
</table>

Owing to the influence of narrow mining space, bad working environment, high drivage ratio, high labour intensity, low level of mechanised mining equipment, low safety factor, low work efficiency and high input-output ratio, the quantity of thin coal seams mined only accounts for 10.4% of the total product annually. The proportion between output and mineable reserves is serious out of balance in China, which shortens the mine service-life and seriously wastes the coal resources. The fundamental way of realizing safety and high-efficiency mining in thin coal seams is to employ advanced automatic mining equipment. Automatic operation and control request use of advanced hydraulic supports, mining machines and conveyors, which are adequate for the geological conditions of thin coal seams. Presently, the fully mechanised mining technology has improved by leaps and bounds, and has formed three main mining models as follows (Wang, 2009): shearer with matching hydraulic supports; plough with matching hydraulic supports and auger mining methods. As a result of the increasing mining intensity in the eastern coal mines and the aged coal mines in China, the mineable reserves of medium thick and thick coal seams have decreased year by year, and hence many key coal mines are facing the

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task of thin coal seam mining (Sheng, et al., 2007; Ma, et al., 2007; Zhao and Ma, 2009). Consequently, thin coal seam mining becomes an important task and has a promising future in China.

DEVELOPMENT STATUS OF FULLY MECHANISED MINING IN THIN COAL SEAMS

Development status overseas

Coal seams with thickness less than 2.0 m are named as thin seams in USA and other western countries, and the international definition is used for coal seams with thickness of 0.8-2.0 m. At present, longwall mining overseas for thin seams has two primary technological methods (Zhai, et al., 2009): one is the fully mechanised mining method equipped with shearer, hydraulic support and scraper conveyor, the other is the fully mechanised mining method using plough, hydraulic support and scraper conveyor.

Longwall mining with shearer and room-and-pillar mining with continuous miners have been applied in thin coal seams in the USA and UK; manless automatic mining has been carried out in Germany with plough and hydraulic supports by electro-hydraulic control. From 2004 to 2005, the USA had 52 longwall faces, of which 21 coal mines had thin seams with thickness less than 2.0 m, 20 coal faces with shearer and only one with plough. The average work efficiency was 35.3 t/d per miner, and the largest and average output of coal faces with shearer were 9.3 Mt/a and 4.5 Mt/a, respectively. The output with plough was 1.6 Mt/a and the work efficiency was 18.7 t/d per miner (Chen, 2007; Bi, 2007). From the conditions of their services, both shearer and plough have achieved great economic results, however, each mining machine has certain advantages and disadvantages in adapting to different geological conditions.

Development status in China

The mining technology of thin coal seam has experienced five development stages in China as follows:

- During the fifties of the 20th century with blasting method;
- In sixties using deep coal cutter to blasting coal;
- During the period of 1970-1980, the machine unit had a big development, the thin coal seam shearer named BM-100 was manufactured in 1974; various ploughs were developed later;
- From 1990 to 2000, efficiency was increased by means of imported foreign advanced fully mechanised mining equipment employed in thin and extremely thin coal seam with complex geological conditions, and
- From 2000 with the shearer of high-power and highly reliable homemade mining equipment, the level of mechanization conspicuously improved.

In 2003, Jinhuagong Coal Mine of Datong Coal Mine Group employed a domestic MG200/450-WD shearer for thin coal seam mining, the output of the coal face was 6766 t/d, and annual output was 1.0 Mt, which created a new record for similar geological conditions (Chen, 2007). In 2006, based on the geological conditions of thin seam with dip angle 3-6°, average thickness 1.3 m and Protodyakovon coefficient f=1.6, Binhu Coal Mine in Zaozhuang City employed the following coordinating equipment: MG340-BWD1 shearer, ZY2400/0.9/2.0 hydraulic support and SGZ-730/320 scraper conveyor. The average yield of coal face was 80 000 t per month and the maximum yield was 3504 t/d. Under the coal seam conditions of average thickness 1.3 m, both Daizhuang Coal Mine in Zibo City and Tongjialiang Coal Mine in Datong City achieved average yields of 68 000-90 000 t per month by Chinese shearer of MG series, the two legged hydraulic shield support and associated equipment (Zhang, et al., 2002; Zhai, et al., 2009). According to the geological conditions of thin coal seam with average thickness 1.3 m and dip angle 5-8°, Tiefa Coal Mine Group Xiaqiong Coal Mine applied W1E-703 plough in the coal face with strike length of 905 m, through the automation, it had achieved prof of 28.5 RMB/t, produced 0.6 Mt in nine months, and created immediate economic benefit of 17.2 million RMB (Liu and Liu, 2002; Yan, 2004). The mines in Jixi Coal Mine Group with 77% of mineable reserves in thin coal seam had an annual output of 1.85 Mt of which 17% come from thin coal seam in 2006. Pinggang Coal Mine with an average thickness of 1.3 m and dip angle of 33-35°, where MG132/310-BW shearer, BY200-06/15 shield hydraulic supports and SGW-15C scraper conveyor were employed and efficient measures were used to prevent the slipping and falling of equipment, produced a yield of 50 000 t per month.
FIELD APPLICATIONS IN THIN COAL SEAMS WITH COMPLEX CONDITIONS

Deep-hole pre-splitting blasting in thin coal seams with concretions of pyrite gangue

Thin coal seams with hard gangue accounts for 52.8% of the total thin coal seam mineable reserves in Shandong Province, and the hard gangue has severely restricted the high performances of fully mechanised mining equipment. As an example, Geting Coal Mine of Zibo Coal Mine Group has abundant thin coal seam resources, the average thickness of the NO.16 coal seam is 1.3 m, and there is a hard gangue layer in the coal seam. According to the geological conditions, a high-power shearer, matching powerful drum and enhanced point-attach picks were employed for the thin coal seam mining, which was supported by Kennametal Co., Ltd. Table 2 shows the coordinative equipment.

Table 2 - Scheme of coordination equipment in Getting Coal Mine

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Type</th>
<th>Number</th>
<th>Main technical specifications</th>
</tr>
</thead>
<tbody>
<tr>
<td>Double-drum shearer</td>
<td>MG200/456-QWD</td>
<td>1</td>
<td>Cutting power 2×200 kW, Mining height 1.15~2.20 m</td>
</tr>
<tr>
<td>Hydraulic support</td>
<td>ZY2800/09/18</td>
<td>22</td>
<td>Effective resistance 2800 kN, Support height 0.9~1.8 m</td>
</tr>
<tr>
<td>Scraper conveyer</td>
<td>SGZ730/2×132</td>
<td>1</td>
<td>Installed power 264 kW, Delivery capacity 600 t/h</td>
</tr>
<tr>
<td>Bridge conveyer</td>
<td>SZZ630/75</td>
<td>1</td>
<td>Installed power 75 kW, Delivery capacity 600 t/h</td>
</tr>
<tr>
<td>Belt conveyer</td>
<td>SSJ1000/132</td>
<td>1</td>
<td>Installed power 132 kW, Delivery capacity 630 t/h</td>
</tr>
</tbody>
</table>

The field trial demonstrated that many shearer picks were damaged while cutting hard gangue, and this slowed the advance rate. To ease the situation, a seismic method of exploration was adopted to explore the distribution regularity of gangue in the thin coal seam, and the partial graph of 100 m length is shown in Figure 1. Hard gangue is mainly distributed below the roof around 0.34 m, the height is 0~0.4 m, and the height between 0.2 to 0.4 m accounts for 29%, the Protodyakonov coefficient of pyrite is \( f = 11 \). Pyrite presents as stratiform and nonuniform distribution, concretions of pyrite are similar to cobblestone, and most of them distributed in the gangue.

Figure 1 - Distribution regularity of hard gangue in thin coal seam

In order to reduce equipment consumption, prolong the service life of equipment and speed up the advance rate, deep hole pre-splitting blasting along gateway was obliged to be carried out in the area, where thick hard gangue were distributed. Based on the geological conditions, theoretical calculation and numerical simulation were used to provide rational blasting parameters.

According to the theory of rock blasting mechanism and the rule of stress wave produced and weakened in the rock blasting process (Song, 1989), the radius of crack zone \( R_p \) can be calculated by the formula:

\[
R_p = \left[ \frac{\nu P S_t}{\rho_0 D_t^2} \right]^{1/2} r_b = 2.54
\]

Where, \( P \) is the initial radial stress peak of stress wave, \( P = \rho_0 D_t^2 \left( \frac{r_c}{r_b} \right) n/8 \); \( a \) is the attenuation value of stress wave, \( a = (2-\nu)/(1-\nu) \); \( D_t \) is detonation velocity, 3650 m/s; \( \rho_0 \) is explosive intensity, 1050 kg/m\(^3\); \( r_c \) is radius of cartridge bag, 0.025 m; \( r_b \) is radius of blast hole, 0.045 m; \( S_t \) is the tensile strength of rock.
mass, 3.6 MPa; $v$ is Poisson’s ratio, 0.2; $n$ is the stress intensification factor, 10. The calculation is $R_p=2.54$ m, which provide a theoretical reference for the later test.

Blasting effectiveness and the stress wave development process can be analysed and simulated by the software of LS-DYNA (LSTC, 2003), Figure 2 indicates the regulation of stress distribution.

![Figure 2 - Regulation of stress distribution for blasting hard gangue](image)

(a) X-Stress distribution (b) Von Mises distribution

The numerical model has analysed the process of smashing rock near the explosion source by shock wave and stress wave. As shown in Figure 2(a), the X-Stress wave velocity in coal is faster than in the rock mass, the stress wave distribution forms a flat “O” shape; it contributes to the pre-splitting blasting of the hard gangue and lessens damage to the roof. As shown in Figure 2(b), the radius of the crack zone $d$ is around 5 m, The Von Mises wave spreads farther along hard gangue than intruding into the roof and floor, which is more beneficial to the mining, and the result agrees with the previous theoretical calculations. The results of both the theoretical calculation and numerical simulation provide the rational blasting parameters for the field experiment.

The field investigations show that the safety and high-efficiency mining in thin coal seam with hard gangue has been achieved by the pre-splitting blasting technology. Blasting and coal mining operated separately, which improves the efficiency of fully mechanised mining equipment. As a result, the operation ratio of shearer is more than 71.4%, and the output of coal face is 11 65.2 t/d, the working environment has an apparent improvement, and economical profit of 29.4 million RMB has been achieved annually.

Safety and high-efficiency mining in the protective coal seam

At present, gas explosions cause the highest proportion in coal mine accidents in China. There were 182 coal mine explosion accidents in 2008, with a death toll 778, so preventing gas accidents shoulders heavy responsibilities (Fang, et al., 2009). Protective coal seam mining is the most effective and economical regional gas control technology (Liu, et al., 2009). The mineable reserve of thin coal seams is more than 102.7 Mt in Huaibei Coal Mine Group. To protect the subjacent outburst coal seam, the overlying 71 thin coal seam was mined in Qinan Coal Mine. This is the first study of mining in thin protective coal seam close to an outburst coal seam in China and abroad.

According to the geological conditions of 71 coal seam with average thickness 1.3 m and Protodyakonov coefficient $f=0.2-0.3$, coordinative equipment was designed and improved for the protective layer mining, the field experiment demonstrated that choosing and matching of coordinative equipment facilitated the realization of rapid advance in the thin protective seam. Table 3 lists the coordinative equipment applied in Qinan Coal Mine.

UDEC (Universal Distinct Element Code) is a two dimensional, discrete element numerical calculation program appropriate for non-continuum modelling. According to the geological conditions of thin protective coal seam, UDEC is employed to analyse the law of movement and fracture development in the 71 coal seam floor. Figure 3 shows the distribution of vertical stress and plastic zones.

Figure 3 (a) shows the distribution of vertical stress and plastic zones developing to the 72 coal seam while the excavation length of 75 m in 71 coal seam, and the overlying strata has fractured and sunk, the stress-relaxed area of the protected coal seam increases under the goaf. Meanwhile, the ground stress
decreases and the seam permeability increases, then the effect of pressure-relief, increasing permeability and fluidity appears in the protected coal seam. When the working face advanced 90 m (Figure 3 (b)), the stress-relaxed area in the protected coal seam increases still further, the degree of pressure relief reaches its peak under the goaf. The range and degree of stress-relaxation increases further especially in the advanced direction and the seam permeability increases. The protective coal seam occurs stress-relaxation and volumetric expansion under the goaf, and emerges plenty of faulted joints and interlamination cracks, which promotes the efficiency of gas drainage and avoidable of the outburst risk.

Table 3 - Scheme of coordination equipment in Qinan Coal Mine

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Type</th>
<th>Number</th>
<th>Main technical specifications</th>
</tr>
</thead>
<tbody>
<tr>
<td>Double-drum shearer</td>
<td>MG200/456-WD</td>
<td>1</td>
<td>Cutting power 2×200 kW, Mining height 1.25-2.50 m</td>
</tr>
<tr>
<td>Hydraulic support</td>
<td>ZY4000/09/20</td>
<td>83</td>
<td>Effective resistance 4000 kN, Support height 0.9-2.0 m</td>
</tr>
<tr>
<td>Scraper conveyor</td>
<td>SGZ-730/400</td>
<td>1</td>
<td>Installed power 400 kW, Delivery capacity 700 t/h</td>
</tr>
<tr>
<td>Bridge conveyor</td>
<td>SZZ-730/132</td>
<td>1</td>
<td>Installed power 132 kW, Delivery capacity 730 t/h</td>
</tr>
<tr>
<td>Belt conveyor</td>
<td>SDJ-150</td>
<td>1</td>
<td>Installed power 150 kW, Delivery capacity 630 t/h</td>
</tr>
<tr>
<td>Lotion pump</td>
<td>BRW-315/31.5</td>
<td>2</td>
<td>Nominal flow rate 315L/min, Nominal pressure 31.5MPa</td>
</tr>
</tbody>
</table>

(a) Excavation length of 75 m   (b) Excavation length of 90 m

Figure 3 - The distribution of vertical stress and plastic zones

The average thickness of the 72 coal seam is 3.0 m, and the distance between the 72 and 71 coal seams is only 5.0 m, so it is an extremely close coal seam group. Coal samples were tested in the lab and the methane content is 12.3 m³/t, which have the danger of outburst. The stress-relaxed gas of 72 coal seam easily swarm into the goaf and working face of the 71 coal seam through rock mass fracture, effective measures: speeding up the advance rate, draining the gas accumulated in the upper corner and goaf, enhancing the ventilation management and monitoring gas in real-time to avoid gas exceeding the limit. The field measurements indicate that the maximum gas concentration is 0.3% in the upper corner, and the working face advanced 16 cuts/d, the maximum monthly output of the coal face is 64,600 t. Safety and high-efficiency mining has been achieved.

EXISTING PROBLEMS AND PROSPECT FORECASTS

Main existing problems

At present, thin coal seam mining technology faces many problems, including:

- The level of equipment automation remains to be improved. A contradiction between installed power, machine height and delivery coal space is still the main technological problem of developing a high-power shearer in thin coal seams. Automatic control and fault diagnosis function for the shearer, and adaptability of hydraulic support to adjust to thin coal seam with complex geological conditions are required to improve rapidly. Meanwhile, the level of self-propelled control between hydraulic supports and conveyor remains to be developed.

- Advance rate of mixed coal-rock drifts needs to be accelerated. In general, coal seam thickness is less than 1.3 m while the roadways are higher than 2.5 m, and hence half of the drift section is rock mass. The speedy drivage of mixed coal-rock drifts is a challenge, which directly influences
the mining-drifting balance. Therefore, the development of a tunnelling and bolting integrated machine in the mixed coal-rock drift is a key to realizing efficient operation and speedy drivage.

- Working environment and less labour intensive effort is needed. The characteristics include narrow mining space, low level of mechanised mining technology and high cost of mining, which cause an intensity of worker labour and bad working environment. Particularly, the labour intensity is still greater in thin coal seams with complex geological conditions like hard gangue, faults and folds, large angles and great undulations.

Prospect forecasts

The development tendencies of thin coal seam mining technology are as follows:

- Evaluating and planning various geological conditions is an important foundational work in thin coal seam mining. According to factors such as coal seam thickness, seam inclination, structure, Protodyakonov coefficient, roof and floor conditions, faults and folds, gas outburst and water inrush, system analysis is needed to study the influence of factors and bring out a rational exploitation program.

- Improving the level of equipment automation and choosing and matching coordinative equipment reasonably are the trend of mining development. According to the evaluation system of geological conditions, there is a rapid change in designing the suitable mining technology and choosing and matching rational coordinative equipment. It is important to manufacture the electric traction shearer to adapt to geological conditions, develop reliability and cutting efficiency, and reduce the dust in the working space in China coal mines. Meanwhile, many advanced automatic machines and techniques have already emerged, e.g. automatic control and fault diagnosis systems, automatic adjustment of cutting height by means of automatic identification in coal and rock, self-control pulling speed through interchange frequency conversion electric traction, self-propelled hydraulic support and conveyor by means of infrared and electro-hydraulic servo valves.

- Developing the technique and equipment to speed drivage has a large market for drifting excavation of thin coal seams. Studying the rational technique and equipment of tunnelling efficiently and transporting in the underground environment quickly, loading and delivering of coal and waste rock separately, so as to realise speedy drivage in mixed coal-rock drifts.

- Realising manless working face in thin coal seams is the representation of scientific mining. According to the automatic monitoring technique, China has already experienced the technique of long distance bidirectional communication, telemeter and telecontrol between dispatch room and centralized control station in roadway and shearer, hydraulic supports, conveyor, etc. which has lessened the labour intensity, improved the safety conditions, and increased the production efficiency and contributed to realise manless working faces in thin coal seam mining.

CONCLUSIONS

Based on the geological conditions of thin coal seams, the main factors which restrict the realization of safety and high-efficiency mining were analysed. The conclusions are summarized as follows:

- Thin coal seams have various disadvantages: low mining height, low level of mechanised equipment, bad working environment, high drivage ratio, high labour intensity, low safety factor, low work efficiency and high input-output ratio, which restricts the development of safety and high-efficiency mining.

- High-performance of high-power coordinated equipment is restricted because thin coal seam contains concretions of pyrite gangue. Therefore, seismic methods of exploration were adopted to explore the distribution and regularity of gangue in thin coal seams. Theoretical calculation and LS-DYNA numerical simulation were employed to analyse the rational blasting parameters. Field experiment shows that by means of the technique of deep hole pre-splitting blasting along roadways, safety and high-efficiency mining can be successfully realised.

- Based on the geological conditions of the thin protective coal seams, which are extremely close to the below outburst coal seam, though advanced fully mechanised mining equipment and various efficient measures separately, safety and high-efficiency mining was realised, and it provides great advantages to mine the outburst coal seam.
The main problems of mining in thin coal seams include the low level of equipment automation, slow advance rate of mixed coal-rock drifts, big labour intensity and bad working environment. Manless working face mining in thin coal seams will be realised by means of improving the mining technology and automatic level of mining equipment.

ACKNOWLEDGMENTS

Financial support for this work was provided by the Qing Lan Project of Jiangsu Province, the State Key Laboratory of Coal Resources and Mine safety, CUMT (NO. SKLCRSM09X02), the Graduate Students Innovation Fund of Colleges and Universities in Jiangsu Province (NO. CX10B_148Z) and the Fundamental Research Funds for the Central Universities (2010QNA32). The authors gratefully acknowledge financial support of the above items. Fangtian Wang is currently on a visiting fellowship to the University of Wollongong.

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GEOTECHNICAL CONSIDERATIONS FOR LONGWALL TOP COAL CAVING AT AUSTAR COAL MINE

Adrian Moodie and James Anderson

ABSTRACT: Austar Coal Mine has had a long association with difficult strata control conditions associated with depth of mining and a highly jointed/cleated coal seam. Poor longwall face conditions, cyclic loading, heavy tailgate roadway conditions and difficulties in maintaining stable roadways on development at < 5.2 m let alone the geotechnical challenges of an 8.5 m roadway required for installation faces have been matters of concern for management. The introduction of Longwall Top Coal Caving (LTCC) to this environment has aided in the management of some of these issues, but has also given rise to other geotechnical considerations. These additional geotechnical issues associated with LTCC not only require management during operations but also require consideration when evaluating new mining areas at Austar or potential LTCC extractable resources throughout Australia and the world.

In September 2006 LTCC commenced at Austar Coal Mine in longwall panel A1. Since that time the LTCC face has been increased from 147 m to 216 m and finally to 227 m in width, and has also been re-handed and modified in the three fully extracted panels to date. The application of LTCC in panels A1, A2, A3 and now A4 has been very successful both from a coal resource recovery point of view and also in the management of the principal hazards of spontaneous combustion and strata control. This paper focuses on the geotechnical aspects of the application of LTCC at Austar Coal Mine and also reviews some advances in general strata control management at the mine.

BACKGROUND

Location

Austar Coal Mine Pty Ltd (Austar), a subsidiary of Yancoal Australia Pty Limited (Yancoal), operates Austar Coal mine, an underground coal mine located approximately 8 km south of Cessnock in the Lower Hunter Valley, NSW (refer to Figure 1). The mine is an amalgamation of the former Ellalong, Pelton, Cessnock No.1 and Bellbird South Collieries and is located in the South Maitland Coalfields. These operations collectively extract, handle, process and transport the coal from the Austar Mining Complex.

Figure 1 - Austar coal mine locality

History

Underground mining commenced in 1916 at the Pelton Colliery and continued until 1992. Kalingo Colliery began as an underground mine in 1921 and ceased operations in 1961. In the late 1960's the Kalingo Colliery was amalgamated into the Pelton Colliery. Longwall production commenced at the

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Pelton Colliery in 1983 and continued until the mine, then known as Ellalong Colliery, was closed in May 1998 by Oakbridge. Southland Coal then acquired the assets of Ellalong and Pelton Collieries and amalgamated those with Bellbird South, which was also owned by Southland Coal.

Southland Coal developed a longwall operation mining the substantial Bellbird South coal reserves utilising the existing Ellalong facilities and infrastructure.

In December 2003, spontaneous combustion in SL4 resulted in Southland Coal ceasing mining activities. The site of the underground fire was sealed and the mine was placed on a ‘care and maintenance’ program for 18 months. Yancoal purchased the mine in December 2004 and changed the name to Austar Coal Mine.

Austar (the last coal mine working the Greta coal seam) introduced an enhanced form of the conventional longwall recovery system called Longwall Top Coal Caving (LTCC) to the Australian coal mining industry in 2006. The LTCC technology has since been utilised to extract the Stage 1 panels known as A1 and A2 and in 2009 commenced in the Stage 2 panel A3. Current LTCC operations are in the A4 panel with development extending into the recently Approved Stage 3 mining area beneath the Quorrobolong Valley.

Geology

Currently the depth of overburden at Austar is approximately 530 m. The Greta coal seam currently mined ranges in thickness from 4.5 m to 6.8 m. The coal produced is used for coking coal or blend coal, exhibiting extremely high fluidity values. The greatest variation to the coal quality is related to the sulphur level in the upper ply of the seam ranging from <1 to 2.5%. The ash in the current mining area is <10%.

The immediate roof strata of the Greta seam are largely laminite 15-20 m in thickness. Across many of the panels the laminates that are immediate to the seam have been eroded by large paleochannels. The paleochannels consist of fine to coarse grained sandstone. Above the immediate strata and paleochannels a massive sandstone bed exists varying in thickness from <1 m to >5 m, this massive sandstone bed is referred to as the Cessnock Sandstone. The Cessnock Sandstone marks the boundary between the massive overlying marine bioturbated Branxton Formation and the Greta coal measures. The Branxton Formation is typically within 20 m of the seam extending to the sub surface. The Branxton Formation is a fine to medium-grained sandstone with some coarse lenses and very few discontinuities such as bedding or jointing. Due to the lack of discontinuities and the >450 m massive nature of the Branxton Formation further geotechnical challenges are expected in addition to the already challenging mining environment at depths extending beyond 500 m.
LONGWALL TOP COAL CAving DESCRIPTION

Longwall Top Coal Caving (LTCC) is an enhanced form of the conventional longwall recovery system, whereby a rear Armoured Face Conveyor (AFC) (refer to Figure 3) is utilised to extract coal from behind the powered supports that would otherwise be left unrecovered in thick seam environments. The major benefit of LTCC is the ability to safely optimise resource recovery in thick seam deposits. This is achieved by operating a retractable flipper at the back of each shield that allows for recovery of the otherwise wasted +3.5 m of top coal that usually enters the goaf.

It is very similar to a conventional longwall system in that the shearer mines coal conventionally at 2.9 m on the floor of the seam. The top coal is then caved through the rear of each shield onto a second AFC. The system has seen a significant increase in resource recovery from Austar with coal recovery in excess of 85% of the entire seam compared to 40-45% when the mine was previously operated using a conventional longwall system.

Further advantages to increased resource recovery include:

- Lower face extraction height: providing a more stable longwall face with less strata failure delays than is typically experienced with extraction heights greater than 4 m in this type of geological environment;
- Operating cost reductions: the LTCC method enables potentially double (or greater) returns of longwall recoverable tonnes, per metre of gateroad development. This reduces the development cost/tonne significantly, and reducing the potential for development rate shortfalls leading to longwall production disruption (Hebblewhite and Cai, 2004);

![Figure 3 - Longwall top coal caving system](image)

GEOTECHNICAL CONSIDERATIONS FOR LONGWALL TOP COAL CAving

Development and installation roadways

The development process at Austar is no different to that of a traditional longwall mining operation. The normal development roadway dimensions are < 5.2 m wide x 3.2 m high, these dimensions are at the lower end of the scale for most coal mines in Australia. The smaller roadway dimensions are used to help with the stability of the strata. The operation experiences significant pressure bumps on development typically in association with the stiffer rock units located above and below the seam. However, the overall development conditions at Austar although challenging are generally good with coal cavities not typically extending into the overlying stone. A nominal 6 x 2.1 m roof bolt pattern supplemented by 4 m tendons is used with 3 x 1.2 m mechanical anchor rib bolts in each rib with full mesh coverage.

The use of mechanical anchor ribs bolts at Austar is not ideal but is the only method of support that can be installed successfully into the very soft rib conditions prior to the holes closing up upon retracting the
drill steel. Several other methods of rib bolting have been trialled but none have proven to be economic and successful. Consequently the use of steel mechanical anchor bolts and steel mesh even on the block side rib is adopted to control buckling for improved rib behaviour (Colwell, 2004). The positive effect that maintaining as good as possible rib conditions has on roadway roof (and in particular tailgate roof) conditions is very evident.

The drivage of installation roads is typical of most operations with the normal width of the roadway being < 8.5 m and < 12 m at the gate ends. Continued refinement of the installation support patterns are proving to be very successful in opening up the installation roads to full width, with stand time >4 months. This stand time is helping the operation maintain a significant development float and provides great opportunity for an early commencement of the installation of the longwall into the new panels.

As time progresses the greatest challenge at Austar is to develop a successful means of chemical anchor rib support moving away from the currently used mechanical anchor bolts in order to improve the rib conditions as the mine advances beyond a depth of 600 m.

Caving recovery

To the end of 2010, LTCC has been utilised in four (4) panels at Austar Coal Mine with various face widths and mining horizons as described in Table 1 and Figure 4. Extraction height is maintained within the operating range of 2.9 m to 3.2 m which is the optimum operating range for the powered supports and has proven to give optimum caving recovery given the powered support geometry.

Table 1- Austar LTCC panel particulars

<table>
<thead>
<tr>
<th>Longwall Name</th>
<th>Panel Width (solid) (m)</th>
<th>Panel Length (m)</th>
<th>Seam Thickness (m)</th>
<th>Seam Depth (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A1</td>
<td>147.0</td>
<td>1412</td>
<td>5.50 to 6.30</td>
<td>390 to 460</td>
</tr>
<tr>
<td>A2</td>
<td>216.4</td>
<td>1179</td>
<td>5.75 to 6.10</td>
<td>390 to 450</td>
</tr>
<tr>
<td>A3</td>
<td>216.4</td>
<td>1319</td>
<td>4.75 to 6.70</td>
<td>490 to 530</td>
</tr>
<tr>
<td>A4</td>
<td>226.7</td>
<td>420*</td>
<td>4.75 to 6.60</td>
<td>500 to 530</td>
</tr>
</tbody>
</table>

*Retreat distance to 15th November 2010

Figure 4 - Typical LTCC face section utilised at Austar

Total seam recovery (excluding chain pillars) has been in the order of 85% to 95% with caving recovery of the top coal section between 75% and 85% and dilution between 8% and 10%. Under cavability classifications this corresponds to a “good to excellent cavability” rating (Jai, 2001). Recent modifications to the LTTC equipment for the A4 panel has enabled the coal recovery to further increase by enabling top coal recovery further towards the gate ends and also removing the tailgate ramp up to the tailgate roadway.

Jia (2001) reported that there are several major factors affecting top coal cavability including coal strength, cover depth, joints/cleats in the coal body, bands, roof strata competence and cutting and caving ratio (Cai, et al., 2004). Experience to date at Austar Coal Mine indicates that coal strength and stress are major contributors to caving recovery with other factors including joint orientation, stone banding, immediate roof strength as well as operating factors (i.e. web thickness) being secondary factors to the caving recovery.

Operational parameters other than web thickness and support set densities also influence overall recovery. At the commencement of a panel no caving recovery occurs to allow adequate ventilation across the face to the point at which sufficient immediate roof material has caved directly behind the supports to direct airflows across the longwall face. Also, during cyclic weighting events, the speed of retreat is increased and as such caving operations are temporarily suspended, this is in part due to the
low capacity coal clearance system at Austar but also in that the caving recovery process can slow retreat rate in itself, particularly in thicker seam sections. The last operational control that can affect caving recovery and has become apparent at Austar is associated with the immediate stone roof. When operating where the immediate stone roof becomes more massive (i.e. a coarse grained sandstone with limited bedding or jointing) and is within 3 m of the cut roof horizon, large blocks have resulted in damage to the rear caving doors and hydraulic rams (Figure 5). Where the coal seam is thick enough to provide a “buffer” coal can be left to protect the rear doors, however it should be noted that advance of the supports themselves without use of the rear doors will still result in ±1 m of top coal recovery on average. Where this risk exists to extended areas of retreat and sufficient coal is not available the rear AFC may be chosen to be removed for the retreat through the risk zone.

Figure 5 - Immediate roof impacting caving doors and caving door hydraulic cylinder damage

Cyclic weighting management

More prevalent cyclic weighting events have occurred in A3 and A4 longwall panels. Weighting events on relatively wide intervals have been observed prior this and have been associated with the Branxton sandstone unit. However in A3 and A4 a more immediate sandstone channel (within 20 m of the seam roof) appears to be further contributing to the loading cycles on the longwall face. Figure 6 displays the Time Weighted Average Pressure (TWAP) of the powered supports as taken from the Longwall Visual Analysis (LVA) program. Distinctive cyclic weighting events can be seen with the following three key observations:

- As the immediate channel converges towards the top of the seam the weighting intervals are shorter at 10-15 m and generally more intense;
- As the immediate sandstone channel diverges away from the top of the seam the weighting intervals spread to between 25-35 m and are generally less intense;
- On cycles between 120-150 m the previously observed Branxton associated weighting events occur, which when combined with the weighting from the immediate sandstone channel cause the most severe loading.

In one instance in A3 where this occurred and again to a similar degree in A4, the combination of these weighting cycles resulted in the face becoming “iron bound” as shown in Figure 7 and eventually the formation of large cavities made recovery more difficult (note the area of blue low support pressure indicating cavities on the TWAP plot).

Austar has implemented a Trigger Action Response Plan (TARP) for weighting management that utilises both observations on the longwall face and also data displayed by LVA in the control room. The use of LVA has enabled earlier detection of an oncoming weighting event (several hours) and also a better indication of the potential severity of the event via triggers based around the support average pressure in combination with loading rate (bar/minute) and yield counts in a cycle (Figure 8). Depending on the severity (Trigger Level) the following responses are then enacted:

- Cease caving and speed up shearer rate;
- Commence taking of convergence readings;
- Stop development operations (to prevent filling of the surge bin);
- Man critical conveyor belts and belt transfers.

Figure 6 - Time weighted average pressure (TWAP) for longwall A3 and A4

Figure 7 - LWA3 iron bound event

Figure 8 - Loading rate and yield count as displayed by LVA
Application of the TARP’s and prediction of weighting zones are largely still reactive measures to the weighting events with the adopted control of increased retreat rate to “move out” from beneath the weighting not always possible. As a more direct control the LTCC system is able to reduce caving (effective extraction height) to assist in reducing the severity of the weighting event. This was trialled in sections of A3 with anecdotal success. However measuring how much this contributed to the improved weighting management in A3 is not clear as there were also several other changes occurring at the time. Conceptually reduced caving recovery means there is less effective extracted height and less mobilisation of the overlying strata that contributes to the weightings. However, as the units we are looking to control are still reasonably close to the coal seam and the additional coal left as a “pillar” to support these has little strength, the use of a “no cave” zone cannot prevent the weighting cycles but may control them. Back analysis of the A3 area where this was adopted has given us more understanding of how this may assist.

Loading rate is considered to give the best indication of weighting intensity being more independent of other factors such as retreat rate. Figure 9 displays this for an area of the same geological characteristics where full caving and then reduced caving was adopted to assist in weighting management. The following was noted:

- In the normal caving recovery area the loading rates were more intense and occurred over a shorter time interval (i.e. event more focussed);
- Where reduced caving recovery was applied the loading rates were less intense and spread over a longer period of retreat.

This data suggests the geometry and proximity to the coal seam of the weighting units means they could not be fully controlled by additional broken coal in the goaf sufficient to stop their rotation and cantilevering forces (induced vertical and horizontal stresses) acting on the longwall face. However the
loading rate data suggests that whilst the total load in the system is not reduced, the additional coal in the goaf can slow this rotation and developing cantilever forces. Thus allowing more time for the event to occur over and enabling greater time for the shields to move from beneath the detached units before reaching critical loads.

This has lead to further trials and assessment of a “no cave recovery” zone between #40 and #70 supports in longwall A4 with early results indicating improved weighting management once again. A program of further powered support pressure monitoring and data analysis with the potential of extensometery and microseismic monitoring is envisaged.

Tailgate control and pillar design

The relationship between pillar size and tailgate roadway conditions is theoretically understood across the industry (Colwell, 1998; Colwell, et al., 2003; Colwell and Frith, 2009). Monitoring of operations at Austar is furthering our understanding of this and how other than just pillar dimension and stress, factors such as immediate roof geology and extraction height can affect the loading environment around the tailgate roadway during both first pass retreat and under tailgate loading.

Longwall A2 and now Longwall A4 are the first two panels to have tailgates with full double abutment loading occur on the chain pillars and about the tailgate roadway. Table 2 summarises the panel geometries and derived Tailgate Serviceability Ratios (SR) from Strata Engineering’s Tailgate Design Model (TDM) (Thomas, 2009).

<table>
<thead>
<tr>
<th>Tailgate</th>
<th>Pillar Width (solid) (m)</th>
<th>Depth (m)</th>
<th>Panel Widths (LW1/LW2) (m)</th>
<th>SR (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A2</td>
<td>40</td>
<td>400 - 450</td>
<td>147 / 216</td>
<td>0.71-0.78</td>
</tr>
<tr>
<td>A4</td>
<td>45</td>
<td>500-535</td>
<td>216 / 227</td>
<td>0.68-0.71</td>
</tr>
</tbody>
</table>

Serviceability Ratio is a ratio of the actual Stability Factor (SF) and the recommended Stability Factor (SFR) and that in effect, a SR of >1 means that the chain pillar has been over-designed and a SR of <1, that the chain pillar has been under-designed (Thomas, 2009).

After approximately 500 m of retreat in A2, conditions significantly deteriorated in the tailgate roadway such that the two 8 m Megabolts installed every 1 m of roadway (prior first pass abutment) required supplementing with up to 216 tonnes per metre of standing support to control conditions to acceptable levels. For A4 the 8 m Megabolt density increases to 2.5 bolts per metre of roadway and the application of 4 m grouted tendons into both rib lines has been applied. Despite this amount of support areas of chain pillar side guttering and centreline roadway roof bagging were observed on first pass retreat in TGA4 as shown in Figure 10.

Figure 10 - TGA4 first pass abutment centreline bagging and chain pillar side guttering

The deterioration in TGA4 after first pass retreat appeared to be associated with lateral relief and movement towards the recently retreated A3 goaf. This was evidenced by two observations:
- In areas of guttering, the outer roof bolt angled over the chain pillar rib line displayed a distinctive bend in the lower 800 mm of bolt towards the A3 goaf direction (Figure 10), suggesting differential shear movement in the roof towards the A3 goaf typically along a stone band (contact with lower bedding plane cohesion);

- Areas where the guttering was occurring along the chain pillar coincided with areas that had increased span between the outer roof bolt and rib line due to rib deformation during development. Where the centreline roof bagging occurred the rib conditions were much improved.

These observations both support lateral movement in the immediate roof towards the A3 goaf with the only difference being that where the chain pillar side roof bolts and Megabolts are able to better provide confinement to the shear movement in combination with the rib line (i.e. rib has not spalled out on development) the movement is impeded at the line of Megabolts and as such centreline bagging occurs. Where they cannot the lateral movement extends to the rib line where the required confinement is produced by the pillar load and consequently rib line guttering occurs. This concept is similar to that proposed by Tarrant (2004) whereby stress rotation created by the adjacent goaf and differential movement along bedding creates a “bulldozer” affect about the roadway (refer to Figure 11). This effect may then be exacerbated by the increased extraction height (±6 m) of the LTCC system.

Figure 11 - Roadway rotation (skew) and movement towards adjacent goaf (Tarrant, 2004)

Given the depth and relatively small pillar size, even under single abutment loading, it could be expected that this roof deterioration, despite the amount of secondary support, is created by the high loads on the chain pillar. However in TGA5 where a 60 m solid chain pillar exists, the first pass abutment loads from A4 are creating similar occurrences. Further, recent chain pillar monitoring in TGA4 has confirmed other pillar load monitoring at the colliery (as discussed by Colwell, 1998 and Wold and Pala, 1986) in that the maingate loading abutment angle is lower at Austar than at most other collieries at around 11.5° (Trueman, 2010) and that the significant deterioration created on first pass is not due to high pillar loads alone given the applied high support densities. This measurement of the pillar load (Figure 12) reveals an unusual profile whereby the stress is highest on the travel road side (TGA4) of the pillar and not the goaf side (Trueman, 2010) potentially being associated with the shear movement and the increased extraction height of the LTCC system. Further assessment via monitoring in the larger TGA5 pillar is planned to examine this theory.

In relation to barrier and chain pillar stress monitoring exercises undertaken at Ellalong Colliery, Wold and Pala (1986) stated, ‘Observational evidence of heavy abutment loads being distributed about the longwall block more broadly than might have been expected on theoretical grounds tended to be supported by the field measurements’.
Now under tailgate loading conditions, TGA4 is able to be managed such that the block side area between the rib and Link n Lock is showing no signs of major deterioration. Supplementing the initial 2.5 Megabolts per metre of roadway has been 1200 mm nine point Link n Locks installed at a similar initial density to TGA2 at 216 tonnes per metre. This has since been reduced to 112 tonnes per metre following further back analysis of TGA2 using ALTS2009 (Cowell and Frith, 2009) and convergence monitoring data obtained in TGA4. This has resulted in no observable change to roadway conditions.

Longwall recovery

LTCC Recovery at Austar differs from traditional longwall operations because it is necessary to recover of the rear drives, chain and pan line from behind the shields in addition to the normal recovery of drives, chain and pans in front of the shields. To enable the recovery of the equipment from the rear of the shields the gate ends need to be heavily supported to allow for the gate end supports to be removed and provide access to the rear drives and pan line. After the rear drives have been removed the rear chain and pan line can be slid along the backside goaf end of the shields and recovered at the designated gate end prior to any additional shields been removed. The longwall recovery is conducted with a traditional pull sequence, each shield removed in order opposed to a leap frog sequence. The traditional pull sequence provides the advantage of protecting the rear caving door as the tail of the shield swings, in addition the use of a traditional pull sequence also provides opportunities to reduce the bolt density above the canopies reducing the bolt-up time.

CONCLUSIONS

The application of Longwall Top Coal Caving to Australian conditions has been successful but has also highlighted several additional operational and geotechnical factors that need to be managed. Key matters learned from the operation of LTCC equipment at Austar Coal Mine that must be considered both in new areas at the colliery and when assessing other potential thick seam applications include:

- Immediate roof geology and its effect on caving recovery and dilution;
- Immediate roof geology and the potential damage to rear caving equipment;
- Cyclic weighting management and extraction height;
- Pillar loading and tailgate support design; and
- Longwall recovery bolt up design and equipment extraction sequences.

Further investigation programs are planned for Austar Coal Mine to better understand these influences as the mine progress towards the next 20 years of operations in the Stage 3 mining area.
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UNDERSTANDING THE CAUSES OF ROOF CONTROL PROBLEMS ON A LONGWALL FACE FROM SHIELD MONITORING DATA - A CASE STUDY

Robert Trueman¹, Rob Thomas¹ and David Hoyer²

ABSTRACT: The results of an assessment aimed at understanding the shield loading mechanisms associated with strata related issues on a longwall face are detailed. Shield load cycle analysis theories developed by the authors were used to quantify the interaction between the shields and the strata. Shield pressure data was back-analysed using a version of the Longwall Visual Analysis (LVA) software that has been extended to enable efficient load cycle analysis to be undertaken. The software outputs enabled the shield strata interactions to be characterised. The results of the analyses indicated that the major cause of the roof falls was high-level periodic weighting, resulting in periodic shield overload. Exploration borehole data was analysed to identify and characterise the overburden unit that was the likely cause of the shield overloading.

INTRODUCTION

A longwall had been experiencing roof falls on a regular basis throughout its retreat; refer to Figure 1 for the location and height. The results of an assessment aimed at understanding the shield loading mechanisms associated with the strata related issues on the face are detailed. Shield load cycle analysis theories developed by Strata Engineering have been used to quantify the interaction between the shields and the strata. Available shield pressure data from the longwall was analysed using a version of the Longwall Visual Analysis (LVA) software that has been extended to the specifications of Strata Engineering (SEA), the results of which have enabled the characterisation of the shield-strata interactions. Exploration borehole data was analysed to delineate the overburden units that could influence shield loading. The reasons for the roof falls were determined and recommendations for alleviating the problem were given.

SHIELD-STRATA ANALYSIS METHODOLOGY

Shield load cycle analysis theories developed by SEA have been used to quantify the interaction between shields and strata. These theories have previously been outlined in Coal 2008 and 2010 (Trueman, et al., 2008 and 2010). A load cycle is the change in support pressure with time, from setting the shield against the roof to the next release and movement of the support.

An off-line version of the LVA software has been extended to provide the following critical load cycle features for each leg of each support:

- **Time Weighted Average Pressure (TWAP) Ma, Figure 2** - note: a) the TWAP, is calculated between the initial setting of the shields to the roof and the final release at the end of the load cycle; b) a value is calculated for each leg of each shield for every load cycle identified by the software; c) zones of high loading are shown in red and zones of low loading in blue, and d) the map gives a good overview of the loading environment.

- **Number of Yield Events Map, Figure 3** - note: a) the number of yield events in individual cycles have been colour coded; blue indicates 1 to 3 yields and orange/red >8, b) the number of yield events in a single load cycle is a very good indicator of high-level periodic weighting being experienced on a face, c) high-level periodic weighting is characterised by a shield reaching yield and continuing to yield throughout the load cycle, d) an increase in the number of yield events in a load cycle leads to more roof closure, e) deterioration in roof conditions between the shield tip and the face is normally seen when the shield reaches a threshold value of closure after multiple yield events, f) if a support continues to yield throughout the load cycle, its mode of operation generally changes from force control to deformation control, g) a shield in the deformation control mode of

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operation can be regarded as overloaded, h) the length of the load cycle is very important when supports are overloaded because more yields and therefore more convergence occurs in long cycles which leads to a deterioration in roof conditions once the abovementioned closure threshold value has been exceeded, i) the intensity of loading also has a bearing because a greater intensity of loading means that more fluid is expelled per yield event, which leads to more closure per yield event; less yields are therefore needed to reach the closure threshold and j) current closure monitors used on longwall faces are unreliable so cannot be used to infer the shield-strata interaction (SEA are currently working to correct this deficiency).

Figure 1 - Roof fall locations

Figure 2 - Time weighted average pressure map

- Low Set Pressure Map, Figure 4 - note: a) set pressures that are too low have been found to lead to roof control problems on the face, because tension is not eliminated in the immediate roof material and as such, natural and mining induced fractures are allowed to dilate and in doing so degrade the mechanical interlock; b) previous experience shows that a set pressure of <40 t/m² is the typical threshold value at which roof control problems can result; c) set pressures of >60 t/m² have been found to ameliorate potential roof control problems; d) for each panel, maps highlighting set pressures equating to <40 t/m² before the cut have been generated, and e) the length of the load cycle is again important when set pressures are too low, as a longer cycle time will allow more dilation of the natural and mining induced fractures within the immediate roof strata.
Figure 3 - Number of yield events map

Figure 4 - Set pressures <180 bar map

Figure 5 - Set pressures <360 bar map indicating where high set disabled

Figure 6 - Initial loading rate map
• **<360 Bar Set Pressure Map, Figure 5** - note: this map can be used to determine the periods and for which shields the high set pump is not operating.

• **Initial Loading Rate Map (Figure 6)** - note: a) this map illustrates the loading rate (bar/min) calculated between five and ten minutes after setting, b) previous experience suggests that the initial loading rate is a good indicator of the intensity of the loading conditions and c) whilst still requiring further verification, indications are that initial loading rates of <10 bar/min are indicative of generally low-level periodic weighting and >10 bar/min, higher level periodic weighting.

• **Anomalous Leg Pressure Map, Figure 7** - note: a) the software identifies differential loading rates between two legs on a single shield and flags the leg with the lower loading rate as potentially having faults with the hydraulics, valves or sensors, b) where anomalous legs are grouped together, low set pressures on the anomalous shields and overloading of the adjacent shields can result, c) both scenarios have been found to result in roof control issues and d) high set and/or guaranteed set when enabled will mitigate, but not eliminate potential roof control problems associated with leaking legs; as will short cycle times.

• **Load Cycle Time Map (Figure 8)** - note: a) the length of the cycle is of particular importance where the shields are being overloaded or being set too low as noted above, b) in such instances the additional cycle time allows more roof convergence when supports are overloaded and greater dilation of fractures when set pressures are too low and c) cycle time is of less importance where the shields are being adequately set and are stabilising the roof within a standard cycle time (i.e. they are in a force control mode of operation).

• **Calibration Map, Figure 9** - note: this map highlights the shields where the pressure sensors are badly calibrated.

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**Figure 7 - Anomalous leg pressure map over one month**

The data are presented as maps in which the x-axis represents the support number counting from the MG end of the face. The y-axis represents the shear number in the direction of mining and is therefore proportional to mining advance. Each shear represents a single load cycle for a shield, which enables load cycle analysis to be carried out.

The plotted value is the variable of interest (i.e. one of the critical load cycle features noted above), and as discussed, is coded by colour. Using colour as the third dimension has been found to be the most effective way of enabling a rapid evaluation of the support-strata interaction.
SHIELD-STRATA INTERACTION ASSESSMENT

Historical shield pressure data was made available to facilitate a back analysis using the extended version of the LVA software. Instances occurred when the leg pressure signals were not available and periods of signal loss of >2 hrs duration are marked on the maps.

The main points of note with regard to the back analysis are as follows:

- The TWAP Map, (Figure 2) shows (i) a distinguishable weighting cycle with 10 to 20 m intervals between peaks; (ii) a few vertical green stripes indicating support legs either with hydraulic problems or where the pressure sensors were badly calibrated; (iii) a number of patches of very low TWAP values (blue areas), which correspond to areas where cavities were mapped (see Figure 1), and (iv) there are areas where cavities were mapped that do not show up, indicating that reasonable set pressures were being maintained in these areas and that the roof above the cavities was capable of transferring load to the shields.

- The Yield Event Map, (Figure 3), shows (i) at most of the peaks of the weighting cycle, >50% of the supports on the face yielded, (ii) a large number of yields (i.e. >10, shown in red) occurred in a single load cycle on several occasions, particularly in the longer cycle times and (iii) multiple shields experiencing multiple yields preceded all of the mapped cavities.

- The Low Set Pressure Map (Figure 4), shows (i) a few vertical stripes that are most likely associated with support legs with either hydraulic issues or faulty / poorly calibrated sensors; (ii) clusters of localised low set pressures that correlate to low TWAPS and correspond well to the mapped cavities noted in Figure 1, and (iii) there are a number of instances where cavities were mapped where low set pressures are not evident, indicating that adequate set pressures were achieved in these areas (these cavities were generally 0.3 to 0.6 m high but a few were 0.6 to 1.0 m high) – note: a) low set pressures in and around roof cavities would tend to increase the height, width and length of cavities, and b) whilst it is important to maintain tip contact and not force the rear of the canopy up into existing cavities or broken roof, where possible (even when setting
manually), attempting to achieve a load density of ≥ 40 t/m² (≥ 180 bar) and preferably ≥ 60 t/m² (≥ 260 bar) on setting would potentially reduce the severity of roof control issues.

- The <360 Bar Set Pressure Map (see Figure 5) shows (i) the areas of the face where the high set pump was disabled; (ii) these areas mainly correspond to where the cavities were mapped but there are a few areas where this was not the case, and (iii) where the disablement was not in response to existing cavities, roof control difficulties did not occur when the high set pump was disabled.

- The Initial Loading Rate Map (see Figure 6) shows (i) a 10 to 20 m weighting interval, and (ii) loading rates at the peaks of the weighting cycle of >10 bar/minute immediately preceding the areas of the face where cavities were reported.

- The Anomalous Leg Pressure Maps shows that at any one time, ~4% of the shields (2% of the legs) on the face have been identified as having anomalous pressure readings (see Figure 7) – note: a) a check of the raw pressure data has indicated that most of these legs had hydraulic issues; b) this proportion of legs with hydraulic issues, on longwalls not operating with new equipment, is well below the average for the industry; c) the fact that no anomalous leg pressures were identified on a single leg for a significant length of retreat means that leaking legs were being repaired in a timely manner, and d) on this basis it is assumed that shield maintenance issues would have had minimal negative effect on the longwall weighting environment.

- In regard to the Cycle Time Map (see Figure 8), it is of note that (i) areas with a high number of yields in single load cycles tended to involve long cycles, and (ii) long cycle times usually preceded cavities.

- The Calibration Map (see Figure 9) shows that 12 shields on the face had badly calibrated pressure sensors.

Based on the above, high-level periodic weighting was being experienced over the majority of the analysis period, with the shields being periodically overloaded on a number of occasions. Cavities were generally formed following times when periodic shield overload was experienced, especially in long cycles. Adequate set pressures were achieved in some of the cavities but not in others. Where adequate set pressures were not achieved this would have probably tended to increase both the height and extent (length and width) of the cavity.

Set conditions deteriorate after shield overloading events, even if cavities are not present. Operational controls can nevertheless be effective in minimising roof control issues in the presence of high-level periodic weighting leading to support overload. Specific attention to attaining the highest set pressure practicable, without compromising the attitude of the support canopy can reduce the extent of cavities and associated delays in many instances. Mining as rapidly as possible through overloading events, to minimise the number of yield events in a single load cycle, is another operational control, as is minimising the number of shields with maintenance issues.

WEIGHTING UNIT ASSESSMENT

Given that high-level periodic weighting had been concluded to be the major cause of the roof control difficulties, a number of boreholes in the vicinity of the longwall were assessed for the presence of thickly bedded to massive. As noted previously, high-level periodic weighting is generally characterised by shields reaching yield and continuing to yield throughout the load cycle. The fact that a shield yields does not in itself constitute high-level periodic weighting. On many occasions, particularly as set to yield ratios have increased, shields yield at the peak of the weighting cycle. However, in the absence of thickly bedded to massive units, shields tend to yield only once or twice in a load cycle and then stabilise below yield. In such a case, and cases where no yielding occurs, this would be termed low-level periodic weighting. In certain circumstances when set pressures are very high relative to the yield value, a shield will reach yield and continue to yield throughout the load cycle but the intensity of loading is such that little to no fluid is lost and so the closure per yield event is small. In such a case, the shield would not be considered to be experiencing high-level periodic weighting. In the absence of reliable closure monitors, care must be taken to ensure that high-level periodic weighting is actually occurring and other factors such as loading rates and roof fall data must also be considered to infer that high-level periodic weighting is actually occurring; i.e. a shield yielding throughout a load cycle cannot be used in isolation to infer high-level periodic weighting.
High-level periodic weighting is caused by thickly bedded to massive strata forming significant cantilevers in the strata overlying the extraction area. SEA's database on low and high level periodic weighting (refer to Figure 10) indicates that the magnitude of the weighting event increases when competent sandstone beds reach a thickness of ~20 m. Conglomerates generally have a greater spanning ability and high-level periodic weighting has been observed at a thickness of ~14 m. Such competent beds have been shown to influence shield loading at interburden thicknesses from the extraction horizon to the base of the unit of up to ~70 m, although historically, units within 40 m of the seam appear to have the greatest potential to influence weighting behaviour. It would be anticipated that the intensity of the high-level periodic weighting would increase with both the thickness and proximity to the seam of the weighting unit. However, in the absence of reliable closure monitors it is very difficult to quantify the intensity of the high-level periodic weighting. As noted previously, SEA are working to overcome this deficiency and it is expected that in the future the intensity will be able to be better quantified.

Information on the near-seam geology was made available from a number of boreholes in the vicinity of the longwall. A thickly bedded to massive conglomerate/sandstone unit located between 56 m and 67 m above the extraction horizon was identified from core photographs and geophysics as the likely cause of the periodic weighting. No other nearer seam thickly bedded to massive unit reached a thickness that has been observed to result in high-level periodic weighting at other mine sites. Although at the time of writing, the geological work had not been completed, an interpretation of the thickness of the thickly bedded to massive portion of this unit was made (see Figure 11). The unit was assessed to have a thickness ranging between 12 m and 25 m over the length of the panel. Given the conglomeritic nature
of the unit, it is anticipated that it would be capable of producing high-level periodic weighting on the face at a thickness greater than 14 m. Furthermore, as the spacing between boreholes was several hundred metres, the proposed thickness contours should be regarded as approximate.

CONCLUSIONS

Shield monitoring data was made available and analysed using an extended version of the LVA software. High-level periodic weighting was observed to be occurring over the majority of the area analysed. A significant number of roof control problems were reported and observed on the load cycle maps.

Over the majority of the area of retreat, the intensity of the periodic weighting at the peaks of the cycles was sufficient to cause yielding of the supports across much of the face. In most of the events the shields continued to yield throughout the load cycle, indicating that the supports were in deformation control mode during these times. Once a shield is in deformation control mode it can be regarded as overloaded, noting however that this does not necessarily mean that roof control problems are inevitable but that the risk does markedly increase. The periodic shield overloading was concluded to be the major cause of the roof control problems experienced. It was noticeable that cavities generally only occurred following long cycle times, which is typical of operations experiencing high-level periodic weighting.

A thickly bedded to massive conglomerate/sandstone unit was interpreted to be the most likely cause of the high-level periodic weighting. This unit was estimated to be between 12 m and 25 m thick and located ~60 m above the extraction horizon.

REFERENCES


DEVELOPMENT OF A CAVITY PREDICTION MODEL FOR LONGWALL MINING

Bronya Wiklund¹, Mehmet S Kizil¹ and Ismet Canbulat²

ABSTRACT: Advancements in technology over the past decade, in data collection and computer modelling systems, have created opportunities to develop and improve the current methods of predicting roof stability issues in longwall mining operations. The ability to accurately predict roof instabilities and cavity developments has great benefits for the coal industry. Early prediction will allow for appropriate actions to be taken to avoid such events, removing the potential for harm to personnel and loss of production. A case study of Moranbah North Mine, investigating the causes of roof stability issues, concentrating on the development of roof cavities in the longwall face is presented. Results from an investigation of the effect of particular geological factors on the occurrence of such instability events are recorded. From the investigation a stability index was developed from geological data collected from boreholes on site. A hazard map was developed, using the index, to indicate areas in the roof where failures and cavities were most likely to occur. Although some correlations were found between the index and geological factors, the results were not entirely satisfactory as some important factors had not been included in the prediction model which is still being improved.

INTRODUCTION

Roof stability issues and the development of cavities in the immediate roof are a key concern for underground longwall mines. New technologies have enabled a greater knowledge and understanding of geological factors and in situ stresses in an underground environment, leading to a more accurate prediction of roof stability. These facilitate a safe working environment, which is imperative to all mining operations.

An investigation into the effects of particular geological factors on the development of cavities at Moranbah North Mine was conducted. The geological factors investigated included:

- Seam thickness;
- Depth of cover;
- Sandstone thickness;
- Interburden thickness; and
- Faulting.

These factors were used to develop a prediction model that highlights areas of concern along a longwall panel. The prediction model was developed using borehole data collected at the site. The prediction model was compared to data collected from the longwall chocks using Longwall Visualisation Analysis (LVA) software, to identify whether a correlation existed between the known cavity events and the prediction model developed. An accurate indicator will allow time for actions to take place to eliminate or at least reduce potential roof-stability issues or developing cavities and to avoid major time losses.

LONGWALL MINING

Longwall mining is the most common method of underground coal extraction used in Australia today. Longwall mining extracts coal in large rectangular blocks, defined during development, in a single continuous operation (Aziz, et al., 2007). Each block of coal, known as a panel, is developed by driving a set of headings on either side of the panel off the main access roads. The start of the working face is created by the joining of these roadways. The longwall face is supported by hydraulic roof supports, whose main function is to provide a safe working environment as the coal is extracted and the longwall equipment advances. A goaf is formed as the immediate roof is allowed to collapse behind the mined out area. Figure 1 shows a schematic of a typical longwall retreat method.

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When designing a longwall panel layout, along with coal property boundaries, several factors will dictate the final result. Peng (2006) listed the following factors:

- Reserve;
- Panel dimensions;
- Geology:
- In-situ (horizontal) stresses;
- Multiple seam mining;
- Rivers/streams or lineaments; and
- Surface subsidence.

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Figure 1 - Longwall retreat mining (Mine Subsidence Engineering Consultants, 2007)

GEOLOGICAL FACTORS

Seam attributes

Longwall units are better suited to regular seam trajectories as undulating seams cause issues because of irregularities in the roof and/or floor. Such instances can affect the stability of the structure or the quality of the product (Carroll, 2005). In general longwall mines have been developed in relatively flat lying seams, with inclinations of no more than 10°. With developing technologies and the increasingly strong coal market, seams dipping 15% to 20% are becoming more economically and physically viable to mine.

The thickness of a coal seam is another major contributing factor in the selection of the longwall mine design. Economically, maximising the recovery in thick seams can prove to be highly beneficial; however mining thick coal seams can often lead to roof stability issues that must be alleviated to maximise the benefits. Geological anomalies, that in some way affect the coal seam, have serious implications for the successful implementation of a longwall extraction method. These include faults, folds, sandstone channels, clay veins, and fractures (hill or mountain seams). It is important that these occurrences are identified, by mapping their locations using geophysical methods or at least determined panel by panel during gate road development. Locating the anomalies will have a strong influence on the layout of the main headings and working panels.

Floor and roof characteristics

Good roof and floor strata conditions are preferred for fast moving longwall operations. It is important to determine the properties of the near seam strata during the investigation stage, and ensure that the properties defined are not restricted to uniaxial compressive strength (UCS) and tensile strength. According to Medhurst (2005), roof stability is a function of lateral confinement, generated by the support resistance and the coal seam. This is of particular importance in areas of weak immediate roof. Confinement generates an increase in shear strength of rocks. Hence the coal seam and support legs of the longwall become the main abutments for arching of the immediate roof strata and the canopy provides active pressure within the arch zone. If the abutment is lost, the support system will break down.
Roof material properties and orientations must be accurately tested to determine the forces to which the longwall supports will be exposed. As the longwall face advances, the vertical forces that were once being applied to the coal will now be directed towards the mining face and also to the armoured supports. Thus, the equipment selected must take into account the yield capabilities of the roof relative to the imposed loads.

Another predominant issue related to underground coal mining and roof stability is the presence of massive overburden strata. Periodic weighting issues can be caused by the presence of massive sandstone channels in the overlying strata. The distance of the massive strata unit from the seam influences support loadings that develop. The strength, distribution and character of massive sandstone structures in the roof lead to several issues when longwall mining, including (Medhurst, 2005):

- Cantilever effects that overload supports under ‘massive’ conditions (including panel start-up);
- Detachment of large blocks that are able to overload supports; and
- Development of small blocks in the tip-to-face area disrupting cutting.

The occurrence of roof cavities can also be attributed to a combination of roof guttering due to multiple yielding events, resulting in poor set conditions, and low set pressures on subsequent shears. The development of cavities generally occurs over two to three load cycles. In some cases when cycle time has increased long enough, for example due to planned maintenance or machine breakdown, cutting can deteriorate into cavities within a single load cycle.

Technical considerations

Roof stability in longwall operations can be affected by several operating factors including:

- Canopy tip-to-face distance;
- Hydraulic supply and control settings; and
- Cutting height.

All of these factors should be taken into account when developing the roof stability model, as each can be resolved by modifying and improving operating techniques. The chances of roof instability issues arising can be reduced by ensuring all operational factors are observed in areas of concern. This will ensure a safer working environment and reduce or eliminate lost time due to operational issues.

PAST STUDIES

Extensive research into the geological factors and roof stability of longwall mines has been performed over the years. From this research many prediction methods and rock mass classification systems have been developed. Quantifying the critical geological factors when evaluating roof stability can be difficult due to the varying geological and structural settings of different deposits. According to Peng and Chiang (1984), various geological factors have been investigated in an attempt to classify the roof strength, including; lithological sequence, roof convergence, unsupported time durations before caving, seismic wave velocity, drill core strength, average frequency of bedding planes and rock strength, and bed separation resistance.

Coal mine roof rating (CMRR)

One example of a rock mass classification method is Coal Mine Roof Rating (CMRR). As this method was developed for US coal mines it lacks adequate boundary definition for low strength lithologies, such as those common to Australian coal seam strata (Hatherly, et al., 2009). Major roof disruptions such as faults or shears are not included in the assessment. The CMRR can be determined from either underground exposures, including roof falls and overcasts, or from exploratory drill cores. The main parameters measured according to Mark and Molinda (2005) are:

- The uniaxial compressive strength (UCS) of the intact rock;
- The intensity (spacing and persistence) of bedding and other discontinuities;
- The shear strength (cohesion and roughness) of bedding and other discontinuities;
- The moisture sensitivity of the rock; and
- The presence of a strong bed in the bolted interval.
Secondary factors to be considered include the number of layers, groundwater, and surcharge from overlying weak beds. All these factors are individually rated, and the summation of these make the final generated CMRR, in the range from zero to 100 (zero being weak and 100 being very strong).

**Longwall shield-strata interactions**

A study released in 2010 by Trueman *et al.* (2010) attempts to quantify the impact of cover depth and panel width on longwall shield-strata interaction. The investigation used recently developed shield load cycle analysis theories, allowing factors influencing shield loading to be isolated and to quantify the interaction between the shield and strata. Five sets of historical data from different mines were back analysed, as well as strata delay data for the longwall faces. Also included in the investigation was an assessment of the near seam overburden geology for each of the sites.

Maps of the critical load cycle parameters implicit to the utilised analysis methodology were provided using Longwall Visualisation Analysis (LVA) software. The LVA software was modified such that it presented the outputs as the individual load cycles for every shield rather than to a time or chainage basis, allowing for the load cycle analysis to be conducted (Trueman, *et al.*, 2010). The investigation found that the presence or absence of thickly bedded to massive units in the immediate roof had the greatest impact on shield loading. The analysis showed that once the thickly bedded to massive sandstone units exceeded 20 m in thickness high level periodic weighting and periodic shield overload occurred. The periodic weighting transitioned between low and high once the sandstone bodies exceeded 16 m. The height of the massive bodies above the shield that still had influence on loading was observed to exceed 70 m. By comparing the data from the five different mines this study concluded that, within the range of the data investigated, depth on its own did not majorly affect the loading on the shields.

All longwall widths examined in the study showed the potential for shield overload. However, the reduced cycle time associated with narrower panel widths was found to have a significant effect on roof control if periodic support overload occurs (Trueman, *et al.*, 2010). This was due to the reduced number of yields and the subsequent roof degradation. The investigation also highlighted the significance of shield maintenance and operation on the shield loading environment. Increased load on the adjacent legs and supports can result from inadequate maintenance of the shields. When set conditions deteriorate, leading to low set pressures, roof control problems were experienced due to the destruction of the mechanical interlock of the strata above the supports.

**CASE STUDY - MORANBAH NORTH MINE**

**Moranbah North Mine**

The data used from this investigation related to longwall 108 at Moranbah North Mine. A plan layout of the mine is shown in Figure 2 and longwall 108 is indicated.

Moranbah North is an underground coal longwall mine, located approximately 18 km's north of the town of Moranbah, Central Queensland. Anglo Coal Australia manages and operates the mine which began operations in 1998. The operation mines approximately 4.5 Mt pa of hard coking coal from the Goonyella Middle (GM) Seam to the northern end of the Bowen Basin.

Previous longwall blocks at Moranbah North have been plagued by a series of weighting events that have been linked to poor longwall face stabilities (AMC Consultants Pty Ltd, 2006). These weighting events have led to significant delays in production, resulting in below-plan performance of the longwall. The incidence of cavities developing in the longwall face has been identified as a result of these weighting events. Such cavities have ranged in depth from a few centimetres to metres in the roof and have spanned from one shield to tens of shields in width. Previous investigation of the cavities has also indicated a potential link between cavity development and poor set pressures of the longwall shields and also horizon control.

**Longwall visualisation analysis software data**

Real time data was collected from the longwall chock legs, and then recorded in LVA software. The data set obtained from the longwall chock legs measures the pressures at the mining face as it progresses. Data is taken from each of the legs that span across the whole of the longwall and is recorded every
minute. This data can be sourced using the LVA program, which collects live data and shows values for a variety of measurements including:

- Time weighted average pressure (TWAP);
- Initial loading rate;
- Yield; and
- Low set pressure.

![Figure 2 - Moranbah North Mine plan](image)

**Geological data**

The second source of data used in this investigation was geological data collected at Moranbah North. The geological data used included:

- Seam thickness;
- Depth of cover;
- Sandstone thickness;
- Interburden thickness; and
- Faults.

Each of these factors has shown to have some effect on the stability of the roof in a longwall mine and also the development of cavities in the longwall face. The analysis of these parameters aims to determine a more accurate way of predicting such events by developing indicators to highlight areas of concern.

The geological data was collected using borehole sampling. Over a lease area numerous boreholes are drilled and core samples are extracted to be analysed to determine vital information about a potential mine. The relevant data to this investigation is the geological properties of the coal and surrounding strata. Figure 3 shows a section of the tailgate view of longwall 108 and the surrounding rocks. This figure also shows the geophysical logging which is used to determine the properties and type of the rock.

**Seam thickness**

The seam thickness simply describes the thickness of the seam being mined. At Moranbah North, the Goonyella Middle (GM) seam is being mined. The GM seams thickness fluctuates between 5.2 m and 6.4 m, within the mining lease. The mining height is approximately 4 m.
Interburden thickness

A rider seam which is called the Goonyella Middle Roof (GMR) seam splits off the GM seam. This seam is approximately 0.3 m thick, and rides approximately 0.5 m above the GM seam, until it splits off and the interburden begins to increase.

Overburden thickness

The overburden at Moranbah North consists of coal seams, siltstone, sandstone and claystone. On top of this is approximately 60 m of tertiary sediments consisting of poorly consolidated sands and clays with occasional basalt flows (Carroll, 2005). The depth of cover increases to the east of the mining lease.

Sandstone bodies

Three major sandstone bodies exist within the overburden. These were investigated to determine their effect, if any, on the occurrences of roof instabilities and cavity development. The three sandstones examined are named MP/MR20, MP/MR42 and MP/MR41.

Faults

Some minor faults were detected within the Longwall 108 block. These were also examined to determine if they contributed to the previous occurrences of roof instabilities at Moranbah North.

Eliminating data inaccuracies

Before analysis of the data could take place, any potentially erroneous data had to be eliminated to assure all results were as accurate as possible. In relation to the LVA records, from observation of the graphs produced, it was obvious that errors occurred in the recording of data at the beginning of the longwall development, which can be seen in Figure 4. This section of the data was excluded from the investigation.

The presence of zeros in the LVA raw data caused problems when modelling the data using Surfer™. This was due to the fact that the program interpolates between points. Instances in the data that show almost instantaneous drops of pressure from close to 400 bars to zero indicate errors occurring in the recordings taken from the chock legs. These errors are most likely due to faults in the technology. Hence, all zeros were removed from the data set.

The geological data had to be modified after it had been contoured in Surfer and new grid files were developed. Surfer™ is an interpolation program which turns scattered X, Y and Z data into maps and contours (Golden Software, 2010). In doing so, some of the data points generated in Surfer™ became negative. This is obviously impossible as all parameters were thicknesses. Thus all values that recorded a negative value were excluded.
Due to the fact that the pressures were measured every minute, the data collected from the LVA software produced tens of thousands of rows for each of the longwall chocks. As such large quantities of data cannot be handled by Surfer\textsuperscript{TM}, the data had to be condensed, by averaging all readings recorded in relation to the chainage, using a macro developed in Windows Excel\textsuperscript{TM}. Once the data had been condensed, it was then organised into three columns representing easting, northing and chock pressures. Once again a macro was used to organise the data.

Once organised into columns the easting’s and northing’s had to be changed to replicate the coordinate system of the geological data so it could be accurately compared. The geological data used included the longwall block and some surrounding areas, as the longwall is positioned at an angle approximately 20° west from north. Hence the LVA data had to be rotated to this angle so an accurate comparison could be made. A two-step process was applied for this rotation.

First the easting’s and northing’s had to be changed, such that the bottom left hand corner of the LVA data corresponds with the bottom right hand corner of the longwall block true coordinates. This coordinate point was to be the point of rotation for the LVA data. Once this point was determined each of the easting and northing points were changed by adding the distance along the length or the width of the longwall block to the coordinate. Two equations were then used to rotate each of the points by 110°, such that it is orientated the same as the geological data. The rotation equations are shown as Equations 1 and 2.

\begin{align}
X' &= X \cos \theta - Y \sin \theta \\
Y' &= X \sin \theta + Y \cos \theta
\end{align}

After rotating the data set to align with the longwall panel, a contour map was generated using Surfer\textsuperscript{TM}. This contour map is shown in Figure 5. The lower pressures (red) indicate where cavities have occurred, as the pressure on the chocks is less due to the void created above.

**Geological data**

The raw borehole data was imported into Surfer\textsuperscript{TM} where grid files were developed for each of the data sets. A rectangular area that encompassed longwall 108 block was selected from the mine plan to
The new grids were developed such that the easting and northing points for each data set were identical. This allows for easy comparison and combining of the data. These grids were exported to excel including the new ‘Z’ values interpolated by Surfer™. This was performed for the seam thickness, depth of cover, interburden thickness and sandstone thickness. The effect of faulting was examined simply by overlaying the LVA data with a map showing where faulting had been detected.

Developing the prediction model

**Geological index**

To develop a prediction model the data for the seam thickness, depth of cover, interburden thickness and sandstone thickness had to be scaled such that their effect was comparable to the other parameters. As the effect of the thickness of the geological parameters on roof instabilities and cavity developments did not follow a simple linear relationship, the thickness values were scaled to a value between one and ten using an exponential relationship to determine the indexes. Figure 6 shows a comparison between the index contour and the original contour for the overburden thickness.

**Weighting of individual parameters**

A weighting factor was applied to each of the individual geological indexes to account for the different effects each had on the roof stability. To determine the weighting of each of the factors, contours of each index were developed and compared with the LVA data. Figure 7 shows the overburden thickness index contour overlaid with the contour produced from the LVA data. With reference to Figure 5, it can be seen that there is little correlation with the individual cavities that have occurred. However, there is some correlation with the depth of cover and the frequency of cavities. As the depth of cover decreases the number of cavity events also decreases.

Figure 7 shows an example of how the weighting factors were determined. Little correlation was found between the parameters investigated. Those that showed the most correlation were the seam thickness, overburden and interburden thickness. One major relationship that was identified was that as the thickness of the overburden and interburden increased the frequency of the cavities also increased. The sandstone bodies showed little correlation, if any. However, each of the sandstone bodies had relatively constant thicknesses, with few peaks or plateaus that the cavities can be compared to.

From the examination of each individual contours it was decided that the weighting factors applied to each of the individual parameters would be:
- Seam thickness = 0.7;
- Overburden thickness = 0.8;
- Interburden thickness = 0.9;
- MP/MR41 sandstone thickness = 0.4;
- MP/MR42 sandstone thickness = 0.5; and
- MP/MR20 sandstone thickness = 0.3.

![Image of Overburden thickness index contour](image.png)

**Figure 7- Overburden thickness index contour**

To determine the index Equation 3 can be used.

$$I = ST_i \times W_{ST} + OT_i \times W_{OT} + IT_i \times W_{IT} + ST_{41i} \times W_{ST41} + ST_{42i} \times W_{ST42} + ST_{20i} \times W_{ST20}$$  \hspace{1cm} (3)

Where: $ST_i$: Seam thickness index;
$OT_i$: Overburden thickness index;
$IT_i$: Interburden thickness index;
$ST_{41i}$: MP/MR41 sandstone thickness;
$ST_{42i}$: MP/MR42 sandstone thickness;
$ST_{20i}$: MP/MR20 sandstone thickness; and
$W$: weighting factors relevant for each of the parameters.

Each of these weighing factors was applied to the index values before they were summed to produce the prediction model.

**Prediction model**

Figure 8 shows the prediction model that was produced after each of the parameters had a weighting factor applied. Greater correlation can be seen from the prediction model compared to the individual parameters. The red arrows show where there was good correlation between the prediction model and the LVA data. The green arrows indicate significant cavities that occurred that were not highlighted in the prediction model. It must be noted that these cavities still exist in an area with a relatively high index number. No major cavities have occurred in areas that have achieved a low index value.

The prediction model produced showed minor correlation with the cavity events that were recorded using the LVA technology. Thus it can be deduced that the thickness of the seam and surrounding strata contained within the roof have little to no effect on the occurrence of roof cavities and unstable roof conditions as individual components. When combined, more correlation with known cavity events was detected. Poor roof conditions are expected in areas where the longwall chocks are exposed to high pressures. This implies areas where the overburden is extensive, and areas where thick bodies of high
density strata exist within the roof. This leads to large amounts of weight force being applied to the immediate roof just above the coal. Such weighting issues can lead to events such as:

- Cantilever effects that overload supports under ‘massive’ conditions;
- Detachment of large blocks that are able to overload supports; and
- Development of small blocks in the tip-to-face area that disrupt cutting (Medhurst, 2005).

These effects are amplified if the strata unit is close to the seam. This justifies the correlation with the increasing depth leading to the increasing numbers of cavities.

One feature that could account for poor correlation with the geological data is lateral confinement, built up when coal is left in the roof. Leaving a thick layer of coal in the roof has been proven to alleviate roof instabilities issues when utilising a longwall mining method, particularly in thick seam mining. This method is applied at Moranbah North as the GM seam is relatively thick and highly variable in parts.

One other issue that could account for poor correlation is the fact that the geological data was extrapolated, using common estimation methods, from boreholes which were spaced relatively far apart. Boreholes for the purpose of determining the properties of the seam and surrounding strata are very costly, thus the minimum necessary boreholes are taken. The contours created from the borehole data simply predict the properties of the strata, where as geology is unlikely to follow such a mathematical model.

As a result of this investigation, it seems possible to predict the modelling areas of a longwall face that may be subjected roof instability.

**Faulting**

One factor not included in the prediction model was the presence of faulting and its potential effect on the stability of the roof. Figure 9 shows the faulting map of longwall 108 overlaying the LVA data contour. The two arrows indicate where a fault has corresponded with a cavity event. The further right example is one case where it was not previously indicated by the prediction model. The left example did fall on a point which was indicated as relatively high risk according to the model. The green arrow is pointing to a relatively significant fault that extends through the width of the panel. At this point no cavities were recorded.
CONCLUSIONS

Longwall mining is the most common method of underground coal mining used in Australia today, with the method becoming ever more prevalent and adaptable to coal seams that were previously too difficult to mine. As such, determining the cause of stability issues that lead to compromised safety of employees and lost production time is of high priority to the coal industry.

This research project, conducted on the evaluation of cavity developments at Moranbah North Mine, aimed to determine the cause of roof instabilities and cavity developments that have plagued the mine site in the past. The knowledge gained from prior literature emphasised features of the coal seam and surrounding strata that contribute to roof instabilities and the development of cavities at the longwall face. Using these features a prediction model was developed, which aimed at accurately highlighting areas prone to instability within the roof and the development of cavities.

Indexes of the geological data were developed for the purpose of developing the prediction model. The geological data was scaled to a value between one and ten, to ensure no feature would overshadow the other contributing factors. Each of these geological factors was then compared with longwall pressure data collected using LVA software. The LVA data showed historical data collected as production on longwall 108 at Moranbah North mine progressed. From the pressure data, the areas where cavities had occurred during mining were able to be identified.

Little correlation was found to exist between the individual geological properties and known cavities that occurred along longwall 108, with the overburden and interburden showing the greatest correlation. The sandstone bodies showed little to no correlation. With this knowledge each parameter was weighted accordingly, and then all indexes were combined to produce the final prediction model.

Some correlation was identified between the LVA data and the prediction model developed using the geological data. Three of the known cavities occurred in areas that were highlighted as high risk areas in the prediction model. However, two cavities occurred in areas that were only considered a moderate risk. No cavities occurred in areas that were calculated to be low risk areas by the prediction model.

The prediction model failed to take into account faulting as a potential contributor to roof instability. The LVA contour was compared to a map that showed the significant faults that affected the longwall 108 panel. Two of the known cavities corresponded to faults that existed. However, one significant fault that spans along the width of the panel did not cause any roof instability issues. As these faults were known to the operators of the mine, some precautions may have been taken to alleviate any issues associated with this particular fault, however this fact is not certain.

It is suggested that the lack of correlation between the model and the pressure data may be a result of the thick seam mining method applied at Moranbah North mine. By leaving a thick layer of coal in the roof, a method proven to alleviate some instability issues when applying a thick seam mining method, higher lateral confinement stresses may have offset the pressures being exerted by the overlying strata. In addition, the potential for inaccuracies to exist in the contour plots of the coal seam and roof strata as a result of the distance between boreholes, could have contributed to the lack of correlation.

The research on developing a model for predicting longwall roof instability continues. The model will be further improved by incorporating the additional geological and operational parameters. It is hoped that
the new model will provide a better indication of where roof failures might occur for longwall mining operations.

ACKNOWLEDGEMENTS

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REFERENCES

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APPLICATION OF THE BRITTLE FAILURE CRITERION TO THE DESIGN OF ROOF SUPPORT IN THE SOFT ROCKS OF COAL MINES

Ross Seedsman

ABSTRACT: The bilinear brittle failure criterion that utilizes the unconfined compressive strength (UCS) and a spalling limit of 3.4, together with a tensile strength cut-off can be used to model the height of failure above coal mine roadways. Transverse isotropy can be incorporated into a continuum analysis by using a Young's Modulus/Independent shear modulus ratio of 15. To predict the height of failure, the two key variables are the roof strength index (UCS/pre-mining vertical stress) and the horizontal to vertical stress ratio. By factoring in the stress concentrations that occur about a longwall excavation the criterion can be used to predict heights of failure on initial roadway development and in the maingate. A support design based on dead-weight suspension of the failed mass can be utilised.

NOTATION

<table>
<thead>
<tr>
<th>CI</th>
<th>Competence Index</th>
<th>UCS/σv</th>
</tr>
</thead>
<tbody>
<tr>
<td>E</td>
<td>Young Modulus</td>
<td></td>
</tr>
<tr>
<td>Fa</td>
<td>Resolves principal horizontal stresses into stress acting across the roadway (Figure 10)</td>
<td></td>
</tr>
<tr>
<td>Fm</td>
<td>Concentration of horizontal stress at the maingate corner resolved across the roadway (Figure 12)</td>
<td></td>
</tr>
<tr>
<td>G</td>
<td>Independent shear modulus</td>
<td></td>
</tr>
<tr>
<td>Hmax</td>
<td>Maximum height of failure</td>
<td></td>
</tr>
<tr>
<td>K</td>
<td>σh/σv</td>
<td></td>
</tr>
<tr>
<td>Ki</td>
<td>K ratio before mining</td>
<td></td>
</tr>
<tr>
<td>L</td>
<td>Bolt anchorage length</td>
<td></td>
</tr>
<tr>
<td>Mv</td>
<td>Concentration of vertical stress at the maingate corner</td>
<td></td>
</tr>
<tr>
<td>RSI</td>
<td>Roof Strength Index</td>
<td>UCS/σv</td>
</tr>
<tr>
<td>UCS</td>
<td>Unconfined compressive strength</td>
<td></td>
</tr>
<tr>
<td>σ1,σ2</td>
<td>Major and minor principal horizontal stresses</td>
<td></td>
</tr>
<tr>
<td>σh</td>
<td>Horizontal stress applied to roadway in 2 dimensional model</td>
<td></td>
</tr>
<tr>
<td>σv</td>
<td>Vertical stress applied to roadway in 2 dimensional model</td>
<td></td>
</tr>
<tr>
<td>σvi</td>
<td>Initial vertical stress</td>
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</tbody>
</table>

INTRODUCTION

In the longwall mining method, two sets of roadways (gateroads) up to 3 km – 4 km long are driven in coal to define a block of coal that may be between 150 m and 410 m wide. The longwall is installed between the gateroads and can retreat at rates of up to 25 m/d. The maintenance of roof stability in these roadways is fundamental to safe and productive mining. The attainment of such stability is needed to ensure the business requirement for fast development rates. In the modern Australian mining context, maintaining a commitment to zero harm needs to simultaneously accommodate a roadway development rate of at least 50 m/d.

Most coal roof support design is either precedent/practice or empirically based using a rock mass classification scheme (Mark and Molinda, 2005; Colwell and Frith, 2009). A relatively dense bolting pattern is typically installed at the face and this is supplemented by longer tendons installed prior to the retreat of the longwall face. Despite these dense patterns, there are still notable roof failures which, in addition to the evident impacts on safety, also result in the longwall face being stood for in excess of several weeks. Many mines suffer from a development constraint whereby the next coal block is not ready and either the longwall face equipment needs to be stood for weeks or months, or blocks are cut short and coal reserves lost. A significant component of this constraint is the time spent installing roof support.

At the consistent mining rates required, the reactionary/remedial components of the observational method are not appropriate and there is a need for robust estimates of likely roof support densities early
in the planning stage, as well as in operations. There is a need for some simple tools that can provide such designs. Recent industry-funded research has sought to provide a more analytical approach to roof support design, one that is readily incorporated into mine planning and operations (Seedsman, Gorden and Aziz, 2009). The research identified three basic failure modes for coal mine roadways. First, there is the de-lamination of thinly-bedded strata close to the mining face. Secondly there is the onset of compressive failure of the roof if the induced stresses exceed the strength of the rock. Thirdly, there are situations whereby the vertical stresses are well in excess of the horizontal stresses (especially in coal) and the roof may fail in tension.

This paper discusses the compressive and tensile failure mechanisms and how a simple continuum numerical code can be used to assess different stress and failure conditions and to develop simple design charts.

ENGINEERING GEOLOGY AND GROUND CONTROL

The lithologies encountered in the roof of coal mine roadways include claystones, siltstones, sandstones and conglomerates. In thick coal seams, the roof may consist of coal. The UCS can range between less than 5 MPa to in excess of 100 MPa. Friction angles are in the range of 25° - 30°. The density of coal measure rocks is in the range of 2.4 to 2.5 t/m³, and for coal it may be in the range of 1.2 to 1.6 t/m³.

Rock masses in coal measures are dominated by the presence of ubiquitous bedding partings, and typically two joint sets aligned orthogonally. Joint spacing is typically equal to the spacing of bedding discontinuities (Hobbs, 1967) such that a valid model is a rock mass of cubes with dimensions from centimetres to metres. By virtue of the depositional environment and the subsequent diagenesis, the rocks are quartz silicates.

There is relatively little information published on the size of roof falls. In one shallow mine, the maximum height of roof falls is reported to be within 400 mm of the width of the roadway (4.8 m-5.2 m) and roughly triangular (Payne, 2007). There is a widely-accepted folklore that the maximum height of falls in deeper mines (up to 500 m) is no more than the width of the roadways. Underground operations collect a large amount of data on the vertical displacement in the roof based on multipoint extensometers installed close to the face at the roadway centreline (Figure 1). The point of zero displacement is referred to the height of fracturing or softening. As defined, the height of fracturing can be used as a proxy for possible roof fall height. One published database (Strata Engineering, 2001) has the maximum height of fracturing of 5 m above 5.2 m wide roadways and 6.5 m above 8.4 m wide roadways (Figure 2). O’Grady and Fuller (1992) reported a height of softening in excess of 7.4 m above a 4.9 m roadway aligned poorly to the horizontal stress field at 420 m depth.

Roof bolting may need to be specified against several different potential collapse mechanisms (Seedsman, Gorden and Aziz, 2009). Given the presence of bedding partings, there may be a need to reinforce the partings so that a thicker beam is created. If compressive or tensile failure is induced, the ground control strategy may need to be based on the suspension of the immediate roof from more stable ground. There is also a need to install skin restraint to control small scat.

The need to install long tendon support in addition to the standard roof bolts and mesh panels has been related to the ratio of the UCS of the rock to the far-field horizontal stress (Gordon and Tembo, 2005). To determine this ratio, the roof strength index (RSI) has been defined as the ratio of the UCS to the pre-mining vertical stress: this has the same formulation as the competence index used in tunneling (Muirwood, 1972). At the Kestrel Mine a value of 3.5 was found to indicate the need for long tendons during longwall retreat and 2.8 to indicate the need for long tendons during development.

ROCK STRENGTH CRITERION

The initial development of brittle criterion (Martin, Kaiser and McCreath, 1991) referenced the sandstones in the Donkin Morien tunnels accessing the coal seams of the Sydney Basin Nova Scotia. The 1991 version invoked a cohesion value of UCS/6 and a friction angle of zero degree (0°). The criterion was further extended to include the spalling limit (Kaiser, et al., 2000), with the implication that the lower limit for the spalling limit is the Mogi line of 3.4. The 3.4 limit applies to silicate rocks (Mogi, 1966) and it is noted the value is equivalent to a friction angle of 33°. A recent back analysis of the Donkin Morien tunnels (Seedsman, 2009) determined a spalling limit less than five, and a transverse anisotropy ratio of less than 20 for the case of a modulus ratio of two. A preliminary assessment for the
spalling limit of coal may be higher – 10. Because of the scale of the excavation and the nature of the rock mass, the failure criterion must also include an absolutely zero tensile strength. The available option is a Mohr Coulomb criterion with a tensile strength cut-off.

![Figure 1](image1.png)

**Figure 1** - Vertical movements above a 4.9 m wide roadway showing a height of fracturing of 4.9 m above a 4.9 m wide roadway (from Mark, et al., 2007)

![Figure 2](image2.png)

**Figure 2** - Compilation of height of fracturing (softening) for shallow longwalls (from Strata Engineering 2001)

Coal measure rocks are transversely isotropic, both by way of the preferred orientation of clay particles within the finer grained lithology and by bedding textures and bedding partings. This can be modelled in a continuum assumption by invoking transverse isotropy, although at the cost of only being able to
conducted elastic analyses. Published Young’s Modulus/Independent Shear Modulus (E/G) values for sedimentary rocks are in the order of 1-3 (Gerrard, Davis and Wardle, 1972), but these are for intact rock. For a bedded rock mass, higher values are possible and as will be discussed, values of between 15 and 20 are indicated from back analysis, with the simplifying assumption that the two Youngs Modulus values are equal. The inclusion of transverse isotropy is more important than the ability to model yielding.

In summary, an appropriate strength criterion for soft rocks in coal measures in the vicinity of an excavation shown in Figure 3 is:

- An E1/E2 ratio of unity,
- An E/G ratio of 15 - 20 is needed,
- SL of 3.4 is appropriate for stone,
- Tensile strength = 0.0,
- Cohesion = UCS/6,
- Friction angle of 0.0.

Figure 3 - Adopted failure criterion for coal measure rocks

FAILURE ZONES ABOUT A TYPICAL ROADWAY

The failure criterion discussed above is soundly based on recent applied research into the behaviour of excavations. It also offers great advantages in terms of developing simple design charts for the extent of rock failure. Failure develops when the minor stress is tensile, or when the deviatoric stress is greater than UCS/3, and then the maximum extent is given by a ratio of the principal stresses. This allows elastic stress analyses with only one material variable - the UCS. Simple boundary element codes (Examine 2D) can be used for specific openings, and the following discussion is based on a 5 m wide by 2.5 m roadway.

A further simplification is possible if the UCS is normalized to one of the principal stress components - in this case to the vertical stress. In the following, the normalization is referred to as the Competence Index (CI), which is defined as:

\[
CI = \frac{UCS}{\sigma_v}
\]

Where: CI = Competence Index, and
\(\sigma_v\) = vertical stress applied to the opening.

Figure 4 shows the distribution of failure zones for different CI values for the case of \(K = 1.3\) and \(E/G=15\), where \(K = \sigma_v/\sigma_h\), and \(\sigma_h\) = horizontal stress applied across the opening.

For the case examined, the failure zones are continuous across the roadway only for CI values less than 6.0. For higher CI values, the failure zones are localised near the sides of the excavation, and this may be the basis for the stress guttering reported from underground roadways. The failure heights cannot exceed those given by a spalling limit of 3.4 shown by the dashed line in Figure 4. The shape of the failure zones defined by the spalling limit is parabolic.
With a series of analyses it is possible to generate design charts that give the height of failure at the centreline of the roadway for different K ratios and CI values. Different charts can be produced for different E/G ratios as shown in Figures 5 and 6. Because the analyses are elastic, it is possible to normalize for different widths if the aspect ratio of the roadway is the same. For different aspect ratios, there is a need to conduct separate analyses.

Close inspection of the Figures 5, 6, and 7 reveals that there is little difference in the heights of failure controlled by the CI for the various E/G ratios, but a large difference for the heights controlled by the spalling limit in Figure 7. The appropriate value to use for design can be determined by back analysis.
Figure 6 - The height of failure above a 5 m by 2.5 m roadway as a function of the K ratio and the competence index (a: Isotropic, b: E/G=10)

Figure 7 - Sensitivity of failure height to the E/G ratio when the spalling limit controls the onset of failure

For low values of K, there is no compressive failure in the roof as the immediate roof stresses may be tensile. Recognising that the joints are vertical, the zone of the tensile horizontal stress component may indicate where vertical shear along joints may develop as shown in Figure 8. The size of the tensile zone does not change with increasing stress magnitudes, while the magnitude of the tensile stresses within the zone does increase. The tensile zone vanishes for K ratios in excess of approximately 0.55 shown in Figure 9; at a K ratio of 0.2, the zone is about 1 m high and 4 m wide.

Figure 8 - Distribution of tensile horizontal stresses for K=0.3 (5 m by 2.5 m opening)
Underground coal mining in Australia is conducted at depths as low as 50 m and in excess of 500 m and in excess of 1000 m in other countries. For gently dipping coal seams, the vertical stress in stone can be considered to be a principal stress with its magnitude related directly to the depth of cover. The in situ stress field in stone is characterised by Ki ratios that are close to unity in some European coal fields to in excess of 2 in some Australian coal fields (Mark and Gadde, 2010). For the coal seams themselves, the situation is different, with Ki values less than 0.2 measured (Seedsman, 2004). At depths, in excess of about 250 m, the direction of the major principal horizontal stress may be related to the latest tectonic events. Different alignments can be expected at shallower depths or near major fault structures.

Development roadways

Mine roadways are rarely aligned parallel to one of the principal horizontal stress directions. The two dimensional failure model discussed above requires that the component acting normal to the roadway centreline is used (Figure 10). The Fa factor is applied to the ratio of the in situ major horizontal/vertical stresses to determine the K ratio acting across the roadway (K=Ki * Fa). Immediately after the roadway is formed, the deformations in the immediate roof (bedding dilations, brittle fracturing) cause the roof to “soften” and the horizontal stress field is redirected higher into the roof (Mark, et al., 2007).
Maingate

At the maingate corner at Position a, in Figure 11, during the retreat of the longwall, there are large changes in the stress regime. Based on numerous pillar stress instrumentations, longwall pillar design methods indicate an approximate doubling of the vertical stress at the maingate (Mv=2) (Mark, 1991; Colwell, 1996). Gale and Matthews (1992) not only reported similar data on the vertical stresses, but also presented a summary of the concentration of horizontal stresses. At heights about 5 m-10 m above the roof line there can be up to a doubling of the horizontal stress magnitudes if the roadway is aligned 45-55° to the principal stress direction as shown in Figure 12a.

![Diagram of longwall nomenclature](image)

**Figure 11 - Nomenclature of a longwall**

The presentation in Figure 12a is not directly applicable to the two dimensional model presented above as it requires the magnitude of the stress acting across the roadway. It is known that during longwall retreat extensive fracturing develops, not only above the extracted seam, but also into the floor. The frequent reports of gas emanating from lower seams (up to 40 m below) indicate that fracturing extends at least that far. On this basis, it is valid to assume that plane strain conditions exist at the coal seam level and hence simple two dimensional models can be used to assess the stress field. By selecting an offset distance from an excavation that gave similar stress concentration patterns as shown in Figure 12a, a nomogram has been produced to allow the determination of the component of the induced horizontal stresses acting normal to the roadway centerline as a function of the orientation of the roadway and the ratio of the principal horizontal stresses, see Figure 12b. Using this figure, the K ratio, resolved across the roadway is given by the following relationship:

\[ K = K_i F_{m}/M_{v} \]  

(2)

![Graph showing stress concentration factors](image)

**Figure 12 - Concentration of horizontal stresses at the maingate corner (a:-principal stress, b: stress factor for component normal to roadway centreline - Fm)**
Tailgate

At the tailgate, Position c, in Figure 11, there are further increases in the vertical stress, but a reduction in the horizontal stresses due to proximity of the adjacent previously extracted longwall. The end result is a reduction in the K ratio. Tailgate stresses have not been extensively monitored so quantification of this impact is not yet possible.

THE E/G RATIO IN COAL MEASURE ROCKS

A number of published case studies have been investigated to determine the appropriateness of the design charts and to obtain an appropriate estimate for the E/G ratio. Table 1 indicates that an E/G ratio of 15 provides a good estimate of the height of fracturing above rectangular roadways. The Donkin case study has been reanalysed using a modulus ratio of unity and a E/G ratio value between 15 and 20 was obtained.

Table 1 - Summary of case studies for failure height

<table>
<thead>
<tr>
<th>Mine</th>
<th>Crinum – 4.8 m roads</th>
<th>Emerald - 4.9 m roads</th>
<th>Tahmoor - 4.9 m roads</th>
</tr>
</thead>
<tbody>
<tr>
<td>Depth</td>
<td>120-200 m</td>
<td>Approximately 200 m</td>
<td>420 m</td>
</tr>
<tr>
<td>Vertical stress</td>
<td>3-5 MPa</td>
<td>6 MPa measured</td>
<td>10.5 MPa</td>
</tr>
<tr>
<td>UCS</td>
<td>8-10 MPa</td>
<td>17.7-36.4 MPa</td>
<td>48 MPa</td>
</tr>
<tr>
<td>RSI (development)</td>
<td>6 – 2</td>
<td>3.5-7.3, average 4.5</td>
<td>4.6</td>
</tr>
<tr>
<td>(\sigma_n/\sigma_v)</td>
<td>1.6-1.2</td>
<td>0.75</td>
<td>1.92, 1.29</td>
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<tr>
<td>(\sigma_n/\sigma_m)</td>
<td>Measured stresses</td>
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<td>0.67</td>
</tr>
<tr>
<td>Angle</td>
<td>45</td>
<td>Measured stresses</td>
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<tr>
<td>Fa (Figure 10)</td>
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<td>Measured stresses</td>
<td>0.98</td>
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<tr>
<td>(K_r) ratio</td>
<td>(=1.6^{*}0.87=1.4)</td>
<td>1.67 – Stage 1</td>
<td>=1.92*0.98 = 1.88</td>
</tr>
<tr>
<td>Estimated height</td>
<td>5.0 m for rocks (C_l=2) if (E/G=15), or 5.7 if (E/G=20)</td>
<td>3.3 m if (E/G=15)</td>
<td>5 m if (E/G=15)</td>
</tr>
<tr>
<td>MV</td>
<td>2</td>
<td>2</td>
<td>2</td>
</tr>
<tr>
<td>Cl (maingate)</td>
<td>2.25</td>
<td></td>
<td>2.3</td>
</tr>
<tr>
<td>Fm (Figure 12b)</td>
<td>Measured stresses</td>
<td></td>
<td>1.9</td>
</tr>
<tr>
<td>Estimated height</td>
<td>4.2 m if (E/G=15), 5.3 m if (E/G=20)</td>
<td>6.6 m if (E/G=15), or 7.5 m if (E/G=20)</td>
<td>7.2 m above a 4.9 m roadway</td>
</tr>
<tr>
<td>Comment</td>
<td>Maximum heights on development within 0.3 m of 4.8 m</td>
<td>3.3 m in stage 1, 4.2 m in stage 2, and 4.9 m in stage 3</td>
<td>7.2 m above a 4.9 m roadway</td>
</tr>
</tbody>
</table>

APPLICATION TO PREVENTING ROOF COLLAPSE IN LONGWALL GATERoads

Roof bolting in Australian mines is conducted from platforms located on the continuous miner about 2.5 m to 3.0 m from the face. The range of bolt locations and possible angles is limited by machinery constraints. Typically 1.8 m or 2.1 m long roof bolts (34 t ultimate tensile capacity) are installed vertically in 27 mm diameter holes using resin grouts. About 0.1 m of the bolt is not inserted into the drill hole. A typical 6 bolt pattern is shown in Figure 13. Depending on ground conditions, 4 m to 8 m long tendons with tensile capacities of around 60 t are used as supplementary support – sometimes these are installed off the miner, or they can be installed at a later stage of the mining process.

Suspension of the failed mass from the un-failed zone higher in the roof is an appropriate basis for support design. Assuming a parabolic shape to the failure zone, it is possible to determine the weight of the block and then to specify the required bolt or tendon length.

For convenience, the following relationships are used: \(RSI = \text{UCS}/(\text{Depth}*\text{density})\) \(=R\) (3)
For development roadways, \(\text{Cl}= RSI, K = K_i*F_a\) \(=K\) (4)
For maingates, \(\text{CSI} = \text{RSI}/M_v, K = K_i*F_m/M_v\) \(=M\) (5)
Where: \(K_i\) is the ratio of the horizontal to vertical pre-mining stresses,
\(F_a\) is a factor to determine the stress acting normal to the roadway direction from Figure 10,
\(F_m\) is a factor to account for the concentration of horizontal stress (Figure 12) and,
\(M_v\) is a factor to account for the increase in vertical stress at the maingate.
Based on the approach of Littlejohn (1993) the minimum anchorage length can be readily estimated from the relationship:

\[
\text{Anchorage (m) } = \frac{0.4}{\text{UCS (MPa)} \times \text{bolt load (t)}}
\]

The longer tendons may be installed in larger diameter holes, so a separate anchorage relationship would be required.

**Geometry of failure zone**

The maximum height of failure can be selected from the E/G =15 chart in Figure 7. The parabolic shape means that the cross sectional area of the failure is given by \(0.67 \times H_{\text{max}}^5\), and the height as a function of distance across the roadway is shown in Figure 13. For this failure mode, the key bolts are those located nearer the side of the roadway as the more centrally located bolts may be overridden by the failure.

![Figure 13 - Typical bolting pattern compared to the geometry of a parabolic shaped failure](image)

**Worked examples**

Table 2 list the steps in the calculation, with the focus being the two bolts installed 2 m from the centreline or 0.5 m from the rib. The parameters used in Table 2 are based on the Gordon and Tembo (2005) observations from Kestrel Mine whereby an RSI of 3.5 indicated the need to install long tendon support in the maingate prior to longwall retreat, and that a RSI of 2.8 indicated the need to install long tendons during development. The UCS values were estimated from the average sonic velocity over the first 2 m of stone roof. The bolting pattern was 6 by 2.1 m long bolts/metre.

**Practical considerations about suspension design**

Some cautions are necessary if this model is to be used. First, there are other failure modes in the roof which must also be considered. A design against compressive failure is necessary but may not be adequate for safe outcomes. In particular, the potential delamination of thinly bedded roof must be addressed and this may require more bolts acting in a bedding reinforcement mode. Secondly, reduction in bolting densities to just the two outer bolts is not recommended – the other bolts have a role to play in developing some bridging action onto the anchoring bolts.

Of particular importance is the possibility that the roof may unravel around the tendons. This is particularly the case for the tensile zones associated with low K values. Consideration needs to be given to how the loads are transferred from the centre of the roadway to the anchoring bolts. While the tensile capacities of the anchors are high (34 t to in excess of 60 t) the mesh through which they are installed fails at about two tonnes (Thompson, 2004).

Decisions on the length of the bolts and long tendons need to acknowledge the shape of the failure zone. Support elements installed towards the roadway centreline need to be longer so that they are anchored above the failure zone – a more efficient design may involve shorter tendons installed in pairs and fanned outwards. It would be advisable to install bolts at least 0.5 m from riblines to avoid encountering the possibly highly broken rock at the roof/rib corners shown in Figure 14.
Table 2 - Calculations for a suspension strategy for roof stability based on reports from Kestrel

<table>
<thead>
<tr>
<th></th>
<th>Kestrel 300 series</th>
<th>Kestrel 300 series</th>
<th>Kestrel 300 series</th>
<th>Kestrel 200 series</th>
<th>Kestrel 100 series</th>
</tr>
</thead>
<tbody>
<tr>
<td>UCS (MPa)</td>
<td>26</td>
<td>23</td>
<td>18</td>
<td>23</td>
<td>15</td>
</tr>
<tr>
<td>Depth (m)</td>
<td>260</td>
<td>260</td>
<td>260</td>
<td>230</td>
<td>230</td>
</tr>
<tr>
<td>RSI</td>
<td>4</td>
<td>3.5</td>
<td>2.8</td>
<td>4.0</td>
<td>2.6</td>
</tr>
<tr>
<td>$\sigma_u/\sigma_r/\sigma_{s1}/\sigma_{r1}$</td>
<td>1.8, 1.2, 0.75</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Angle, Fa</td>
<td></td>
<td>10°, 0.75</td>
<td></td>
<td></td>
<td>30°, 0.81</td>
</tr>
<tr>
<td>CI</td>
<td>4.0</td>
<td>3.5</td>
<td>2.8</td>
<td>4.0</td>
<td>2.6</td>
</tr>
<tr>
<td>$K = Fa * Ki$</td>
<td>1.2</td>
<td>1.2</td>
<td>1.2</td>
<td>1.3</td>
<td>1.3</td>
</tr>
<tr>
<td>$H_{max}$ (development)</td>
<td>1.4</td>
<td>2.0</td>
<td>4.0</td>
<td>2.0</td>
<td>4.7</td>
</tr>
<tr>
<td>Weight (t)</td>
<td>11.7</td>
<td>16.8</td>
<td>33.5</td>
<td>16.8</td>
<td>39.4</td>
</tr>
<tr>
<td>L (at 2.0m)</td>
<td>0.5</td>
<td>0.72</td>
<td>1.44</td>
<td>0.72</td>
<td>1.69</td>
</tr>
<tr>
<td>Anchorage length (m)</td>
<td>0.09</td>
<td>0.15</td>
<td>0.38</td>
<td>0.15</td>
<td>0.54</td>
</tr>
<tr>
<td>Required bolt length (m)</td>
<td>0.70</td>
<td>0.97</td>
<td>1.92</td>
<td>0.97</td>
<td>2.33</td>
</tr>
<tr>
<td>Comment</td>
<td>2.1 m bolt length adequate</td>
<td>2.1 m bolt length adequate</td>
<td>Marginal bolt length if anchorage is not ideal</td>
<td>2.1 m bolt length adequate</td>
<td>2.1 m bolts not adequate</td>
</tr>
<tr>
<td>CI (maingate)</td>
<td>2.0</td>
<td>1.75</td>
<td>1.4</td>
<td>2.0</td>
<td>1.3</td>
</tr>
<tr>
<td>Fm</td>
<td>Major stress concentrated, 1.4</td>
<td>Minor stress concentrated, 1.47</td>
<td>Major stress concentrated, 1.58</td>
<td></td>
<td></td>
</tr>
<tr>
<td>$K$ (maingate)</td>
<td>$K_i * F_m / M_v$</td>
<td>1.12</td>
<td>1.18</td>
<td>1.26</td>
<td></td>
</tr>
<tr>
<td>$H_{max}$ (maingate)</td>
<td>4.1</td>
<td>4.1</td>
<td>4.1</td>
<td>4.2</td>
<td>4.6</td>
</tr>
<tr>
<td>Weight (t)</td>
<td>33.5</td>
<td>35.2</td>
<td>35.5</td>
<td>35.2</td>
<td>35.5</td>
</tr>
<tr>
<td>L (at 2.0m)</td>
<td>1.44</td>
<td>1.51</td>
<td>1.66</td>
<td>1.51</td>
<td>1.66</td>
</tr>
<tr>
<td>Anchorage (m)</td>
<td>0.26</td>
<td>0.30</td>
<td>0.38</td>
<td>0.31</td>
<td>0.52</td>
</tr>
<tr>
<td>Required bolt length (m)</td>
<td>1.8</td>
<td>1.84</td>
<td>1.91</td>
<td>1.92</td>
<td>2.28</td>
</tr>
<tr>
<td>Comment</td>
<td>Marginal bolt length if anchorage is not ideal</td>
<td>Marginal bolt length if anchorage is not ideal</td>
<td>No major change in support requirements</td>
<td>Marginal bolt length if anchorage is not ideal</td>
<td>No major change in support requirements</td>
</tr>
</tbody>
</table>

Figure 14 - Common roof deformation pattern is consistent with outer bolts being the only ones anchored

Finally there are a number of uncertainties involved in the quantification of various parameters. A number of judgment calls are required: the selection of the average strength of what is always layered roof, the confidence in the stress model and particularly Figure 12b, and also the selection of a factor of safety value to apply to the anchorage calculation.

CONCLUSIONS

The brittle failure criterion can be applied to coal measure rocks with the impact of bedding incorporated by invoking transverse isotropy through an E/G ratio of 15. With currently available software, a
convenient implementation is by the Mohr Coulomb criterion with a friction angle of 0.0 and a tensile strength cut-off of zero. The simplicity of the criterion allows normalization of some of the variables and the production of design charts.

A support design approach based on the suspension of failure zones has been developed. The two key data sets are the roof strength index (RSI) which can be readily determined in mine exploration programs and the nature of the stress field. Because of the simplicity of the failure criterion, back analysis of previous failures can be used to refine the estimates of some of the key parameters.

Suspension of failed roof is a valid approach to design. When applied to existing case studies, the validity of the approach is strongly supported. Further considerations leads to the recognition of the importance of having a better tensile member across the roof line. This may be a serious weakness in current roof support designs.

REFERENCES


WHY DEAD LOAD SUSPENSION DESIGN FOR ROADWAY ROOF SUPPORT IS FUNDAMENTALLY FLAWED WITHIN A PRO-ACTIVE STRATA MANAGEMENT SYSTEM

Russell Frith

ABSTRACT: Risk-based roadway roof support design is now a critical part of the Australian Coal Industry. Safe and efficient mining demands that roof support be tailored to the prevailing geotechnical conditions and legislation in both NSW and QLD is clear in requiring that formalised roof support design be undertaken.

The assumption of dead-load suspension of an otherwise unstable roof has been used in roadway roof support design for many years. However longevity does not necessarily translate into either best or even reasonable practice in the current industry, particularly when pro-active strata management systems are now routinely used and reinforcing support design methods are available.

The paper discusses why the assumption of dead-load suspension is fundamentally incorrect in almost all instances when pro-active or reinforcing roof support is being applied and how it can easily result in misleading and potentially under-designed roof support systems under various circumstances. The commonly used design methodology of balancing the installed axial capacity of long tendons with the assumed “weight” of a future roof fall is summarised and several fundamental flaws are identified. The critical area of selecting the design Factor of Safety is also discussed in detail.

Accepting that under certain circumstances the roof of a roadway will require to be “suspended”, suggestions for how a robust support system can be developed, designed and applied are given.

INTRODUCTION

How many times has a roadway roof fall occurred, the long tendons broken as a direct result and the back-analysis concluded that there must have been something faulty with the steel due to the dead-load of roof material that fell out being insufficient to fail the installed tendons? In other words the question is asked as to why the roof was not stabilised by roof support that appeared to have more than sufficient carrying capacity to fully suspend the roof material that fell in. This type of roof fall back-analysis is almost always based on a basic roof suspension design that inevitably leads to a misleading conclusion about the primary causes of the fall and in the same way, if used for roof support design in the first place is equally flawed. This paper explores the flaws in such a design process and cautions the industry about its continued use.

COMMONLY PRACTICED SUSPENSION DESIGN

Suspension design is commonly practiced in roadway roof support design due to the perception that (a) it is relevant and (b) that it is straightforward (which it isn’t). Furthermore until more recently the availability of design methodologies that directly address roof reinforcement have been limited. Faced with a legislative requirement to undertake roof support design and with few choices in terms of alternative methodologies, it is easy to understand why suspension design is commonly practiced. However, common use and longevity do not demonstrate that such a design methodology is appropriate though, simply that it is popular.

Suspension design is a key part of the design methodology put forward by Seedsman et al. 2009. In three of the four defined collapse modes for roof instability (i.e. non-vertical joints, compressive failure and tensile failure) the recommended method of support design is that of suspending a dead load. Furthermore as part of his current ACARP Project on the design of roof support in wide roadways, Colwell 2010 confirmed that suspension design is commonly practiced by mine site geotechnical personnel and also some geotechnical consultants for primary and secondary roadway roof support.
Lastly Canbulat 2010 when discussing roof support design practices within Anglo American’s Metallurgical Coal’s operations in Australia lists design against “suspension failure” as one element of the overall process. The following quotations are taken from Canbulat 2010:

“The suspension mechanism is the most easily understood roof bolting mechanism. The design of roof bolt systems based on the suspension principle has to satisfy the following requirements (Canbulat and van der Merwe 2009):

- The strength of the roof bolts and/or long tendons have to be greater than the relative weight of the loose roof layer that has to be carried.
- The anchorage forces of the roof bolts have to be greater than the weight of the loose roof layer”.

Based on the above it is clear that the use of suspension theory for roadway roof support design is currently “endemic” to the Australian Coal Industry at both a mine site and consulting level. Therefore it is fully appropriate to critique the manner in which the method is being used as part of assisting the on-going process of geotechnical design improvement.

DEFINITION OF THE CURRENT DESIGN APPROACH

It is clear that the basic suspension design methodology in common use in the coal industry utilises the following generic equation:

$$\text{Factor of Safety (FoS)} = \frac{\text{carrying capacity of the installed support}}{\text{weight of the dead load that needs to be suspended}}$$

At a basic level this makes sense and is appropriate to solving a simple suspension problem as illustrated in Figure 1.

![Figure 1 - Simple suspension problem involving a dead load and a vertical restraint](image)

In the simple suspension problem shown in Figure 1 the load of the hanging weight is fixed (and presumably known) and the suspending capacity of the restraint is its axial strength. With these two parameters being readily defined, an outcome can be designed according to whatever Factor of Safety is deemed appropriate for use.

In reality this appears to be no different to the manner in which suspension design for roadway roof support is currently being undertaken in the Australian Coal Industry. It certainly fits exactly with the description provided earlier according to Canbulat 2010 and is commonly observed in numerous third-party roof support design reports.

Unfortunately the design problem is nowhere near as simple as that shown in Figure 1 when it is applied to roadway roof support. The following subject areas require further consideration:
• The weight of the dead load to be suspended is not necessarily a constant but can vary over time when for example secondary extraction is considered and the ground stresses acting will inevitably change.

• Not all suspending support elements are installed vertically (e.g. cable slings or angled long tendons – see Figure 2) hence they are not solely loaded in axial tension but will be subjected to both axial and shear components.

![Figure 2 - Schematic illustration of suspension roof support using inclined long tendons](image)

• On the assumption that secondary roof support in the form of either tendons or cables is installed prior to the roof being in the fully failed state that requires suspension (which it must be otherwise it would not be safe to install such support), such support will inevitably be subjected to additional stress-driven ground movements before the roof reaches a fully failed and critically unstable state, these movements including the development of horizontal shear along bedding planes.

• No industry guidance is available as to what may constitute a suitable design Factor of Safety which is particularly concerning given that the result of an inadequate design is by definition a major roof collapse with attendant safety and business risks.

Each of these subject areas will now be discussed in more detail as part of examining the true complexities and limitations involved in undertaking credible roof support design using suspension theory.

**DETERMINING THE SIZE AND WEIGHT OF THE UNSTABLE ROOF BLOCK**

The first and most obvious problem to be solved is in assigning a credible value to the size and hence the weight of the unstable roof block that needs to be suspended. It is evident that several different approaches can be applied to this aspect of the design process.

The size of the unstable roof block is defined by a span, height and in some cases a shape. The span is relatively straightforward as it is defined by the roadway width plus any large scale rib spall that may have occurred when the roof enters its design condition.

The shape of the roof block is one of either vertical or non-vertical sides, resulting in either a rectangular roof block as illustrated in Figure 2, a triangular or trapezoid block as shown in Figure 3 based on joint inclinations (after Seedsman, et al 2009) or the development of a roof failure “arch” as shown in Figure 4 (after Payne, 2008).

Some practitioners utilise the “Height of Softening” method whereby roof extensometry or borescope observations are used to physically measure or infer the height of a potentially unstable roof block. Alternatively Seedsman et al. 2009 provide basic equations whereby the likely height of roof failure is linked to such parameters as the ratio between the horizontal and vertical stress (K) and the Roof Strength Index or RSI. Obviously in some circumstances the roof strata may contain a unit that acts as a
“capping band” such that roof failure is unlikely to progress any higher than its lower limit. This also allows a likely roof failure height to be determined.

![Diagram](image)

**Figure 3 - Non-parallel joints defining triangular prisms (after Seedsman et al., 2009)**

The main problem area identified in this paper is in determining whether the height of roof failure is likely to increase between the time that suspension support is installed and when the roof is in its final designed condition (for suspension). An increase in the height of roof failure could occur as a result of either ground stress changes as a result of secondary extraction (e.g. corner of the MG belt road) or roof geometry changes due to roadway widening (e.g. longwall installation roadway).

There are various methods by which the height of a failed roof block can be determined; however when subsequent changes in ground stress or roadway geometry are considered it is clearly not as straightforward as is commonly portrayed.

The remainder of the paper will focus on the far more complex and relevant issues of determining a realistic carrying capacity for a tendon system that is required to suspend the roof and defining an appropriate design Factor of Safety.

**ANALYSING AXIAL AND SHEAR LOADS IN INCLINED TENDONS**

The use of long tendons for roof suspension purposes angled at say 20° to the vertical (as illustrated in Figure 2) will not allow the full axial capacity of each tendon to be generated as vertical support resistance (which is what is required for suspension design). Each tendon will inevitably be loaded both axially and in shear if it crosses a vertical shear plane in the roof and is therefore deformed by vertical shearing. If a shear component is generated in the tendon the available axial capacity is reduced by some amount and this is well catered for in general reinforcement design for concrete as will now be detailed.
Figure 5 (from Simpson Anchors 2010a) shows the combined “allowable stress” condition for a structural element that is loaded both axially and in shear. The combined allowable stress condition is given by the following equation as found in Simpson Anchors 2010b:

\[(T / T_{all})^n + (V / V_{all})^n \leq 1.0\] (2)

Where:
- \(T\) = tension or axial load being applied;
- \(T_{all}\) = maximum allowable tension in the element;
- \(V\) = shear load being applied;
- \(V_{all}\) = maximum allowable shear in the element;
- \(n\) = constant that determines the shape of the allowable stress condition line (see Figure 5).

What Figure 5 and the above equations indicate is that if there is no shear on the element, then the full axial strength can be utilised. Conversely if there is no axial tension on the element then the full shear strength can be utilised. However if both axial and shear loads are being generated, neither the full axial nor shear strength can be generated in the tendon as it will inevitably undergo yield and/or failure as a direct consequence.

**Figure 5 - Combined allowable stress condition for axial and shear loading (after Simpson Anchors, 2010a)**

In relation to Equation 2, Simpson Anchors 2010b state the following:

“For anchors subjected to simultaneous tension and shear loading, the following Equation (2) must be satisfied where the value of \(n\) is product-specific. Use a value of \(n = 1\) unless otherwise specified in the applicable products load table”.

Equation 2 will now be used assuming \(n = 1\) to illustrate the reduction in vertical load-carrying capacity of a tendon inclined at only 20º to the vertical (refer Figure 6 which provides for a graphical illustration).

**Figure 6 - Work1ed example in graphical format**
At an inclination of 20° a tendon being subjected to vertical shear movement (as is likely if the roof has become a dead-load and is moving downwards under self-weight) will generate 0.364 t of shear load for every 1 t of axial load, this being governed by tan20°. This is plotted on the graph in Figure 6 to show the developing axial and shear load condition in the tendon as the roof continues to move downwards.

Assuming the use of current day long tendons with an axial capacity of 60 t and a shear strength of around 50% of this at 30 t, the allowable stress condition (in absolute terms) can also be plotted on Figure 6 (noting that no strength reductions have been applied to account for what are useable axial and shear capacities in practice as compared to the rated ultimate strengths). Where the two lines intersect is the maximum allowable stress condition in terms of both axial and shear loading. In this particular example it is evident that this is defined by an axial load of 34.7 t and a shear loading of 12.6 t.

When these values are substituted back into Equation 2 it is found that:

\[(12.6 / 30)^1 + (34.7 / 60)^1 = 1\]

This is as expected and represents the absolute highest combination of axial and shear loading for a tendon inclined at 20° to the vertical when being loaded in vertical shear.

Referring back to Figure 6 these two load components can be combined to find the resultant maximum vertical loading of the tendon using simple trigonometry, as follows:

maximum vertical loading = \((34.7^2 + 12.6^2)^{0.5} = 37\) t.

In other words by inclining the long tendon by only 20° off the vertical, the impact of shear in the tendon reduces the vertical load-carrying capacity from 60 t to 37 t. This is a highly significant reduction yet is rarely if ever considered in suspension design analysis in the Australian Coal Industry.

Clearly then if the roof of a roadway is required to be suspended using long tendons, inclining them by as little as 20° in order to anchor them over the adjacent coal pillars (as shown in Figure 2) which is common practice, significantly reduces their vertical load carrying capacity. This must be brought into the suspension design process if the resultant FoS value is to have any credibility and risk-based meaning.

STRESS DRIVEN GROUND MOVEMENTS FOLLOWING TENDON INSTALLATION

Following on from the discussion on inclining the tendons and inducing shear across the tendons, the same effect can also occur if the tendons are installed early in the roof deterioration process (as is commonly the case when pro-active strata management is being applied) whereby the roof has yet to fully fail (and so become a dead-load problem) and additional external stress driven roof movements are likely to occur and so induce strain in the tendons.

Seedsman et al. 2009 when discussing “delamination failure” as a roof failure mechanism state the following:
"As the roadway is formed, shear can develop along bedding partings, the bedding may open and individual beams of rock develop".

They illustrate this using Figure 7 which is taken from Brady and Brown 1985 and demonstrates that as a roof delaminates and displaces downwards (whether it be by horizontal stress driven buckling or vertically driven bending), bedding plane shear movement is inevitable across all of the roof except the very centre (in a symmetrical bending or buckling profile).

The phenomenon of bedding plane shear can also be reproduced in physical models as shown in Figure 8. This is a snapshot taken from a foam model used by the University of New South Wales (UNSW) to illustrate the phenomenon of roof buckling under the action of horizontal stress. Horizontal shearing along bedding planes is clearly evident.

![Figure 8 - Snapshot of UNSW foam model illustrating bedding plane shear associated with stress driven roof buckling](image)

Furthermore, anyone who has ever installed a sonic probe extensometer can attest to the frustrating loss of the unit due to shearing of the hole as a result of excessive vertical roof displacement.

The point is that unless suspension tendons are installed at a time when the roof has largely failed and all future movement is purely vertical and driven by self-weight (an example of which will be provided later), they will inevitably be subjected to horizontal shear movement as a result of subsequent stress driven buckling of the roof measures. By the same argument as that presented in the previous section, such shear will reduce the axial load-carrying capacity of those tendons (see Figure 5).

This then explains the statement made at the start of the paper as to why when a roof fall occurs and long tendons are observed to have broken, the suspension back-analysis fails to explain why those tendons have broken and in fact usually indicates that they should not have broken. This inevitably leads to the questioning of the quality of the steel used in the tendons and commences a process of investigation that is almost certainly not warranted in the first place.

**A SUITABLE DESIGN FACTOR OF SAFETY**

Design Factor of Safety is not a geotechnical consideration but one of an acceptable level of risk based on the design being inadequate. Determining acceptable levels of risk involves mine management in addition to the geotechnical engineer. Therefore it is appropriate herein to at least provide discussion on the subject.

In terms of defining a suitable Factor of Safety for an engineering design, one has to consider the consequences of the design being inadequate. For example a major pillar collapse in an underground coal mine has the potential to result in significant loss of life, as was the case at the Coalbrook Mine in South Africa in 1960 (Van der Merwe, 2006). Similarly the business consequences can also be very
severe. As a result it is common for long-term and mining-critical coal pillars to be designed at failure probabilities in the order of one in one million.

With suspension design for roadway roof support, the consequence of an inadequate design is a major roof collapse as the support is effectively the “last line of defence”. If the roof is failed and suspended by long tendons and the tendons fail, a major roof collapse is inevitable. In this regards it is interesting to note that at least one major mining house defines “not working under a suspended load” as one of its cardinal safety rules. Whilst this rule was obviously intended to apply to suspended heavy loads in workshops when being lifted by cranes etc. it does at least mirror the obvious safety risks associated with an inadequate roof suspension design in mine roadways where men are working on an on-going basis.

It could be argued that an appropriate standard for determining a suspension design Factor of Safety is found in cranes and hoists whereby heavy loads that are otherwise free to fall are suspended and in some cases such as elevators, persons are directly exposed to the consequences of a design failure. However this would result in design Factors of Safety in excess of 5 being applied which would clearly be costly and uneconomic. The author is not suggesting that the industry necessarily follow this approach, but is simply pointing out the sorts of considerations that would be required for a suspension design to conform to some form of relevant and accepted design standard.

When roadway roof support is designed for reinforcement then the consequences of a design failure is typically the triggering of a TARP rather than a major roof collapse. As a result the design FOS can potentially be significantly lower than in the case of a suspension design. This issue should be considered when selecting a roof support design methodology.

SUGGESTED LIMITS OF SUSPENSION DESIGN

Taking into account everything stated previously in the paper, the one example whereby a suspension design is almost certainly appropriate is where the roof has deteriorated to the point that heavy standing support has been installed but the standing support now needs to be removed for mining reasons. If the roof has already moved say 200 mm to 300 mm and the mine has installed heavy standing support to stabilise the roof it is perhaps reasonable to assume that the roof is now in a failed state such that the design and application of a suspension system is the correct control approach. Figure 9 illustrates the inferred condition of the roof when it has been allowed to deteriorate to such an extent.

Figure 9 - Schematic of failed roof and flanking sub-vertical shear zones

If it is accepted that little can be done to increase the horizontal stress acting across the sub-vertical shear zone to stabilise the roof, and that even if the horizontal stress could be increased, it might only further exacerbate the condition of the roof material (i.e. cause more fracturing), there are only two control options available:
• to reduce the shear stress acting along the plane by holding up part or all of the roof that would otherwise fall in (i.e. roof suspension)

• to improve the shear characteristics of the surface by filling voids with material (e.g. grout) that has significant shear strength that can be mobilised without relying on horizontal confining stress (i.e. strata consolidation)

As well as the control of broken roof material, other instability mechanisms may also eventuate during longwall retreat for example that may exacerbate an already difficult situation:

• Due to voids within the roof, increased roof buckling and softening may occur higher up in otherwise currently stable strata units. This would inevitably reduce overall roof stability and increase the load to be suspended.

• Stress-induced shear movements may cause additional shear across the tendons thus reducing their available carrying capacity. Therefore tendon load-carrying capacities that were assumed as part of suspension support designs may be compromised in practice.

The most effective solution to both of these mechanisms is logically void filling and re-consolidation of the failed roof mass. Higher softening can only occur if there is a void for the roof to buckle into, therefore by filling such voids with competent material, the potential for further roof buckling and softening during retreat is significantly reduced. Similarly shear movements occur along pre-existing planes of weakness in the rock mass, particularly voids which have no shear strength. Therefore filling such voids reduces the potential for further shear within the roof, thus providing a level of protection to any tendons being used for suspension purposes.

From a design perspective both control approaches of suspension and consolidation can be analysed, the problem being that it is difficult to be certain that the designs have either been implemented as intended (i.e. grout migration cannot be fully defined) or the assumed support capacities are achievable in practice (due to such uncertainties as tendon anchorage and the reduction of tendon capacity through strata shearing effects).

The most reliable approach in this situation is to apply both suspension (using vertical tendons) and consolidation methods as this provides a level of redundancy in the design outcomes. Therefore any implementation uncertainties are far less significant and the combined stabilising effect of void filling and suspension provides for a control approach with far greater robustness than either technique in isolation. It is also noted that the most effective method by which axial load can be generated in a tendon without the development of associated shear is by pre-tensioning. As with roof reinforcement applying as high a pre-tension load to long tendons designed for suspension purposes will improve their overall effectiveness.

CONCLUSIONS

The paper has provided a review of the common use of suspension design for roadway roof support and highlighted a number of critical technical and risk-based issues that are not currently being given due consideration in the design process. Of most concern is that ignoring these issues results in the design being far more optimistic than should otherwise be the case.

The reason that many suspension designs are proving to be effective, thus also potentially convincing the designer that their design was appropriate, is that long tendon support is being applied pro-actively to the roof when it is not in a failed state and so in reality is acting to reinforce rather than suspend the roof. In essence the design process being used is inconsequential as compared to the use of a TARP and the early application of additional roof support. In lay terms the designer using suspension theory is typically getting the right answer for all the wrong reasons and this is a concern.

With the intent of attempting to improve the standard of geotechnical engineering being applied to underground coal mines, it is stated that in reality there is no need to use suspension design for roof support as credible methodologies are now available that directly address the design and application of reinforcing roof support. Methodologies such as ALTS 2009 (Colwell and Frith, 2009), AMCMRR (Colwell and Frith, 2010) and ARBS from the US (Mark, et al., 2001) are all focused on the design of reinforcing roof support within a risk-based framework (noting that acceptable levels of design risk as found in ARBS from the US may not be appropriate in Australia) and can be utilised at a mine site level.
by appropriately trained and supported strata control engineers. Therefore it is no longer fair to say that suspension design for roadway roof support can be justified on the basis of there being no credible alternative.

All underground coal mines rely on reinforcement mechanisms in order to provide for efficient and economic roof control and few would argue this point. Furthermore reliance on suspension roof support designed to a Factor of Safety appropriate to the exposure of persons to a design failure would in fact render the industry as uneconomic within a very short period. Therefore it is not only advisable that strata control engineers dispense with suspension design for roof support (other than for very specific situations), it is in fact obligatory if the design is to mirror both the manner in which roof support almost always acts to stabilise the roof and also its application within a pro-active strata management process where the design outcome is not in fact preventing a roof fall but in reality is to prevent a triggering of the TARP.

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CALCULATION OF SUBSIDENCE FOR ROOM AND PILLAR AND LONGWALL PANELS

Anton Sroka\textsuperscript{1,2}, Krzysztof Tajdus\textsuperscript{2} and Axel Preusse\textsuperscript{3}

ABSTRACT: A European method of surface subsidence calculation for partial room and pillar and longwall panel mining of coal deposit is presented. The article focuses on the Knothe’ method, that describes the subsidence trough by the Gauss influence function method. This method provides calculation of tilt, curvature radius, horizontal and vertical movements, and horizontal strain of the surface subsidence trough. The method to forecast surface subsidence serves to design exploitation with respect to minimalisation of its influence so that the forecast subsidence should not exceed a limit value.

INTRODUCTION

The development of surface subsidence methods refers to a mining exploitation process started by Keinhorst (1925). The breakthrough dissertation was published in 1931/1932 and it is called Bals Method (Kratzsch, 1983). The function influencing the aforementioned method was descended from Newton Low of Gravitation and interaction between two substances.

By considering the accepted the influence function method, the segments mesh of equal impact relates to subsidence on the middle point of the potential mesh that have been created. The subsidence of any point on the ground caused by free shape panel excavation can be determined using the graphic integration. This graphical integration as described in the exploitation field where the middle point of the mesh of segments will be superimposed with the calculated point.

Reference to similar solution is reported in Beyer (1944), Sann (1949) (see Kratzsch, 1983), Knothe (1953), Kochmanski (Collective Work, 1980), Ehrhardt and Sauer (1961), and Geertsma (Kratzsch, 1983).

The essence of surface subsidence is schematically presented in Figure 1.

![Figure 1 - The general scheme of surface subsidence methods](image)

In general, the occurrence of surface subsidence (present on Figure 1) might be:

- Filling of the void caused by mining and subsequent roof caving);
- Gradual convergence of some competent rock layers such as rock salt cavern; or
- Compaction of porous deposits during exploitation of oil and gas.

CALCULATION OF SURFACE SUBSIDENCE

The general formula to calculate the subsidence of a surface point by geometrical-integration methods can be presented in the form:

$$w(x_p, y_p) = a \cdot g \cdot \int \int_{p} \phi(x - x_p, y - y_p) \, dx \, dy$$

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On condition that:
\[ \int_{-\infty}^{\infty} \int_{-\infty}^{\infty} \phi(x, y) \, dx \, dy = 1 \]

Where: 
- \( a \) - subsidence factor, its value depends on an established system of exploitation and from a method of filling the void;
- \( g \) - thickness of excavated seam;
- \( \varphi \) - influences function, whose value is dependent on the method of calculation.

The combination of the most important past and currently applied influences function is presented in Table 1.

### Table 1 - An influences function \( \varphi \) according to various authors

<table>
<thead>
<tr>
<th>Author</th>
<th>An influences function ( \varphi )</th>
<th>Parameters</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bals</td>
<td>( \frac{1}{\pi \cdot \ln \left( \left( 1 + \tan^2 \frac{\xi}{r} \right) \right)} \cdot \frac{1}{t^2 + H} )</td>
<td>( \xi, , k = H \tan \xi )</td>
<td>( l ) - the horizontal distance of reference point from taken out element, ( H ) - depth of exploita</td>
</tr>
<tr>
<td>Beyer</td>
<td>( \frac{3}{\pi} \cdot \frac{1}{r_B^2} \cdot \left( 1 - \frac{l^2}{r_B^2} \right) )</td>
<td>( \gamma_B, , r_B = H \cot \gamma_B )</td>
<td>( r ) - range of main influence, its value depends on applied method,</td>
</tr>
<tr>
<td>Knothe</td>
<td>( \frac{1}{r_K^2} \cdot \exp \left( - \frac{l^2}{r_K^2} \right) )</td>
<td>( \beta, , r_K = H \cot \beta )</td>
<td>( \gamma, , \xi, , \beta ) - angles of ( r ) influence, its value depends on applied method,</td>
</tr>
<tr>
<td>Kochmanski</td>
<td>( \frac{1}{2\pi \cdot r_0^2} \cdot \exp \left( - \left( \frac{l}{r_0} \right)^b \right) )</td>
<td>( c(b) = \int_0^\infty \lambda \cdot \exp(-\lambda^b) \cdot d\lambda )</td>
<td>( \beta ) - scale coefficient horizontal influence</td>
</tr>
<tr>
<td>Ruhrkohle</td>
<td>( \frac{k}{\pi} \cdot \frac{1}{r_B^2} \cdot \exp \left( - k \cdot \frac{l^2}{r_B^2} \right) )</td>
<td>( \gamma, , k = -\ln 0.01 )</td>
<td>( b(H) = \frac{5 - 1.120 \log H}{1 + 0.672 \log H} ) ( r_c = H \cot \gamma )</td>
</tr>
<tr>
<td>Geertsma</td>
<td>( \frac{1}{2\pi} \cdot \frac{H}{(l^2 + H^2)^{3/2}} )</td>
<td>( \nu )</td>
<td>( b ) - influences function shape coefficient</td>
</tr>
</tbody>
</table>

From Table 1, the most important function applicable to the European mining industry includes both ‘the Knothe and the Ruthrkohle methods (Knothe 1953; Ehrhardt and Sauer 1961). Knothe assumed that the influence function method was adequately presented by Gauss’ function with proper parameters. This assumption has a strong theoretical background e.g. dissertations written by Kolnogorow (1931), Litwiniszyn (1956) and Smolarski (1967) (collective work, 1980) which are related to stochastic medium theory and the application of probability calculus.

According to the Knothe’ method, the elementary subsidence trough has the Gauss’ function shape, on condition that an elementary mining exploitation has the Dirac’s function shape (Figure 1). The assumption was achieved in 1960 - 1970 at The Strata Mechanic Research Institute of the Polish Academy of Science in Cracow, where the explorations in the loose medium (loose sand model) as a medium with a high degree of freedom have been carried out (Krzyszton, 1965). The single example of obtained results for the test are presented in Figure 2 and Figure 3.
The conformity between results of the explorations and the Gauss’ curve is shown in Figure 2. The graph of Figure 3 shows the dependence of subsidence distribution \( w \), horizontal displacement distribution \( u \), and their relation on \( u/w \).

Based on the initial results the following relationship hold:

\[
\frac{u(x)}{w(x)} = -\alpha \cdot x
\]

(2)

Where: \( x \)- distance of the consider point from the taken out element realised as “dump slotted”; \( \alpha \)- certain coefficient dependent on density of medium.

Sroka (1984) proposed a computable model based on discretization (division) of the completed exploited field on small ended surface elements. The elements are square shaped with a side of \( \Delta x \). The formula of such a subsidence value is:

\[
w(t, l) = \frac{M(t)}{r^2} \cdot \exp \left( -\pi \frac{l^2}{r^2} \right)
\]

(3)

\[
M(t) = \alpha \cdot V \cdot z(t)
\]

Where: \( M(t) \)- volume of elementary trough subsidence in moment \( t \).
\( l \)- distance of calculation point from excavated element of deposit,
\( r \)- radius of main influence according to Knothe’ theory,
\( V \)- volume of excavated element \( (V = \Delta x^2 \cdot g) \),
\( \beta \)- angle of main influence.
\( z(t) \)- function of time.

Radius of main influence \( r \) is calculated as:

\[
r = H \cdot \cot \beta
\]

(4)

The advantage of this solution is the fact, that each element can be described separately through a chosen thickness of excavated seam \( g \), depth of exploitation element \( H \), and time of excavated element \( t \). The calculation of element subsidence for the whole exploitation can be estimated by linear superposition, i.e. trough sum up partial subsidence from single elementary field.
Figure 4 - The scheme of determination of point subsidence \( P \) caused by taken out single deposit element

Because of accuracy of the calculation, the length of square side should not be higher than \( \Delta x_{\text{max}} \leq 0.1r \), and because of the possibility of analysis of the influence on mining exploitation velocity and stoppage of face advance on strata deformation in time – the length of element side should not be higher than average face advance.

The Ruhrkohle’ method, which is widespread in Germany, and the Knothe’ method are the same because of identical background of the Gauss' function.

The selection of time function is a key element during the space-time continuum analysis. The collection of functions, which are used in European mining industry, was presented in Table 2. From all functions, which were described, the Knothe’ function (1953), the Sroka and Schober’ function (1985), the Kwiatek’ function (2002) are worthy of mentioning.

In time function \( c(t) \), time \( t \) is calculated from the moment of excavation of the deposit element. The time equations given by Knothe’, Schober’ and Sroka are approximately equal for dependences:

\[
 c = \frac{\xi}{\xi + f}
\]

(5)

For time function according to Kwiatek, the following dependences can be presented:

\[
 c_1 \gg c_2 \\
 0.60 \leq A_1 \leq 0.85
\]

In Europe, the primary hazard index is an index of horizontal strain. Other indices like curvature (or curvature radius) do not have significant meaning because of increased depth of mining over 1000 m.

Fundamental dynamic hazard indices, among the static one are (especially in German mining industry), the speed of element subsidence, the growth speed of horizontal displacement and disturbance index of subsidence arising from the stoppage of longwall panel exploitation (Table 3).

Both the Knothe’ method and the Ruthrhohle’ method assumes, that the horizontal displacements need to be calculated according to the Awierszyn’ hypothesis (1947). This hypothesis makes an assumption on a proportionality between tilt vector \( T \) and a horizontal displacements vector \( u \) (Formula 6). This assumption also determines the relationship between a tensor strains and a tensor curvature \( K \) (Formula 7).

\[
 u = -B \cdot T \\
 \varepsilon = -B \cdot K
\]

(6)

(7)

Where \( B \) is the factor of proportionality, its value is located in limits of particular inequality \( 0.28r < B < 0.40r \). Factor \( B \) is a computational additional parameter and it should be estimated considering in-situ measurements.

The fact, that the Awierszyn’ hypothesis is qualitatively compatible with results of laboratory research results shown in Figure 3. The same hypothesis is also compatible with a modified method “centre of gravity point” which was published by Keinhorst.
Table 2 - Time function $z(t)$ based on various authors (Sroka 2003)

<table>
<thead>
<tr>
<th>Author</th>
<th>Time function</th>
<th>Parameters</th>
</tr>
</thead>
<tbody>
<tr>
<td>Knothe (1953)</td>
<td>$z(t) = 1 - \exp(-\alpha t)$</td>
<td>$w_k$ - final subsidence in time $t$,</td>
</tr>
<tr>
<td></td>
<td>$\dot{w}(t) = c[w_i(t) - w(t)]$</td>
<td>$w(t)$ - subsidence in real time $t$,</td>
</tr>
<tr>
<td>Martos (1956)</td>
<td>$z(t) = 1 - \exp(-\alpha \cdot \tau^2)$</td>
<td>$c$ - the relative velocity of mining influences propagation,</td>
</tr>
<tr>
<td></td>
<td>$\dot{w}(t) = c(t)[w_i(t) - w(t)]$</td>
<td>$\xi$ - the relative velocity of convergence (e.g. value 0.02 year$^{-1}$ correspond to clip passing velocity characterized by value of 2 % annually),</td>
</tr>
<tr>
<td>Trojanowski (1972/1973)</td>
<td>$z(t) = 1 - \exp\left(-\int_0^t c(\lambda)d\lambda\right)$</td>
<td>$f$ - the relative velocity of transmission of mining influences by rock mass,</td>
</tr>
<tr>
<td>Schober, Sroka (1983)</td>
<td>$z(t) = 1 - \frac{f}{f - \xi} \exp(-\xi \cdot t) + \frac{\xi}{f - \xi} \exp(-f \cdot t)$</td>
<td></td>
</tr>
<tr>
<td></td>
<td>$z(t) = C \ln(Bt + 1)$</td>
<td></td>
</tr>
<tr>
<td></td>
<td>for $t \leq (\exp\left(\frac{1}{C}\right)-1)/ B$</td>
<td></td>
</tr>
<tr>
<td>Sroka (1985)</td>
<td>$z(t) = 1 - \exp(-mu^{\frac{2}{5}})$</td>
<td></td>
</tr>
<tr>
<td>Kittlaus (1986)</td>
<td>$z(t) = 1 - \exp\left(1 - \frac{t}{t_0}\right)^2$</td>
<td></td>
</tr>
<tr>
<td>Schreiner, Kamlot (1991)</td>
<td>$z(t) = \left[1 - \exp\left(-\frac{t}{t_0}\right)^2\right] \left[1 - \exp\left(-f \cdot t\right)\right]$</td>
<td></td>
</tr>
<tr>
<td></td>
<td>$z(t) = 0$ for $0 \leq t \leq t_0$</td>
<td></td>
</tr>
<tr>
<td>Kowalski (1999)</td>
<td>$z(t) = 1 - A \cdot \exp\left(-c(t-t_0)\right)$ for $t &gt; t_0$</td>
<td></td>
</tr>
<tr>
<td>Kwiatek (2002)</td>
<td>$z(t) = 1 - 1 - A \cdot \exp\left(-c(t)\right) - A \cdot \exp\left(-c(t)\right)$</td>
<td></td>
</tr>
<tr>
<td>Sroka (2003)</td>
<td>$z(t) = 1 - \exp\left[-\left(\frac{t}{t_0}\right)^{m}\right]$</td>
<td></td>
</tr>
</tbody>
</table>

In European mining industry (excluding UK) the longwall panel exploitation is carried out without leaving pillars between longwall panels. In consequences, the fact of leaving pillars between longwall panels (especially in the American and Australian mining industry) leads to irregularities of subsiding troughs characterised by huge hazards, not even on the surface but also in the overburden. Additionally, leaving the pillars can lead to crumps, earth tremors and other hazards in water-bearing levels.

In longwall panel exploitation without leaving the pillars between them, formulas to calculate the maximum value of deformation indices are relatively simple, e.g. for full field (field bigger than $2r \times 2r$) the formulas are following:

Maximum subsidence: $w_{max} = a \cdot g$  \hspace{1cm} (8)

Maximum tilt: $T_{max} = \frac{w_{max}}{r} = \frac{w_{max}}{H} \tan \beta$  \hspace{1cm} (9)

Minimal radius of curvature: $R_{min} = \sqrt{\frac{e}{2\pi}} \cdot \frac{r^2}{w_{max}} = 0.66 \frac{H^2}{w_{max}} \cot^2 \beta$  \hspace{1cm} (10)
Maximum value of horizontal strain: \( \varepsilon_{\text{max}} = \pm \sqrt{\frac{1}{\varepsilon_e} \cdot \frac{w_{\text{max}}}{r}} = 0.60 \frac{w_{\text{max}}}{r} \tan \beta \) \hfill (11)

Table 3 - Boundary values of strain factors describing a dynamic and static impact of mining exploitation on the objects

<table>
<thead>
<tr>
<th>Category of strength of buildings</th>
<th>( R ) [km]</th>
<th>( \varepsilon ) [mm/m]</th>
<th>( \dot{w}_{\text{sr}} ) [mm/d]</th>
<th>( \dot{e}_{\text{sr}} ) [mm/d]</th>
<th>( \Delta w_{\text{sr}} ) [mm]</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 ( 40 &lt;</td>
<td>R</td>
<td>)</td>
<td>(</td>
<td>\varepsilon</td>
<td>\leq 0.3 )</td>
</tr>
<tr>
<td>1 ( 20 &lt;</td>
<td>R</td>
<td>)</td>
<td>(</td>
<td>\varepsilon</td>
<td>\leq 1.5 )</td>
</tr>
<tr>
<td>2 ( 12 &lt;</td>
<td>R</td>
<td>)</td>
<td>(</td>
<td>\varepsilon</td>
<td>\leq 3.0 )</td>
</tr>
<tr>
<td>3 ( 6 &lt;</td>
<td>R</td>
<td>)</td>
<td>(</td>
<td>\varepsilon</td>
<td>\leq 6.0 )</td>
</tr>
<tr>
<td>4 ( 4 &lt;</td>
<td>R</td>
<td>)</td>
<td>(</td>
<td>\varepsilon</td>
<td>\leq 9.0 )</td>
</tr>
</tbody>
</table>

This same method can be used for other systems of exploitation in mines and water bearing or aqueous deposits (Sroka and Tajdus, 2009). In general, what is important to determine the convergence (compaction) of the deposit element caused by mining. Later on using the influence function method an adequate description of surface subsidence is achieved (Formula 3).

Figure 5 shows the scheme of surface subsidence calculation when shortwall panel is applied together with leaving pillars between excavations. Roof layers are influenced (deflection occurs) by taken out same part of deposit and the left pillar are loaded by pressure \( p_z \). The pillar is partly crushed by value \( \Delta q \) (Figure 5).

![Figure 5 - Surface subsidence and convergence of pillar among shortwall exploitation](image)

Empirical value \( \Delta q \) can be presented in form of:

\[
\Delta q = \frac{p_z}{E} \cdot g
\]

\[
p_z = \gamma \cdot H \cdot \frac{f + l}{f}
\] \hfill (13)

Where: \( E \)- elastic modulus,

\( f \)- pillar width with an infinite length,

\( l \)- excavation width with an infinite length.

The scheme of stress distribution under the pillar is presented in Figure 6.

In this method, the calculation of potential drift (excavation) convergence and value of pillar \( \Delta q \) depression is essential. To achieve this numerical methods of Finite Element Method (FEM) and Finite difference method, (DEM) and alike, etc. (Figure 7) can be applied in which lots of factors should be taken into consideration (i.e. faults, discontinuity, fracturing, porosity, etc.) (Tajduš, 2009; Tajduš, et al., 2009).
The presented solution can also be used for continuous miner system (Sroka et al., 2009). Then calculation is carried out for full exploited mining field, in addition the value of subsidence factor received form of:

$$\eta_a = \alpha \eta^s$$

(14)

Where: $\eta$ - coefficient of coal deposit exhausted;
$\alpha$ - coefficient of subsidence for full exploitation;
$s$ - coefficient dependent on type of rocks (for weak rocks and average strength $s=1.5$; for strong rock $s=2.0$).

**CONCLUSIONS**

European computational methods are used mostly in designing a mining exploitation over build-up areas such as: Ruhrkohle (Germany) and the Upper Silesian Coal Basin (Poland). These methods have a positive impact on the protection of buildings and other infrastructure e.g. heat distribution network, road nets, power network. Moreover, the methods positively influence on industrial infrastructures such as: storage reservoirs and water-bearing layers in a rock mass.

The designing is concerned with the establishment of exploitation geometry, a run of the exploitation considering the time; maximal speed of exploitation and the face advance stopping during the exploitation. Other elements, which have to be taken into consideration during the exploitation designing, are settlement of safety pillars or estimation of final borders of mining exploitation. The presented computational method allows for calculating of surface deformation, taking into account an optional shape and optional time of exploitation.

This method is currently been used not only in the mining industry of hard coal, but also in metalliferous mines, such as salt and copper mines and in oil and gas fields. The computational method, which is based on the Knothe’ method can be fully applied to help in predicting subsidence and minimise its impact; this can also be in the Australian mining industry.

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SIMULATION OF DEVELOPMENT IN LONGWALL COAL MINES

Mehmet S Kizil, Alex McAllister and Robert Pascoe

ABSTRACT: Longwall equipment has been constantly developed and has experienced improvements in cost, efficiency and production. The effect of improvements to overall longwall capacity has put an increasing strain on development operations. Development lag causes issues within mining operations and there are substantial flow-on costs, including higher unit costs, lower production and impacts to market relations. Longwall development processes and the factors that affect productivity are examined. A development program that could be used to simulate the advance of development and production in a longwall coal mine is described. With this tool, it is hoped that mine management can effectively and efficiently respond to changes in mining conditions and combat development lag. Additionally, the simulation package can be used as an educational and demonstration tool.

The main feature of the program is to be able to accommodate differences in advance rates through the development and production panels and hence calculate and simulate a total time for both. The program allows the accommodation of changes in mining conditions as they are experienced.

INTRODUCTION

A significant amount of development is needed before longwall equipment can be moved into the next panel in the sequence and therefore the timing of development to coincide with the end of panel production is vital. In the event that the next panel in the mining-sequence is not ready for the longwall equipment to be relocated, significant amounts of money can be lost as production equipment remains idle. This problem has been exacerbated in recent years with greater developments made to longwall productivity over improvements to the Continuous Miners (CM) used in development and consequently development has become the weakest link in the mining cycle (Misra, 1996). Norwest (2008) expressed the magnitude of the problem with the following statement: “The number one issue affecting the future of longwall mining is continuous miner development”.

Understanding the progression of roadway development is vital in the planning and management of an active mine. An easy-to-use tool that can simulate, assess and show the impact of changing mining conditions would enable mine management to initiate strategies to deal with projected longwall stand. Also, an easy-to-use simulation package acts as a valuable teaching and demonstration tool within the mining industry. The previous version of the software which was developed within Microsoft Excel (Barker and Kizil, 2009) received interest from industry personnel and hence, further development was the focus of research.

Longwall equipment has been constantly developed and has experienced improvements in cost, efficiency and production. The effect of improvements to overall longwall capacity has put an increasing strain on development operations, with the consequent development lag causing problems for mining operations. There are substantial flow-on costs which result from development lag, including higher unit costs, lower production and impacts to market relations. It has been suggested (Barker and Kizil, 2009) that a development unit can remain well ahead of longwall production as a result of the efficient use of hardware, personnel, supply logistics, planning and scheduling.

The simulation focuses on the key activities that occur in the development sequence in order to help provide an accurate recreation of the pillar cycle process. Several common sequence steps were identified when mining a standard gateroad pillar. Optimal time frames were recorded for completing each task within the sequence plan. With the tool, it is hoped that mine management can effectively and efficiently respond to changes in mining conditions and combat development lag. Additionally, as an easy to use simulation package it can be used as an educational and demonstration tool.

Roadway development at a Northern Queensland mine was monitored to provide primary and secondary data for the project. Comprehensive research of the standard panel and pillar sequence plan was conducted to identify the inherent delays associated with the method. This information provided a
basis for designing a super panel configuration and sequence plan. Major risks and hazards within roadway development will need to be identified and accounted for to allow for safe cycle design. Development of a basic super panel layout has been performed to provide enough information to design a super panel sequence plan for the Australian mining industry.

**ROADWAY DEVELOPMENT**

Roadway development in longwall mining involves creating roadways or headings to the longwall panel to provide ventilation, power and services to the longwall operations. The development unit is considered a cost centre. This demands that roadway development occurs as efficiently and effectively as possible. Australian roadway development generally utilises two heading gateroads. These headings are driven using the in place method.

The standard panel method is the most commonly used method for roadway development within the Australian coal mining industry. The standard or conventional method will be defined as the driveage of both headings in a gateroad, using one continuous miner, shuttle car, auxiliary fan and boot end. Current Australian development methods and practices are becoming outdated. The term discontinuous mining refers to the period of planned nil production currently experienced in development pillar cycles. Gibson (2005) suggests that the current mining methods in both standard development and super panel development are discontinuous.

Rapid roadway development

The Rapid Roadway Development (RRD) system was formulated to increase gateroad development rates, ensuring longwall continuity, and hence maintain a reliable supply of coal to consumers (Kelly, et al., 2000). Gibson (2005) identified that the cyclic, stop-start nature of current development methods was seen to be a limit to development rates. Especially when haulage distances exceeded 70 m. The following solutions were put forward to counteract this (Kelly, 1999):

- An integrated development system concept;
- Monorail continuous haulage; and
- Monorail mounted services to speed up panel moves and reduce manual handling.

Kelly (1999) found that the solutions mentioned above were not as effective as first expected due to the lack of methodology in project management where these processes were occurring. It was found that partial gains could be made from continuous haulage, but significant gains could be achieved through improving current development processes. Of particular importance was the time constraint recorded when advancing conveyer and panel services whilst employing dual continuous miners in a two heading development, more commonly known as a super panel (Gibson, 2005). A lot of the flexibility of a standard panel unit is removed when running a super panel unit, increasing the duration of panel moves from 27 to 36 hrs. This suggests that there are problems with the super panel method and that improvements can be made.

It has been identified that a benchmarking process is needed across industry to let individual operations pinpoint the time constraints in their operation. One way to achieve this is to incorporate programs such as UCDelay, which categorises an operation’s generic delays. This allows mines to standardise their delay categories which will allow for benchmarking across the industry (Porter, Baafi and Boyd, 2010).

**CLASSIFYING AND IDENTIFYING DEVELOPMENT TIME CONSTRAINTS**

Data was sourced from the selected Northern Queensland operation. Daily, weekly and monthly production reports were compiled using Microsoft Excel. This allowed for the breakdown of work in development units which comprise the total available time of a development panel. It was decided not to collect or analyse outbye panel data as these delays are not recorded as development delays. The data analysed was sourced over 2009. This year was selected because it was the most current data with a full year of production statistics.
Development delay classification

In a mining operation a delay is any event that holds up production or prevents mining equipment operating under standard conditions. These delays are identified and categorised in order to find where major time losses are occurring. It is logical that minimising development delays will generally increase overall output and more specifically reduce development pillar cycle times. Delays can be classified as either planned or unplanned. Planned or ‘scheduled’ delays are process delays required for the safe function of a mining operation. Unplanned or ‘unscheduled’ delays are generally random in nature and unpredictable (Porter, Baafi and Boyd, 2010).

Figure 1 illustrates calendar time availability for a typical underground longwall operation. There are several elements that need to be identified when reporting delays. These include, the date and time of delay, duration and type of delay, scheduled or unscheduled delays, and which division is responsible for the delay.

![Figure 1 - Breakdown of a development equipment calendar time (Porter, Baafi and Boyd, 2010)](image)

Time breakdown

The overall productivity of a mine is highly influenced by development delays. In order to quantify this it is necessary to identify where and how much an operations resourced time is allocated. Figure 2 provides a breakdown of the production summary for 2009. It is evident that development delays (37%) occupy a significant portion of the production summary, almost as much time as actual production (39%). The major external delay units are the outbye panel and outbye conveyor units. Planned maintenance time also falls under resourced time. The remaining section of the production summary is unscheduled or non-resourced time.

![Figure 2 - Production summary breakdown](image)
major delay causes in roadway development. Each unit is comprised of individual delays, which combine to calculate the total delay time for each unit.

Figure 3 identifies the total delay time for each major unit for 2009. The development unit delays recorded are quite significant. The total delay time of 142 days only leaves 201 days available for productivity under resourced time. One particular delay may take up a large period of scheduled time. This does not particularly suggest that the delay is a common problem to development over a long period of time. Such an example would be delays experienced due to inrush from a 1 in 100 year event. This may take up a substantial amount of time but it is not frequent. Therefore it is important to consider the frequency of delays that are affecting the development panel. Figure 4 provides a breakdown of the frequency of major individual delays affecting the development panel in 2009. These delays are not restricted to one development unit. These types of delays need to be focused on to improve current super panel design. Panel delays occur when performing standard panel activities or processes within roadway development.

Figure 3 - Total development unit delays for 2009

Figure 4 - Total development unit delays for 2009

Super panel delays

What differentiates the super panel method from the single panel method in terms of delay recording is the classification of the super panel as a dual productive unit. A dual productive unit can be defined as a process or piece(s) of equipment where more than one primary productive unit is being used concurrently. It is possible to have more than one primary productive unit in what would normally be considered a single process, such as using two continuous miners in the panel. Each unit is tracked separately in order to classify delays correctly.

The super panel configuration allows for parallel or multiple production paths that enable continued production at a reduced capacity when parts are shut down. For example, if one continuous miner is shut down both shuttle cars are available to transport coal from the other continuous miner. These multiple sub processes have an effect on design and layout of the super panel. Delays that were noted to affect super panel performance were water control, supply miner, flooding, road maintenance, bogged, trip/reset/overload, supply/ trailing cable, transformer move, and process ventilation work. Some of the major delays recorded in a standard panel were omitted when classifying delays for a super panel. This was not because these delays do not occur in a super panel, just that these delays are not particularly relevant in developing the layout and sequence plan of a super panel. As longwall production is improving linearly, development output needs to increase yearly. Sensitivity to time is high and an idle longwall results in the under-utilisation of a very expensive asset. The layout of a super panel was designed to aid in the design of a sequence panel.

Super panel layout

A generic layout of a super panel was designed to improve the super panel method. Fundamentally the super panel layout aimed to:

- Ensure the highest level of personnel safety in the panel;
- Prevent unplanned and process delays;
- Improve the ease of operations by providing a consistent work flow; and
Minimise equipment damage.

The option with the most significant performance was the premium panel. This method used the same equipment layout as the super panel, with one alternating crew operating both continuous miners as per the sequence plan. Although the cutting rate was lower, the one pillar cycle using the premium panel was completed 43 h faster than the super panel. This method is suited to most operations and has the capacity to be incorporated to any operation over time.

Super panel pillar sequence

Figure 5 illustrates the steps involved in completing a pillar sequence for a super panel. As both continuous miners are running concurrently, the majority of required tasks, needed to complete a pillar sequence, run in parallel. Each step of the sequence drives approximately 20 m of roadway. By breaking the pillar cycle up into smaller steps it is easier to ensure all tasks are completed.

Panel options

A study at a mine in the Hunter Basin revealed that there are similar panel methods that yield high metres per cycle and high cutting rates. Table 1 shows the panel performance of each method trialled.

<table>
<thead>
<tr>
<th>Method</th>
<th>100 m Cycle (days)</th>
<th>Metres Per Cycle</th>
<th>Cut Rate (m/h)</th>
<th>Panel Advance (h)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Standard Panel</td>
<td>6.4</td>
<td>230</td>
<td>4</td>
<td>17.6</td>
</tr>
<tr>
<td>Super Panel</td>
<td>4.6</td>
<td>236</td>
<td>5.7</td>
<td>25.7</td>
</tr>
<tr>
<td>Premium Panel</td>
<td>3.7</td>
<td>236</td>
<td>5.5</td>
<td>18.5</td>
</tr>
</tbody>
</table>

SIMULATION PROGRAM

A relatively “straight-forward” simulation program has been developed for non-technical personnel for the front-line analysis of changes in mining conditions. Given that the first version of the package (Barker and Kizil, 2009) was built around a Microsoft Excel framework, it was decided that a stand-alone package built within the VB.Net framework would better withstand the rigors of the mining industry.

The simulation was created to continue the research performed by Barker and Kizil (2009). The objectives of the simulation were to:

- Quickly determine development float/longwall stand time given the appropriate input parameters;
- Assess the impact of changing mining rates or operating time;
- Determine the effects of changing mining parameters such as longwall block length, cut-through length and longwall face length;
- Estimate the financial outputs given input parameters;
• Clearly show/teach the impacts of altering various mining parameters on the mine schedule to crews, students and non-mining personnel;
• Calculate total advance rates as a result of geological input conditions defined by domain; and
• Utilise user-friendly input and operation menus.

The simulation can therefore be used as an effective tool to quantify the impact of changes in advance rates due to either alterations in efficiencies or geotechnical conditions. Additionally, given the fast and efficient nature of the software, the program can help in making decisions regarding the allocation of mining resources, manning requirements and the impact of changes in production/development efficiencies. The main control screen of the simulation can be seen in Figure 6.

![Figure 6 - The main DevSIM control screen showing both simulations in progress](image)

**Configuration**

The longwall and development configuration shown are broken into three separate tabs: dimensions, performance and financial. The configuration windows were given this arrangement to minimise perceived complexity and to also minimise the amount of screen real-estate used, to account for the use of low resolution monitors.

While the input fields in the ‘dimensions’ tabs are specific to either the longwall or development input windows, the ‘performance’ and ‘financial’ tabs for both are exactly the same; the performance and dimension tabs can be seen in Figure 7. Changes to the parameters in each of the tabs have a direct affect on the values in the lower ‘summary’ portion of the window, acting as an effective KPI calculator.

The biggest update to the research performed by Baker and Kizil (2009), apart from moving the simulation to the VB.Net framework was the addition of the facility to delineate areas within the longwall or development panels that experience different rates of advance to the default set for the respective panel. The ‘advanced performance configuration’ button within the ‘performance’ tab of the configuration window for longwall and development will open the ‘advanced configuration’ window. The advanced configuration window allows the user to define the metres per operating hour, start and end position of areas within the panel with a non-standard advance rate. When attributing an advance rate to a section of the panel, the program works downward through the recorded section. Given this downward progression of parameters, if the first section in the list delineated an area starting at 100 m and ending at 400 m whilst the second section in the list delineated a section starting at 150 m and ending at 200 m; there would be effectively three delineated sections between the 100 and 400 m marks. If however, the order of these sections were reversed, the advance rate for the 100 to 400 m section would be attributed to the whole area.
Deficiencies

The simulation program is still in the development and improvement stage and therefore still has a number of deficiencies. The first of these deficiencies is due to the software not taking changes in operating efficiencies into account across different advance domains. The impact of changes in operating efficiencies through harder or more abrasive areas of advance would enable the software to better simulate the behaviour of a development of production panel as a whole. The program needs to deal with variations in mining conditions across two development headings within the same section of development panel. Currently, the latest version of the simulation package does not accommodate this variation.

CONCLUSIONS

It has been established that the biggest factor affecting increases in longwall performance is ensuring that development and preparation of the following panel coincides with the end of panel production. Managing panel development can prove difficult, as there are many factors to consider when estimating development advance and compliance with schedule and budget.

From the case study conducted, it was found that the important delays affecting roadway development were unplanned and process delays associated with travel times, flooding, water control and supplying the continuous miner. These delays were both frequent in occurrence and long in duration. By implementing the defined super panel sequence operations have a basis to work off for future gateroad development when development is struggling to produce an adequate longwall float.

The development of a longwall development simulation program is discussed. It is important to understand all of the factors that affect longwall and development advance. As development cutting rate and support regimes are directly influenced by the geological domain, it makes sense to suggest that development rates could be correlated to ‘development domains’ within the coal seam. This idea of course depends on mining and installation of support being the limiting factor in the development process.

The longwall development simulation program with the incorporation of variable advance rates mapped to geological and support domains should provide a useful frontline tool for use in longwall coal operations.
REFERENCES


STABILITY ASSESSMENT AND SUPPORT DESIGN for Water Deviation Binary Tunnels of Bakhtiyari Dam-Iran

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ABSTRACT: Analysis of the stability of deviations binary tunnels at Bakhtiyari dam situated in the southwest of Iran is presented. The diameter of each tunnel is 13.7 m and they are around 1000 m long. The tunnels are excavated in steps of consecutive cuts. The tunnels pass through seven different geological zones with various specifications. To study the characteristics of these zones nine boreholes of total length of 904 m were drilled and around 140 various laboratory tests were conducted on the core samples. Testing and analysis of the cores from the boreholes have resulted in series of data required for the investigation. These data present physical and mechanical properties of the seven various rock zones, including RQD, joint sets and joint properties. Based on these data the values of RMR and Q and therefore the class of the rocks of all seven zones were determined. Stability analysis has been conducted and appropriate supports were suggested for both tunnels by RMR and Q methods. Based on the ratings ascribed to each zone by the two methods a relationship has been driven between RMR and Q for this particular project.

INTRODUCTION

Rock mass characterization is normally carried out through the application of empirical classification systems, which use a set of geotechnical data and provide an overall description of the rock properties. Moreover, they provide other important information like support needs, stand-up time, geotechnical parameter among others (Sing and Goel, 1999).

Different classification systems have well known drawbacks and limitations, due mainly to their empirical base (Palmstrom, 1995). However, they are still very useful in practice. Therefore, there is a need to improve their efficiency. Two of the most used classification systems are the RMR-Rock Mass Rating and the Q-system (Sing and Goel, 1999). The RMR and Q systems have evolved over time to better reflect the perceived influence of various rock mass factors on excavation stability (Rajnish and Bhawani, 2006). This paper discusses the evolution of these systems, as well as problems associated with estimating the Q, RMR indices for water deviation binary tunnels in the Bakhtiyari dam of Iran.

SITE GEOLOGY

Bakhtyari dam site is in the South West of Iran, almost 70 km North-East of Andimeshk town (Khuzestan province) and some 65 km South-West of Doroud town (Lorestan province). The dam axis lays at 290725 E and 3648729 N points. Figure1 shows the Location of the project area (Iran Water and Power Resourced Development Co, 2006).

The geological formation consists of a series of asymmetric folding and faults. The project area is covered by the sedimentary bedrocks of Sarvak and Garau formations. The Sarvak Formation is divided into 7 units from SV1 (oldest) to SV 7 (youngest). At project site the Garau Formation is younger than the Sarvak Formation and is divided into two units (Iran Water and Power resource Development Co, 2008). Figure 2 shows longitudinal geological section of right diversion tunnel.

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\(^2\) Lecturer, Shahid Bakeri Branch, University of Oromieh, Miandoab, Iran
\(^3\) Project Expert, Iran Water and Power Resource Development Co., Tehran, Iran
Figure 1 - Location of the Project Area on Iran Map (Iran Water and Power resourced Development Co, 2006)

Figure 2 - Longitudinal geological section of right diversion tunnel (Iran Water and Power Resourced Development Co, 2006)

PROJECT DESCRIPTION

Deviation system of Bakhtiyari dam includes two tunnels, namely upper and lower tunnels. The diameter of circular cross section of the upper tunnel is 13.7 m and the length of this tunnel is 1181 m. The cross section of the lower tunnel is D-shaped with 13.2 m width and 13.7 m height. This tunnel is 1151 m long. Both tunnels are approximated with a diagonal pattern that is excavated with heading and benching method (Iran Water and Power resourced Development Co, 2006).

A number of nine boreholes were drilled with five boreholes at the upstream and downstream cofferdams and four boreholes along the diversion tunnels path. Total drilling length was 904.1 m consisting of 7.30 m in overburden and 811.78 m in the bedrock.

DISCONTINUITIES SYSTEM

Rock mass in the Bakhtiyari dam diversion section consists of four set of discontinuities. The characteristics of these discontinuities have been studied in the galleries and boreholes located in the dam site. Stereographic plot of discontinuities along the diversion tunnel is shown in the Figure 3.
EVALUATION OF ROCK MASS QUALITY IN THE BOREHOLES

The first set of the information taken from the freshly recovered drill cores was Rock Mass Quality or RQD parameter. It is defined as the ratio of the total length of intact, sound core pieces longer than 10 cm to the length of the core run.

Thus, the RQD is a direct measurement of the degree of the bedrocks fracturing and by this also an indirect account of the grade of weathering. Technical fractures, produced during drilling and recovery of the cores from the core barrel therefore have been disregarded. The RQD value is significantly depending on the relationship between orientation of the discontinuities and the borehole axis. In the project area tectonic structures such as faults, kink bands, the joint sets and in some cases the lithological bedding planes have a remarkable effect on the RQD value. Figure 4 shows the variation of RQD versus the elevation (m.a.s.l) for the boreholes number B435 and B302 (Iran Water and Power resourced Development Co, 2008).

<table>
<thead>
<tr>
<th>parameter</th>
<th>SV3(Disturbed)</th>
<th>SV3-SV2</th>
<th>SV4</th>
<th>SV5</th>
<th>SV6</th>
<th>SV7</th>
</tr>
</thead>
<tbody>
<tr>
<td>RQD(%)</td>
<td>40-60</td>
<td>55-75</td>
<td>65-75</td>
<td>75-90</td>
<td>65-85</td>
<td>50-80</td>
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<td>fair</td>
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<td>good</td>
<td>good</td>
<td>Fair</td>
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</tbody>
</table>

Table 1 - RQD values in diversion tunnels (Iran Water and Power resourced Development Co, 2008)
Table 2 - The Ratings and values of the various rock mass parameters in two systems

<table>
<thead>
<tr>
<th>INPUT PARAMETERS</th>
<th>SV3 (disturbed)</th>
<th>SV2&amp;SV3</th>
<th>SV4</th>
<th>SV5</th>
<th>Items</th>
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<th>SV7</th>
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<td></td>
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<td></td>
<td></td>
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<td>DEGREE OF JOINTING</td>
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<td>RQD(%)</td>
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<td>40-60</td>
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<td>55-75</td>
<td>16-17</td>
<td>65-75</td>
<td>17-20</td>
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<td>10</td>
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<td>4</td>
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<td>4</td>
<td>1</td>
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<tr>
<td>JOINT CHARAC-TERIS TICS</td>
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<td>Smoothness joint roughness</td>
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<td>Joint alteration</td>
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<td>TOTAL</td>
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<td>55-58</td>
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<td>Fair Rock</td>
<td>Poor Rock</td>
<td>Fair Rock</td>
<td>Good Rock</td>
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</tr>
</tbody>
</table>
RMR AND Q CLASSIFICATION OF THE CASE

The main classification systems for rock support estimates, Q and the RMR, use the most important ground features or parameters influencing on stability as inputs. Each of these parameters is classified and each class given a value or rating to express its influence on tunnel stability (Palmstrom, 2008). Table 2 shows the values of the various rock mass parameters in the two systems.

Although the rating methods of RMR and Q-system are additive and multiplicative, respectively, the basic Concepts of both schemes are similar. Both schemes allocate the ratings to the properties that influence the rock mass behavior and then quantitative figures such as total-RMR and Q-value are produced. These values would be used to judge the goodness of rock mass for construction (Rajnish and Bhawani, 2006). Table 2 indicates that in both systems the least quality is due to the zone SV7 and the maximum quality due to the zone SV5.

COMPARISON BETWEEN THE TWO CLASSIFICATION SYSTEMS

Figure 5 depicts the results from comparisons conducted. Table 3 shows correlation equations between maximum, minimum and average values found for RMR and Q systems. Comparison of the average values obtained by the two systems was done through regression. The result showed that average difference between the two systems was not more than 2%. The maximum difference was about 3% which was due to minimum values estimated by the two systems.

![Figure 5 - Comparison between RMR and Q systems](image)

Table 3 - Correlation equations between the values found for RMR and Q

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Equation</th>
<th>R²</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average</td>
<td>( R_{MR} = 9.52 \ln(Q) + 42.39 )</td>
<td>0.9815</td>
</tr>
<tr>
<td>Maximum</td>
<td>( R_{MR} = 9.65 \ln(Q) + 42.52 )</td>
<td>0.9749</td>
</tr>
<tr>
<td>Minimum</td>
<td>( R_{MR} = 9.1 \ln(Q) + 43.23 )</td>
<td>0.9733</td>
</tr>
</tbody>
</table>

CONCLUSIONS

The following conclusions could be drawn from the current study:

- RMR classification system ranks the various units of rock mass of Bakhtiari dam tunnel as medium to good where Q system ranks it as poor to good.
- In most cases the class of “medium”, estimated by RMR coincides with the class of “poor” offered by Q.
- Both classifications suggest “good” class for SV5 unit.
• In both systems most of the rock units hosting the tunnel fall into medium class.
• The results obtained from both classifications demonstrate a high correlation where the differences between the values suggested by them are around 2% for medium values and 3% for minimum values.

REFERENCES

Rajnish K, Bhawani S G, 2006. Tunneling in weak rocks, ELSEVIER GEO-ENGINEERING BOOK SERIES.
GEOTECHNICAL APPRAISAL OF THE THAR OPEN CUT MINING PROJECT

Raghu N Singh¹, Abdul Ghani Pathan², David J Reddish¹ and Anthony S Atkins³

ABSTRACT: This paper is concerned with a slope stability appraisal of the proposed open cut mining operations in the Thar lignite field in Sindh, Pakistan. The Thar coalfield covers an area of approximately 9,000 km² and is estimated to contain 193 billion tonnes of lignite resources. The design of safe high wall slopes is necessary to ensure mine safety and overall economical viability of the mining operations. In the Thar lignite field, the presence of three main aquifers induces pore pressure in the rock mass surrounding the lignite seams and makes high wall slopes potentially unsafe. It is, therefore, necessary to dewater the rock mass before commencing mining excavations. A proposed mine dewatering scheme to facilitate rock mass dewatering surrounding the mining excavations and a description of the slope stability analysis of the high wall using the software “SLIDE” version 5 is outlined. Three computer models with slope angles of 28°, 29° and 30°, incorporating a plane failure mode, were analyzed to investigate the stability of pit slopes. The generalized stratigraphy of borehole RE-25 has been used for the development of the computer models. The main conclusions of this study are that the slope angle of 28° is quite acceptable for a Factor of Safety (SF) ≤ 1.3 whereas the excavated slopes with slope angles ≥ 29° are not safe against the plane failure for SF > 1.3. This assessment was followed by a slope stability analysis incorporating circular failure modes. Five models incorporating various slope angles ranging from 23° to 27° and one model incorporating combined slope angles of 23° in dune sand and 26° in the rest of the strata were developed and analysed. The main conclusions from this study are that the dune sand layer (having a thickness of 48 m) is acceptable for a SF of 1.3 at slope angle ≤ 23°, while the rest of the strata is acceptable for SF=1.3 at slope angles ≤ 26°. The overburden to lignite extraction ratio for this slope design has been calculated as 3:1 or 3 m³ of overburden over 1 t of lignite.

INTRODUCTION

The design of the high walls of an open pit mine is one of the most important aspects in surface mining planning because slope stability calculations are necessary to ensure the safety of excavated slopes coupled with the economic viability of the mining operations. The main objective of this preliminary study is to determine the overall slope angle that is safe during the working life of the mine and this will determine the optimum pit limit. A slope stability analysis was carried out on various models for a geological section of block-I at the Thar lignite deposit. Various slope angles of between 230 and 270 have been investigated using a safety factor of 1.3. Slide software version 5 was used for the development of models and slope stability analysis using a plane and circular slip surface search. The proposed mine dewatering scheme to ensure stable slopes surrounding the mining excavations is described.

THAR LIGNITE FIELD

Pakistan now possesses the seventh largest lignite resource in the world with 193 billion tonnes of lignite/coal reserves mainly concentrated in the Thar region in the eastern part of Sindh Province, about 400 km east of Karachi as shown in Figure 1. The Thar coalfield covers an area of approximately 9,000 km², where lignite/brown coal beds lie at depths of between 130 m and 250 m. The cumulative seam thickness varies between 1.45 m and 42.6 m and the maximum thickness of an individual seam is 28.6 m. The Geological Survey of Pakistan (GSP) and the United States Geological Survey (USGS) under the Coal Resources Exploration and Assessment Program (COALREAP) first discovered this lignite field in 1994.

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The Geological Survey of Pakistan (GSP) and Deep Rock Drilling (DRD) have completed a detailed assessment of coal resources in eight blocks of the Thar coalfield. The area covered by this exploration program is 730 km², containing some 19.344 billion tonnes of reserves. This represents 89% of Pakistan's total recoverable reserves. The eight blocks explored in the Thar coalfield are shown in Figure 2. Analysis of the Thar lignite indicates a relatively low heating value, between 9.4 and 12.7 million Btu per tonne. The most appropriate large-scale application of the lignite is for power generation, and worldwide more than 90% of lignite and brown coal is used for this purpose (Ahmad and Farzana, 2001).

Chemical analysis of some 2000 coal samples has been undertaken, and the rank of coal has been determined, ranging from lignite-B to sub-bituminous-A. The weighted average composition of lignite-B at Thar is presented in Table 1.
Table 1 - Showing the composition of Thar lignite (after Thomas, et al., 1994)

<table>
<thead>
<tr>
<th>S.No.</th>
<th>Parameter</th>
<th>% composition</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.</td>
<td>Moisture (as received)</td>
<td>46.77 %</td>
</tr>
<tr>
<td>2.</td>
<td>Fixed Carbon (AR)</td>
<td>16.66 %</td>
</tr>
<tr>
<td>3.</td>
<td>Volatile Matter</td>
<td>23.42 %</td>
</tr>
<tr>
<td>4.</td>
<td>Ash (AR)</td>
<td>6.24 %</td>
</tr>
<tr>
<td>5.</td>
<td>Sulphur</td>
<td>1.16 %</td>
</tr>
<tr>
<td>6.</td>
<td>Heating Value</td>
<td>5,774 btu/lb</td>
</tr>
</tbody>
</table>

STRATIGRAPHY AND LITHOLOGY OF THAR LIGNITE FIELD

Lignite seams in the Thar area are found in the Bara formation of the Paleocene/Eocene age. The Bara formation is some 95 m thick consisting of sandy/silty claystone and a sandstone formation overlying the basement granite lying at a depth of 100 m to 220 m. The basement rock is very light grey, weathered, medium compacted-granite containing fine to coarse quartz grains. The overlying Bara formation consists of layers of carbonaceous claystone, sandy claystone and silty claystone. Carbonaceous claystone is medium light grey to brown in colour containing carboniferous petrified roots, carbonaceous materials and rare sandy resin globules. The olive grey to dark-grey claystone containing petrified plant roots and pyretic resin globules overlies this sediment.

![Figure 3 - Stratigraphy of Thar lignite field (based on R W E Power International, 2004)](image)

There are number of coal seams of varying thickness ranging from 3 m to 21 m at an average depth of 170 m. The Bara formation is overlaid by the sub-recent formation comprising inter-beded carbonaceous sandstone, siltstone and claystone up to 65 m thick, at a depth of 52 – 125 m. The recent formation overlying the sub-recent formation consists of some 50 m of thick dune sand. This sand is fine to medium grained, yellowish grey in colour, containing sub-rounded and moderately sorted grains of ferromagnesian minerals. Figure 3 shows the stratigraphic section and lithology of the Thar coalfield.

OUTLINE OF THE PROPOSED ROCKMASS DEWATERING SCHEME

Mine dewatering arrangements comprise of four main elements namely, surface drainage ditch, dewatering wells to the top aquifer, dewatering wells for intermediate aquifer, dewatering well for the base aquifer as shown in Figure 4.
Design of surface drainage ditch

A review of the rainfall data from the Mithi district indicates that the maximum daily rainfall of around 100 mm/day occurs during the months of July or August. It is expected that this will lead to a certain amount of flooding during unexpected rainfall, meaning that the entire operation of the mine may completely close down for a period of one to two days. The peak flow to the surface drainage system for Thar mine has been calculated using the rational formula in a previous publication (Pathan, et al., 2008) as $7.5 \times 10^4$ L/s. Pumping rates from the three aquifers at the Thar prospect have been calculated using the equivalent well approach by Pathan et al. 2008 and are as follows:

Pumping out rates from Top aquifer is calculated = $116$ m$^3$/d;
Pumping out rates from the intermediate aquifer $r = 147$ m$^3$/d;
Pumping out rates from the Bottom aquifer $r = 1.34 \times 10^6$ m$^3$/d.

DESIGN PARAMETERS FOR SLOPE STABILITY ANALYSIS

A slope stability analysis was performed for the geologic section shown in Figure 5, which is the generalized geology of borehole RE25. The thickness of various strata layers is given below:

![Figure 5 - Stratigraphy of Block-4 of Thar lignite field, Pakistan (After Pathan 2007)](image)

The following design parameters shown in Table 2 were used for the slope stability analysis:
Table 2 - Showing the input design parameters after Pathan, Singh and Shah (2006)

<table>
<thead>
<tr>
<th>Stratum</th>
<th>Formation</th>
<th>Density, $\gamma$ (kN/m$^3$)</th>
<th>Angle of internal friction ($\phi_o$)</th>
<th>Cohesion, $c$ (kN/m$^2$)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dune Sand</td>
<td>Recent</td>
<td>17</td>
<td>30</td>
<td>0</td>
</tr>
<tr>
<td>Siltstone</td>
<td>Sub-recent</td>
<td>21</td>
<td>14</td>
<td>200</td>
</tr>
<tr>
<td>Sand</td>
<td>Sub-recent</td>
<td>17</td>
<td>30</td>
<td>0</td>
</tr>
<tr>
<td>Claystone1</td>
<td>Bara</td>
<td>16.9</td>
<td>27</td>
<td>120</td>
</tr>
<tr>
<td>Claystone2</td>
<td>Bara</td>
<td>18.8</td>
<td>23</td>
<td>200</td>
</tr>
<tr>
<td>Claystone3</td>
<td>Bara</td>
<td>19.4</td>
<td>25</td>
<td>40</td>
</tr>
<tr>
<td>Lignite</td>
<td>Bara</td>
<td>11.8</td>
<td>30</td>
<td>200</td>
</tr>
</tbody>
</table>

DESIGN PARAMETERS FOR STABILITY ANALYSIS USING PLANE FAILURE SURFACES

Three computer models of pit slopes, with slope angles 28°, 29° and 30°, have been investigated against plane failure. The computer software “Slide” version 5 was used for the design of slopes. An example of one model with a slope angle of 30° is shown in Figure 6 incorporating the orientation of a potential failure plane. The following parameters were used in all the models:

- Analysis technique = Bishop;
- Surface Type = Plane, Non-circular;
- Search Method = Block;
- Number of surfaces = 5000;
- Left angle start = 135;
- Left angle end = 135;
- Right angle start = 45;
- Right angle end = 45.

Plane of failure was added in each model and the coordinates of the plane were entered at the prompt line.

RESULTS AND DISCUSSIONS OF THE PLANE FAILURE ANALYSIS

All the computer models were analyzed for a slope stability assessment against plane failure along potential slip plane. Figures 6, 7 and 8 show the results of a ‘Block Search’ indicating the ‘Global Minimum’ slip surfaces for 28°, 29° and 30° slopes respectively using the Bishop Analysis technique. Figure 6 also indicates the safety factors for the Global Minimum slip surface. Figure 6(a) indicates the safety factor of 1.331 for the slope angle of 28°, whereas the safety factors for slope angles of 29° and 30° are 1.292 and 1.266 respectively as shown in Figure 6(b and c). These results clearly indicate that the excavated slopes with slope angles $\leq 28°$ are acceptable against plane failure for SF $\leq 1.3$, whereas the excavated slopes with slope angles $\geq 29°$ are not acceptable against plane failure for SF $\leq 1.3$.

Figure 6 - Slope stability analysis of open pit slope in Thar lignite mine predicting plane failure
28° is safe for SF ≤ 1.3. However, Figures 8(b) and 8(c) show some slip surfaces, indicating the potential plane failure for slope angles of 29° and 30° respectively.

Figure 7 - Slope stability analysis of open pit slope in Thar lignite mine showing plane failure

Figure 8 - Slope stability analysis of open pit slope in Thar lignite mine predicting Plane failure

Stability Factor vs Location for 28° to 30° slopes indicating plane failure mode

Figure 9 shows the graph for various safety factors plotted against the horizontal location for slope angles of 28°, 29° and 30°. All the data points for a slope angle of 28° are above the minimum stability factor = 1.3, which means the pit slope with an inclination of 28° is quite safe for SF ≤ 1.3. However, some data points plotted for slope angles of 29° and 30° lie below the minimum safety factor of 1.3, therefore it can be concluded that the excavated slopes with slope angles ≥ 29° are not safe against the plane failure at SF ≤ 1.3.

Figure 9 - Stability factor along X-direction for plane failure from 28° to 30° slope angles

DEVELOPMENT OF COMPUTER MODELS USING CIRCULAR SLOPE FAILURE SURFACES

For the circular slope stability analysis, the computer software “Slide”, version five, was used to create five models incorporating various slope angles ranging from 23° to 27° and one model incorporating a combined slope angle of 23° in dune sand and 26° in the rest of the strata. Slope stability analysis was performed on the geologic section shown in Figure 5 and design parameters as indicated in Table 2 were used.
Following parameters were used for the development of models:

- Analysis technique: Simplified Bishop Method;
- Surface type: Circular and composite;
- Grid spacing: 20 x 20;
- Radius increment: 10.

Interpretation: Minimum surfaces and filter surfaces for $S_F \leq 1.3$ Figures 10 (a) and (b) show the minimum surfaces and filter surfaces respectively for a factor of safety $\leq 1.3$ and slope angle of $27^\circ$.

FIGURE 10 - Slip circles for various factors of safety, and slope angle of $27^\circ$

RESULTS AND DISCUSSION OF SLOPE STABILITY ANALYSIS USING CIRCULAR SHAPE OF FAILURE SURFACE

A slope stability analysis was executed for five slope angles ranging from $23^\circ$ to $27^\circ$ using Slide Version 5.0 Software. Slip circles for $S_F \leq 1.3$ were generated along the excavated slope for various slope angles. In a preliminary analysis, no slip circle was observed (diagram not shown), which means that for the excavated slope angle of $23^\circ$ with a factor of safety $\geq 1.3$ no slip was observed.

Figures 11 (a) to (c) show the development of slip circles in the dune sand region, which indicates the possible failure of the excavated slope in dune sand. It can be concluded from these results that the dune sand zone is not safe for $S_F < 1.3$ with slope angles greater than $23^\circ$. In Figure 11(d), an additional slip circle is observed, extending from the surface down to the sand layer of the strata, which indicates the possible failure of the excavation in this strata region. It is, therefore, concluded that the overall slope of the excavation is not safe for $S_F \leq 1.3$ with the slope angle greater than $26^\circ$. In view of the above discussion it can be concluded, that the dune sand zone is safe for the slope angle $\leq 23^\circ$ and rest of the strata is safe for slope angle $\leq 26^\circ$. It was decided to analyze the stability of the combined slope angles of $23^\circ$ and $26^\circ$. The results of this slope analysis, with combined angles of $23^\circ$ in dune sand and $26^\circ$ in rest of the strata, showed that there is no slip circle. This means that the combined slope is safe for $S_F \leq 1.3$ and slope angles $\leq 23^\circ$ in dune sand and $\leq 26^\circ$ in rest of the strata.

FIGURE 11 - Slip circles for various stability factors and slope angles
Stability factor versus location for 24° to 27° circular slopes

In order to evaluate the quantitative analysis of the stability of excavated slopes, factor of safety (FS) vs. location graphs were generated for various slope angles between 23° and 26°. These graphs are shown in Figure 12 (FS1 to FS6). In Figure 12, the FS 1 curve shows the minimum SF of 1.36 and the maximum FS of 2.1, which means the excavated slope of 23° is quite acceptable for the FS ≥ 1.36. It is clear from Figure 12, FS 2, for slope angle 24°, that the initial part of the graph is touching the Y-axis at SF=1.3, which shows that the dune sand zone of the strata is not acceptable at a slope angle of 24°. Figure 12, FS 3 and FS 4 indicate that the initial portion of the graph, comprising of dune sand, lies below FS=1.3, which means that the dune sand layer is not considered stable for slope angles of 25° and 26°. It is also observed from Figure 12, FS 5, that not only does the initial part (comprising of dune sand) of the graph lie below 1.15, but the other portion of the graph, at a location of X=260m, also falls below FS=1.3, which means that the overall slope is not considered stable at an inclination of 27°.

Figure 12 - Stability factor versus location for circular failure surface

The FS versus location graphs shown in Figure 12, FS 1 to FS 5 reveal that the dune sand layer of the strata is safe for SF=1.3 at slope angles ≤ 23° and rest of the strata is acceptable as safe for FS=1.3 at slope angles ≤ 26°. On the basis of these results it was necessary to evaluate the stability of the combined slope angles of 23° and 26°, therefore an FS vs. location graph was generated for the combined slope as shown in Figure 12 FS 6. It is very clear from the graph FS 6 in Figure 12 that no single point on the graph lies below the stability factor 1.3.

Sensitivity analysis

Table 3 - Sensitivity analysis of slopes at various angles in Dune Sand and other strata for different values of cohesion (C) and internal angle of friction (ϕ)

<table>
<thead>
<tr>
<th>STRATA</th>
<th>SLOPE ANGLE</th>
<th>10% ϕ</th>
<th>20% ϕ</th>
<th>30% ϕ</th>
<th>40% ϕ</th>
<th>Mean value ϕ</th>
<th>60% ϕ</th>
<th>70% ϕ</th>
<th>80% ϕ</th>
<th>90% ϕ</th>
<th>100% ϕ</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dune Sand</td>
<td>23°</td>
<td>0.2</td>
<td>0.5</td>
<td>0.75</td>
<td>1.05</td>
<td>1.363</td>
<td>1.700</td>
<td>2.20</td>
<td>2.85</td>
<td>3.50</td>
<td>4.65</td>
</tr>
<tr>
<td></td>
<td>24°</td>
<td>0.19</td>
<td>0.35</td>
<td>0.70</td>
<td>1.00</td>
<td>1.307</td>
<td>1.650</td>
<td>2.15</td>
<td>2.70</td>
<td>3.35</td>
<td>4.50</td>
</tr>
<tr>
<td></td>
<td>25°</td>
<td>0.10</td>
<td>0.34</td>
<td>0.68</td>
<td>1.00</td>
<td>1.238</td>
<td>1.62</td>
<td>2.10</td>
<td>2.55</td>
<td>3.20</td>
<td>4.30</td>
</tr>
<tr>
<td></td>
<td>26°</td>
<td>0.10</td>
<td>0.32</td>
<td>0.65</td>
<td>0.91</td>
<td>1.200</td>
<td>1.45</td>
<td>2.00</td>
<td>2.39</td>
<td>3.15</td>
<td>4.25</td>
</tr>
<tr>
<td></td>
<td>27°</td>
<td>0.15</td>
<td>0.28</td>
<td>0.63</td>
<td>0.90</td>
<td>1.136</td>
<td>1.44</td>
<td>1.84</td>
<td>2.36</td>
<td>3.05</td>
<td>4.20</td>
</tr>
<tr>
<td>Other Strata</td>
<td>23°</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
<td>1.363</td>
</tr>
<tr>
<td></td>
<td>25°</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
<td>1.238</td>
</tr>
<tr>
<td></td>
<td>27°</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
<td>1.136</td>
</tr>
</tbody>
</table>

Due to the poor quality of the geotechnical data obtained to date it was considered appropriate to carry out sensitivity analysis on assumed values of C and ϕ (average values as given in table 2 ± 10%) for
different slope angles. Table 3 shows the sensitivity analysis results for the slopes in dune sand and other strata for slope angles between 23° and 27° for different assumed values of cohesion and internal angle of friction. Table 3 indicates that pit slopes having inclination of 23° and 24° are safe for the calculated stability factor of 1.3 for the average values of C and Ø. Table 3 also indicates that the stability factors at 50% Ø are 1.238, 1.2 and 1.136 for the pit slopes of 25°, 26° and 27° respectively in the dune sand which means that Dune sand slopes, having inclination > 24°, are not safe.

Sensitivity curves of cohesion (C) and angle of internal friction (φ) for different strata layers with combined slope angles are shown in Figure 13 indicating that stability factor for the mean values of cohesion (C) and internal angle of friction (Ø) in dune sand and other strata is 1.345 which means it is safe for stability factor of 1.3.

CONCLUSIONS

The computer software package SLIDE version 5 has been conveniently used to carry out a slope stability analysis of the high walls of the open cut operations in the Thar Coalfield. Three, computer models with slope angles of 28°, 29° and 30° incorporating plane failure modes have been used to analyze the stability of the pit’s high wall slopes. This model assisted in concluding that the slope angle of 28° is safe for a factor of safety ≤ 1.3 whereas the excavated slopes with slope angle ≥ 29° are not safe against the plane failure for SF ≥ 1.3. The analyses using the circular failure mode were carried out on five models for various slope angles ranging from 23° to 27° and one combined model containing a slope angle of 23° in dune sand and 26° in the rest of the strata, and concluded that the dune sand layer, which is 48m thick, is safe for SF≤1.3 at slope angle 23°, while the rest of the strata is safe for the stability factor of 1.3 for slope angles 26°. Sensitivity analysis indicated that the stability factor for sand dune slope range from 0.2 to 4.65 for angle of friction variation from 10% to 100% φ values.

It may also be noted that the sensitivity analysis of slopes at various angles in Dune Sand and other strata, for different values of cohesion (C) and internal angle of friction (φ), revealed that the slopes with inclination ≤ 24° are safe where as combined slope is safe for inclination of 23° in dune sand and 26° in rest of the strata. The overburden to lignite extraction ratio for the present slope design was estimated as 3:1 or 3 m³ of overburden to be removed to recover 1 t of lignite (Pathan, et al., 2007).

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Thanks are due to the School of Civil Engineering for inviting the first author to Nottingham Centre of Geomechanics and for encouraging the co-operative research with the Mehran University of Engineering and Technology, Sindh, Pakistan on the Thar project. Thanks are also due to the British
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REFERENCES

PREDICTION OF ROCK MASS RATING USING FUZZY LOGIC WITH SPECIAL ATTENTION TO DISCONTINUITIES AND GROUND WATER CONDITIONS

Hossein Jalalifar¹², Saeed Mojeddifar¹, Ali Akbar Sahebi¹

ABSTRACT: The Rock Mass Rating (RMR) system is a classification based on the six parameters which was defined by Bieniawski. This system may possess some fuzziness in its practical applications. For example, experts mostly relate discontinuities and ground water conditions in linguistic terms with approximation. Descriptive terms vary from one expert to the other, while in the RMR system; values which are related to these terms are probably the same. The other hand, sharp transitions between two classes create uncertainties. So it is proposed to determine weighting intervals for discontinuities and water condition. Two fuzzy models based on the Mamdani algorithm were introduced to evaluate proposed weights, so that the first fuzzy model includes 55 scores using fuzzy model and the remained scores which are related to discontinuities and ground water conditions are obtainable by the RMR system. But the second fuzzy model obtains all scores of the RMR system using fuzzy model. Results of fuzzy models are adapted with actual RMR, but second fuzzy model predicts more acceptable results, because it has the ability to use qualitative terms in fuzzy state. But first fuzzy model uses descriptive terms in classic state. So it seems, proposed weighting intervals can manage fuzzification of discontinuities and water conditions.

INTRODUCTION

Bieniawski developed a Rock Mass Rating (RMR) system based on six parameters: (1) The Uniaxial Compressive Strength of intact rock (UCS), (2) Rock Quality Designation (RQD), (3) Joint or Discontinuity Spacing (JS), (4) Joint Condition (JC), (5) Ground Water Condition (GW) and (6) Joint Orientation (JO). He assigned numerical rating values to all these parameters. Based on the value of the rock mass rating, Bieniawski divides the whole universe of rock mass into five classes, and then assigns stand up time to each class (Hudson and Harrison, 2005).

The arithmetic sum of the rating corresponding to the five main parameters is referred to as “the basic RMR” (Figure 1). But the total RMR is obtained by adjusting the basic RMR for the influence of joint orientation for a specific excavation face (Figure 1) (Aydin, 2004).

Bieniawski Rock Mass Classification often involves criteria whose values are assigned in linguistic terms and the other hand, sharp class boundaries are a subjective uncertainty in rock mass classification. Fuzzy set theory enables a soft approach to account for these uncertainties. Actually, fuzzy sets make them more objective, particularly through the process of construction of Membership Functions (MFs).

Figure 1 - RMR classification for characterization and design purposes (after Aydin, 2004)

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UCS, RQD and JS are the numerical criteria but JC and GW are expressed mostly in descriptive terms. For fuzzification of descriptive criteria (JC and GW), it is necessary to consider these criteria quantitatively.

In this study, for quantitation of JC and GW, the weights are proposed for each of them. Also for validation of these weights two fuzzy models are introduced, so that, the first fuzzy system obtains basic RMR without descriptive criteria and the second one obtains basic RMR with descriptive criteria.

**FUZZY SET THEORY**

The fuzzy set was introduced as a mathematical way to represent linguistic vagueness. The fuzzy logic is useful to process the imprecise information by selecting a suitable MF. In a classical set, an element belongs to or does not belong to a set. That is, the membership of an element is crisp, 0 or 1, against; a fuzzy set is a generalization of an ordinary set which assigns the degree of membership for each element to range over the unit interval between 0 and 1 (Iphar and Goktan, 2006). The membership or non membership of an element \( x \) in the crisp set \( A \) is represented by the characteristic function \( \mu_A \) of \( A \), defined by (Acaroglu et al., 2008)

\[
\mu_A(x) = \begin{cases} 
1 & \text{If } x \in A \\
0 & \text{If } x \notin A
\end{cases}
\]

Where \( \mu_A(x) \) is the membership degree of the variable \( x \). An MF fulfills fuzzification of input/output variables. The shapes of the MFs normally were considered trapezoidal or triangular. Describing input–output relationship, conditional rules is an important aspect in the fuzzy system. The fuzzy proposition is represented by a functional implication called as fuzzy “if–then” rule (Iphar and Goktan, 2006).

**FUZZY INFERENCE SYSTEM**

The Fuzzy Inference System (FIS) is a famous computing system which is based on the concepts of fuzzy set theory, fuzzy if–then rules, and fuzzy reasoning (Ross, 1995). Several FIS have been employed in different applications. The most common models are the Mamdani fuzzy model, Takagi–Sugeno–Kang (TSK) fuzzy model, Tsukamoto fuzzy model and Singleton fuzzy model (El-Shayeb, et al., 1997), but among the aforesaid models, Mamdani is one of the most common algorithms used in fuzzy systems. The Mamdani fuzzy algorithm takes the following form (Iphar and Goktan, 2006).

\[
R_i = \text{If } x_1 \text{ is } A_{i1} \text{ and } x_2 \text{ is } A_{i2} \text{ then } y \text{ is } B_i \quad \text{ (for } i=1, 2, \ldots, k) \tag{2}
\]

Where: \( x_1 \) and \( x_2 \) are input variables, \( A_{i1} \), \( A_{i2} \) and \( B_i \) are linguistic terms (fuzzy sets), \( y \) is output variable and \( k \) is the number of rules. Figure 2 is an illustration of a two-rule Mamdani FIS which derives the overall output “\( z \)” when subjected to two crisp inputs “\( x \)” and “\( y \)” (Jang et al., 1997). Inputs in the FIS, “\( x \)” and “\( y \)”, are crisp values. The rule-based system is described by Eq. 2. For a set of disjunctive rules, the aggregated output for the “\( k \)” rules is given by

\[
\mu_{Ak}(z) = \max_k\{\min\{\mu_{A1}(input(x)), \mu_{A2}(input(x))\}\} \quad \text{ (for } k=1,2,\ldots,t) \tag{3}
\]

Where: \( \mu_{A1},\mu_{A2} \) and \( \mu_{Bk} \) are the membership function of output “\( z \)” for rule “\( k \)”, input “\( x \)” and input “\( y \)”, respectively. Eq. 3 has a simple graphical interpretation as shown in Figure 2.

In Figure 2 symbols A1 and B1 refer to the first and second fuzzy antecedents of the first rule, respectively. The symbol C1 refers to fuzzy consequent of the first rule, A2 and B2 refer to the first and second fuzzy antecedents of the second rule, respectively, C2 refers to fuzzy consequent of the second rule. The minimum membership value for the antecedents propagates through to the consequent and truncates the MF for the consequent of each rule. Then the truncated MFs for each rule are aggregated. In Figure 2, the rules are disjunctive so the aggregation operation max results in an aggregated MF comprised of the outer envelope of the individual truncated membership forms from each rule. If a crisp value is needed for the aggregated output, some appropriate defuzzification technique should be employed to the aggregated MF (Ross, 1995). There are several defuzzification methods such as Centroid of Area (COA) or Center of Gravity, Mean of Maximum, Smallest of Maximum, etc (Grima,
2000, Hellendoorn and Thomas, 1993). In Figure 2, the COA defuzzification method is used for obtaining the numeric value of output.

![Figure 2 - The Mamdani FIS (after El-Shayeb et al., 1997)](image)

**QUANTITATION OF DESCRIPTIVE CRITERIA**

Experts mostly relate JC and GW condition in linguistic terms with approximation and possibility. It means expression descriptive terms vary from one expert to the other, while in the RMR system (Bieniawski, 1989), values which are related to these terms are probably the same.

Therefore it is proposed to determine weighting intervals for descriptive classes in RMR system (Bieniawski, 1989) (JC and GW). Table 1 shows the proposed weights for qualitative criteria. As can be seen in table.1 JC and GW criteria are weighted in intervals [0, 1] and [0, 0.8], respectively.

**Table 1 - Proposed weights for JC and GW**

<table>
<thead>
<tr>
<th>Descriptive of JC</th>
<th>Smooth soft filling separation ≤5</th>
<th>Smooth to slightly rough, soft filling mud - weather</th>
<th>Slightly rough, highly-weathered separation &lt;1mm</th>
<th>Slightly rough slightly weathered separation &lt;1mm</th>
<th>Very rough unweathered separation &lt;0.1mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>RMR score</td>
<td>0</td>
<td>10</td>
<td>20</td>
<td>25</td>
<td>30</td>
</tr>
<tr>
<td>Proposed weighting</td>
<td>0-0.1</td>
<td>0.1-0.45</td>
<td>0.45-0.65</td>
<td>0.65-0.8</td>
<td>0.8-1</td>
</tr>
<tr>
<td>Descriptive of GW</td>
<td>Flowing</td>
<td>Dripping</td>
<td>Wet</td>
<td>Damp</td>
<td>Completely rry</td>
</tr>
<tr>
<td>RMR score</td>
<td>0</td>
<td>4</td>
<td>7</td>
<td>10</td>
<td>15</td>
</tr>
<tr>
<td>Proposed weighting</td>
<td>0-0.1</td>
<td>0.1-0.25</td>
<td>0.25-0.4</td>
<td>0.4-0.6</td>
<td>0.6-0.8</td>
</tr>
</tbody>
</table>

**CONSTRUCTION OF TWO MAMDANI FIS FOR RMR PREDICTION**

Two fuzzy models based on the Mamdani algorithm are introduced and applied for basic RMR prediction. In both models, the COA defuzzification method is used for obtaining the numeric value of output and also “min” and “max” are employed as “and” and “or”, respectively. The crisp value adopting the COA defuzzification method was obtained by (Grima, 2000)

\[
Z_{*}^{COA} = \frac{\int \mu_{A}(z)Z \, dz}{\int \mu_{A}(z) \, dz}
\]  

(4)

Where: \(Z_{*}^{COA}\) is the crisp value for the “z” output and, \(\mu_{A}(z)\) is the aggregated output MF.
As can be seen in Figure 3, the first fuzzy model (FIS A) constructed with three inputs and one output and the second fuzzy model (FIS B) constructed with five inputs and one output. To estimate RMR; UCS, RQD and JS are used as input parameters for FIS A and three aforesaid inputs mid JC and GW are used as input parameters for FIS B.

![Figure 3 - Main structure of fuzzy models: (a) FIS A; (b) FIS B](image)

In Figure 4 the MFs of input parameters were abbreviated and indicated. As can be seen in Figure 4, triangular and trapezoidal MFs were considered appropriate for the proposed fuzzy models. For example “VB” is used for “very bad”, “B” for “bad”, “M” for “medium”, “G” for “good” and “E” for “excellent”.

![Figure 4 - Fuzzy input parameters: (a) MFs of RQD; (b) MFs of UCS; (c) MFs of JS; (d) MFs of GW; (e) MFs of JC](image)

In both models, the output MFs consists of eight fuzzy sets (Figure 5) in terms “Very Very Bad”, “Very Bad”, “Bad”, “Medium”, “Good”, “Very Good”, “Very Very Good” and “Excellent”. Also triangular and trapezoidal MFs were considered for the fuzzy model outputs. The range of rating FIS A output is interval [0, 55], but FIS B output is interval [0, 100]. A total of 125 rules for FIS A and 375 rules for FIS B were utilized and a decision was made out of the combined input(premise part) and output(consequent part) membership functions based on expert experience and the applied database. An example of the if-then rules in FIS A and FIS B is as follows:

Rule of FIS A: If (UCS is G) and (RQD is M) and (JS is E) then (SCORE is VVG)  

(5)

Rule of FIS B: If (UCS is E) and (RQD is G) and (JS is B) and (JC is B) and (GW is B) then (SCORE is G)  

(6)
In FIS A, the range of rating output belongs to the interval \([0, 55]\), and to achieve a basic RMR, it is necessary to add FIS A output to the sum of scores obtained from JC and GW. It should be noted that scores of JC and GW, which are achieved from the RMR system, were used as the reference classification structure. But FIS B output is equal to the basic RMR. The shape and range of FIS A inputs MFs are equal to the first three FIS B inputs MFs; in addition to this, the shapes of output MFs for both models are the same. But the range of output MFs in the FIS A model belongs to the interval \([0, 55]\) and the range of output MFs in the FIS B model belongs to the interval \([0, 100]\). So the range of JC and GW MFs in FIS B are modified upon proposed weights in Table 1. In fact, the purpose of expression of two fuzzy models (FIS A and FIS B) in this study is evaluation of proposed weights for the JC and GW. In other word, FIS A and FIS B are compared to show the importance of fuzzification JC and GW parameters in the classification system. FIS A obtains 55 scores with the use of fuzzy model and the rest of scores, which consist of descriptive terms obtained by the RMR system, but FIS B obtains the total scores of the RMR system (Bieniawski, 1989) from fuzzy model.

SIMULATION RESULTS

To validate and compare the acquired results between the FIS A and FIS B models, correlation \(R^2\) and Root Mean Square Error (RMSE) can be used. Here \(R^2\) is used to validate the predictive models based on the comparing predicted and measured (real) values, whereas, RMSE is used to compare the result of FIS A and FIS B models. RMSE is calculated by the following equation:

\[
RMSE (A) = \sqrt{\frac{1}{n} \sum_{i=1}^{n} (A_{\text{meas}} - A_{\text{ipred}})^2}
\]  

(7)

Where: \(A_{\text{meas}}\) is the ith measured element, \(A_{\text{ipred}}\) is the ith predicted element and \(n\) is the number of dataset.

Evaluating the performance of proposed models has been done using database from the TABAS Coal Mine (Daws, 1992). The data testing has about 20 datasets. Table 2 show the results of the FIS A and FIS B models. RMSE are 5.37 and 3.32 for FIS A and FIS B, respectively. As can be seen in Figure 6, correlation coefficients are 0.922 and 0.959 for FIS A and FIS B, respectively, which shows a very good agreement.
### Table 2- Results of FIS A and FIS B

<table>
<thead>
<tr>
<th>Data NO</th>
<th>Real RMR</th>
<th>Fis A RMR</th>
<th>Fis B RMR</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>79</td>
<td>81.00</td>
<td>77.00</td>
</tr>
<tr>
<td>2</td>
<td>96</td>
<td>87.82</td>
<td>85.79</td>
</tr>
<tr>
<td>3</td>
<td>69</td>
<td>71.34</td>
<td>68.02</td>
</tr>
<tr>
<td>4</td>
<td>47</td>
<td>45.00</td>
<td>47.09</td>
</tr>
<tr>
<td>5</td>
<td>47</td>
<td>45.00</td>
<td>47.58</td>
</tr>
<tr>
<td>6</td>
<td>50</td>
<td>53.00</td>
<td>50.00</td>
</tr>
<tr>
<td>7</td>
<td>47</td>
<td>49.62</td>
<td>48.28</td>
</tr>
<tr>
<td>8</td>
<td>56</td>
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<tr>
<td>9</td>
<td>45</td>
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<td>10</td>
<td>31.5</td>
<td>28.74</td>
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<td>11</td>
<td>24</td>
<td>14.18</td>
<td>13.57</td>
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<tr>
<td>12</td>
<td>54</td>
<td>55.99</td>
<td>50.43</td>
</tr>
<tr>
<td>13</td>
<td>44</td>
<td>53.27</td>
<td>50.00</td>
</tr>
<tr>
<td>14</td>
<td>39</td>
<td>35.86</td>
<td>39.94</td>
</tr>
<tr>
<td>15</td>
<td>44</td>
<td>45.73</td>
<td>51.41</td>
</tr>
<tr>
<td>16</td>
<td>54</td>
<td>63.78</td>
<td>51.15</td>
</tr>
<tr>
<td>17</td>
<td>40.5</td>
<td>33.00</td>
<td>38.83</td>
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<tr>
<td>18</td>
<td>68</td>
<td>69.59</td>
<td>64.27</td>
</tr>
<tr>
<td>19</td>
<td>53.5</td>
<td>59.92</td>
<td>50.75</td>
</tr>
<tr>
<td>20</td>
<td>24</td>
<td>22.17</td>
<td>21.44</td>
</tr>
</tbody>
</table>

### CONCLUSIONS

- In this study the fuzzy set theory is applied to one of the conventional RMR system (Bieniawski, 1989) by two fuzzy models. RMSE was obtained equalled 3.32 and 5.37 for FIS B and FIS A respectively. Moreover, $R^2$ was 0.959 and 0.922 for FIS B and FIS A respectively.

- The results of fuzzy models were in good agreement with actual RMR. However, FIS B predicts more valuable results, which is due to the application of qualitative terms in the fuzzy model, while, FIS A uses descriptive terms in the classic method.

- It seems, weighting intervals which were proposed for JC and GW, can manage fuzzification of JC and GW, because these proposed weights solve problem of sharp transitions between two adjacent excavation classes and the subjective uncertainties on data that are close to the range boundaries of rock classes.

### REFERENCES


EXTERNAL REINFORCEMENT OF CONCRETE COLUMNS

Muhammad N S Hadi

ABSTRACT: With the technology development on the compressive strength of concrete over the years, the use of high strength concrete has proved most popular in terms of economy, superior strength, stiffness and durability due to many advantages it could offer. However, strength and ductility are inversely proportional. High strength concrete is a brittle material causing failure to be quite sudden and ‘explosive’ under loads. It is also known that structural concrete columns concentrically compressed rarely occur in practice. The stress concentrations caused by the eccentric loading further reduce the strength and ductility of high strength concrete. Therefore, studies for high strength concrete columns under eccentric loading are essential for the practical use. A number of high strength concrete columns that are externally reinforced with galvanised steel straps and fibre reinforced polymers subjected to concentric and eccentric loading are experimentally investigated. The experimental results show that external reinforcement can enhance the properties of high strength concrete columns.

INTRODUCTION

Since high strength concrete has become a familiar phase in concrete technology in the late 1980s, the application of high strength concrete in the construction industry has steadily increased over the past two decades. The wide application for high-strength concrete has stimulated a number of research studies in many countries including Australia during the last few years. However, the studies are not enough to predict the behaviour of the material with reasonable accuracy. As a consequence, important issues related to design and construction of high strength concrete structures are not adequately addressed in building codes, therefore, structural designers are unable to take full advantage of the material because of the inadequate information.

The increase in brittleness with the increase of strength of concrete is of major concern in using the high strength concrete. The lack of ductility of high strength concrete results in sudden failure without warning, which is a serious drawback of high strength concrete. Extensive previous studies have shown that addition of compressive reinforcement and confinement will increase the ductility as well as the strength of material effectively (Razvi and Saatcioglu, 1994; Hadi and Schmidt, 2002). The higher the concrete strength, the more it becomes necessary to provide confinement (Attard and Mendis, 1993). Confining the concrete can reduce its brittleness. In the recent years, considerable attention has been focused on the external reinforcement, as one of the methods of confinement, which has been proved by previous studies as an effective method to enhance the structural properties of high strength concrete members (Pessiski, et al., 2001).

Externally reinforcing high strength concrete enhances the properties of concrete columns, most importantly reducing the effect of its brittle behaviour, and allowing the column to attain maximum load carrying capacity. These higher strengths are achieved as a result of the lateral pressures, applied by the external reinforcement, to the extreme fibres of the concrete column. The confinement prevents the lateral expansion of the specimen under axial load, improving the column's stiffness. As a result, the high strength concrete column is able to carry higher loads than if it were unreinforced.

Among the various external reinforcements, steel straps and fibre reinforced polymers are being used popularly. Previous studies have shown that external steel reinforcement increases a column's strength and enables the steel straps to be smaller in size than internal steel reinforcement. As the corrosion for the steel straps resulted in bond deterioration, the steel is galvanised to resist against corrosion. In recent years, fibre-reinforced polymers wrapping in lieu of steel jacket has become an increasingly popular method for external reinforcement in which fibre reinforced polymers offer improved corrosion and fatigue resistance compared to the steel reinforcement (Pantazopoulou, et al., 2001). The high tensile strength and low weight make fibre reinforced polymers ideal for use in the construction industry. Another attractive advantage of fibre reinforced polymers over steel straps as external reinforcement is its easy handling, thus minimal time and labour are required to implement them.
This study considers various types of external reinforcement and compares them with experimental results. The effect of the two types of external reinforcing material, galvanised steel straps and fibre reinforced polymers are evaluated. Two sets of tests in terms of eccentric loading and concentric loading are conducted. Then, the effectiveness of the external reinforcement as a confining material under different loading conditions is investigated.

**EXPERIMENTAL PROGRAMME**

In order to test the performance of concrete columns confined with various reinforcing materials, two sets of tests were designed: five cylindrical concrete columns of 205 mm diameter and 910 mm height were tested under concentric loading. Another set of six cylindrical concrete columns of 205 mm diameter and 920 mm height were tested under eccentric loading with an eccentricity of 50 mm. The configuration of external reinforcement varies for both sets of columns as well. The testing variables selected for this study are: (1) the type of external reinforcement: galvanised steel straps and fibre reinforced polymers, (2) the number of layers for FRP, (3) the spacing of steel straps, (4) the types of FRP materials and (5) the loading pattern.

In order to have a better insight about the contribution of FRP on confinement and to be able to conduct the theoretical analysis on the behaviour of the column specimens in the main testing program of this study, a preliminary testing on the all reinforcing materials used in this study was conducted as well.

**Columns' details**

**Concentrically loaded columns**

Five columns without internal reinforcement were designed for this testing. Each column had a diameter of 205 mm and a height of 910 mm. Four columns continually wrapped with FRP had the following configurations: one-layered and three-layered Carbon fibres, one-layered and three-layered Kevlar Fibres. The remaining plain column was used as control column. The configuration of this set of columns is summarised in Table 1.

**Eccentrically loaded columns**

Six concrete columns were cast and tested. Three of the columns were wrapped with three layers of unidirectional fibre reinforced polymers. Two were externally reinforced with galvanised steel straps, each steel strap was 20 mm wide and 0.5 mm thick. Two spacings for the steel straps were used: 10 mm and 20 mm. The final column was internally reinforced with steel helix and longitudinal reinforcement. All the columns were eccentrically loaded until failure with an eccentricity of 50 mm. The testing matrix is summarised in Table 2.

### Table 1 - Configuration of the concentrically loaded columns

<table>
<thead>
<tr>
<th>Column</th>
<th>Diameter (mm)</th>
<th>Height (mm)</th>
<th>Reinforcing Type</th>
<th>Reinforcing Material</th>
<th>Loading Pattern</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>205</td>
<td>910</td>
<td>External</td>
<td></td>
<td>Concentric</td>
</tr>
<tr>
<td>2</td>
<td>205</td>
<td>910</td>
<td>External</td>
<td>Single-layered Carbon</td>
<td>Concentric</td>
</tr>
<tr>
<td>3</td>
<td>205</td>
<td>910</td>
<td>External</td>
<td>Single-layered Kevlar</td>
<td>Concentric</td>
</tr>
<tr>
<td>4</td>
<td>205</td>
<td>910</td>
<td>External</td>
<td>Three-layered Carbon</td>
<td>Concentric</td>
</tr>
<tr>
<td>5</td>
<td>205</td>
<td>910</td>
<td>External</td>
<td>Three-layered Kevlar</td>
<td>Concentric</td>
</tr>
</tbody>
</table>

### Table 2 - Testing matrix of the eccentrically loaded columns

<table>
<thead>
<tr>
<th>Column</th>
<th>Diameter (mm)</th>
<th>Height (mm)</th>
<th>Reinforcing Type</th>
<th>Reinforcing Material</th>
<th>Loading Pattern</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>205</td>
<td>920</td>
<td>External</td>
<td>Three-layered Carbon</td>
<td>Eccentric</td>
</tr>
<tr>
<td>2</td>
<td>205</td>
<td>920</td>
<td>External</td>
<td>Three-layered E-glass</td>
<td>Eccentric</td>
</tr>
<tr>
<td>3</td>
<td>205</td>
<td>920</td>
<td>External</td>
<td>Three-layered Kevlar</td>
<td>Eccentric</td>
</tr>
<tr>
<td>4</td>
<td>205</td>
<td>920</td>
<td>External</td>
<td>Galvanised Steel Straps at 10 mm Spacing</td>
<td>Eccentric</td>
</tr>
<tr>
<td>5</td>
<td>205</td>
<td>920</td>
<td>External</td>
<td>Galvanised Steel Straps at 20 mm Spacing</td>
<td>Eccentric</td>
</tr>
<tr>
<td>6</td>
<td>205</td>
<td>920</td>
<td>Internal</td>
<td>6 N12 Bars and N10 Helix</td>
<td>Eccentric</td>
</tr>
</tbody>
</table>
Eccentric loading

Where eccentric loading differs from concentric loading is that it involves concentrating the load a certain distance from the neutral axis of the cross section. As shown in Figure 1, two plates were designed and manufactured in order to apply eccentric loading on the columns. These plates are used on either end of the columns during loading.

![Eccentric Loading Image](image)

**Figure 1 - Steel end plates for eccentric loading**

**Specimen preparation**

Two batches of concrete were used to cast the concentrically and eccentrically loaded columns. The design compressive strength of both batches of concrete was 100 MPa. However, 73.62 MPa and 51 MPa were achieved for the concentrically loaded and eccentrically loaded columns, respectively.

The three types of fibre reinforced polymers used in this study were Carbon, Kevlar and E-glass. The epoxy system consisted of two parts, resin and slow hardener, were used to bond the FRP to the surface of the concrete columns. The process of applying the FRP is known as the wet lay up method and was used to wrap all the columns with external FRP confinement.

The band-it method was employed to apply the galvanised straps on the two concrete columns in this study. The galvanised steel straps were placed along the length of column at 20 mm spacing for one column and 10 mm spacing for the second column.

**Specimen testing**

The testing program consisted of testing the five concrete columns under concentric loading and testing the six cylindrical concrete columns with different external confinement under the eccentric load. The hydraulically operated 5000 kN Denison compression testing machine, located in the Engineering Laboratory at the University of Wollongong was used to test all the columns in this study. All the columns were tested to failure.

**OBSERVED BEHAVIOUR AND TEST RESULTS**

The failure of the columns in all cases was brittle and in the case of the plain specimen, a very explosive failure. In the case of the FRP confined columns, the snapping of the fibres could be heard throughout the loading as the concrete tried to expand. While for the two galvanised steel straps reinforced columns, failure was sudden and soundless. In each case the straps may have yielded but did not break. This type of failure suggested that the failure of the columns was a direct result of cracking of the concrete tensile flexure. This type of failure can be explained that this type of reinforcement may not be suitable for columns under eccentric load. Table 3 and Table 4 present the testing results of the concentrically loaded and eccentrically loaded columns, respectively.
Table 3 - Testing results for the concentrically loaded columns

<table>
<thead>
<tr>
<th>Column</th>
<th>Configuration</th>
<th>Ultimate Load (kN)</th>
<th>Axial Deflection (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Plain</td>
<td>2351</td>
<td>5.048</td>
</tr>
<tr>
<td>2</td>
<td>Single Layered Carbon</td>
<td>2860</td>
<td>4.404</td>
</tr>
<tr>
<td>3</td>
<td>Single Layered Kevlar</td>
<td>2490</td>
<td>4.234</td>
</tr>
<tr>
<td>4</td>
<td>Three Layered Carbon</td>
<td>2980</td>
<td>6.514</td>
</tr>
<tr>
<td>5</td>
<td>Three Layered Kevlar</td>
<td>2490</td>
<td>5.574</td>
</tr>
</tbody>
</table>

Table 4 - Testing results for the eccentrically loaded columns

<table>
<thead>
<tr>
<th>Column</th>
<th>Configuration</th>
<th>Ultimate Load (kN)</th>
<th>Axial Deflection (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Three Layered Carbon</td>
<td>840.0</td>
<td>5.20</td>
</tr>
<tr>
<td>2</td>
<td>Three Layered E-glass</td>
<td>630.8</td>
<td>4.38</td>
</tr>
<tr>
<td>3</td>
<td>Three Layered Kevlar</td>
<td>906.0</td>
<td>5.50</td>
</tr>
<tr>
<td>4</td>
<td>Galvanised Steel Straps at 10 mm spacing</td>
<td>720.0</td>
<td>4.22</td>
</tr>
<tr>
<td>5</td>
<td>Galvanised Steel Straps at 20 mm Spacing</td>
<td>704.9</td>
<td>3.94</td>
</tr>
<tr>
<td>6</td>
<td>Internally Reinforced</td>
<td>636.8</td>
<td>--</td>
</tr>
</tbody>
</table>

The concentrically loaded plain concrete column, as expected, presented a very brittle explosive failure. The column did not experience any excess deflection after reaching the maximum compressive load due to the lack of confinement, which led to the brittle failure.

For the single layered Carbon fibres column under concentric loading, the external confinement provided to this column resulted in a higher ultimate load. However, the failure was still quite explosive and resulted in no increased deflection after reaching the maximum load as well.

The single layered Kevlar column under concentric loading achieved a slight increase in ultimate load over the plain specimen. And the most promising aspect about this column is that there was a small amount of excess deflection achieved after ultimate load. The failure was less explosive and the column was almost fully confined even after failure.

The three layered Carbon column under concentric loading achieved significantly better results than the single layered specimen both in strength and deflection. It is of significance to note that the column still appeared to be fully intact after failure. Upon closer inspection, it could be seen that the jacket had a section where the jacket has frayed rather than actually fractured. This meant that even after failure the column still had the ability to withstand load and still maintain its integrity.

The three layered Kevlar specimen under concentric loading also out-performed the single layered specimen, also achieved higher strength and ductility. However, there was not as much excess deflection achieved as the Carbon wrapped specimen. This specimen also remained intact after failure except for the presence of small fractures in the jacket.

This internally reinforced specimen under eccentric loading specimen exhibited a brittle failure under the eccentric loading. The concrete cover started to fall away due to lateral dilation under the loading. However, even after the concrete cover had spalled away, the confined core continued to carry an increasing load. Figure 2 shows the column after failure.

![Figure 2 - Failure of the internally reinforced concrete column](image-url)
The failure of the Carbon fibre jacketed concrete columns under Eccentric loading was marked by brittle rupture of the hardened fibres at the bottom of the column, which can be seen in Figure 3. Failure was sudden and quite explosive. During various stages of loading, the snapping sounds could be heard, which were attributed to the cracking of the concrete and the stretching of the hardened fibres. The test results show that this column could withstand much higher ultimate load than the internally reinforced column. This finding reveals that carbon confinement could provide significantly greater confining pressure to the high strength concrete column.

Figure 3 - Failure of the carbon fibres column

For the E-glass wrapped specimen under eccentric loading, the failure of the E-glass wrapped column specimen was marked by fibre rupture at the top of the column. Although it was sudden, the failure could be predicted by the appearance of white patches at the top of the column as the result of the fibre stretching. From Figure 4, it can be seen the layers of E-glass were torn as a result of the eccentric load applied to the column. As the external E-glass confinement tried to prevent the concrete from expansion under loading, it was ruptured when the tensile stress, applied by the concrete lateral expansion, became too large. The results show that the load carrying capacity of this column wrapped with E-glass was slightly lower than the internally reinforced column.

Figure 4 - Failure of the E-glass wrapped column

For the Kevlar wrapped concrete column under eccentric loading, the material used to wrap this column is one sheet of Kevlar in 920 mm wide rather than the roll of tape. The failure mode of this column was similar to that of E-glass specimen: fibres were ruptured at the top end of the column. This can be seen in Figure 5. Cracking of Kevlar fibre could be heard throughout the testing with the failure of the column signified by a loud snap of the Kevlar jacket. The largest load carrying capacity was achieved by this column compared to other eccentrically loaded columns. This is contributed to the external confinement in terms of continuous sheet. Also, the failure was sudden and loud.

Figure 5 - Failure of the Kevlar wrapped column

For the galvanised steel strapped column at 10 mm spacing under eccentric loading, it was found this column failed in the tensile bending region under eccentric load. Figure 6 shows the failure of this column occurred in the space of two straps. This can be explained as a result of there being no
reinforcement in this region. However, the crack was much smaller and failure occurred closer to the bottom of the column when compared to the column with steel straps in 20 mm spacing. The failure was brittle and soundless.

Figure 5 - Failure of the Kevlar wrapped column

For the galvanised steel straps at 20 mm spacing under eccentric loading, the failure of this column was similar to that of another galvanised steel straps wrapped column, in that the cracking of the concrete on the tension side marked the failure. Also evident in Figure 7, is that failure, again occurred in between the galvanised steel straps and the straps themselves again, did not show any sign of failing. As the increased spacing of straps resulted in a larger area of the column being un-reinforced, the column failed in a substantial crack in the concrete. The results shown in Table 4 confirm that the larger the spacing between the straps, results in a lower load carrying capacity.

Figure 6 - Failure of the steel strapped column with 10 mm spacing

Figure 7 - Failure of the steel strapped column with 20 mm spacing
COMPARISON AND ANALYSIS

From the two sets of experiments conducted in this study, it can be noted that the Carbon wrapped columns outperformed the other types of reinforced columns except the Kevlar sheet wrapped column, which was proved by the testing results of the eccentrically loaded columns. The testing results indicated that Carbon fibres wrapping is more effective for the external confinement compared to the galvanised steel straps and E-glass. However, this is not the case in the Kevlar sheet wrapping column, which presented the largest loading capacity as the continuous sheet was used as the external confinement. Again, this finding proved the layout of fibres has a significant influence on the behaviour of the eccentrically loaded columns.

The comparison among the eccentrically loaded columns shows that all the externally reinforced columns outperformed the internally reinforced column excluding the E-glass specimen, which almost achieved the same strength. The E-glass was confirmed to be the weakest reinforcing material, which presented an ultimate load 10% lower than that of the two Band-It columns and a 44% decrease in compressive load compared to the Kevlar fibre sheet confined column.

Another comparison made between the two galvanised steel straps wrapped columns shows that the larger the spacing between the straps results in a lower load carrying capacity. However, the column with galvanised steel straps in 20 mm spacing exhibited only 2.2% decrease in the loading carrying capacity. Nonetheless, there was a 26% and 16% decrease in ultimate load over the Kevlar fibre sheet and carbon fibre confined columns respectively. And both columns achieved slightly higher ultimate load compared to the internally reinforced column, which proved the external confinement with galvanised steel straps is also more effective than the internal reinforcement. But the failure of the columns with this type of reinforcement is sudden, which indicates that the galvanised steel straps have very little effect on improving the ductility of the columns.

The comparison among the concentrically loading columns confirmed that the confinement significantly enhances the strength, stiffness and ductility of high strength concrete, in particular when applied in multiple layers.

CONCLUSIONS

The work carried out in this study involved two sets of testing: five columns under concentric loading and six columns under eccentric loading, which are mainly set to evaluate the effectiveness of various types of the external reinforcement. The results from both sets of tests allow the following conclusions to be drawn:

- The methods of external reinforcement can be used as an alternative method of reinforcement to enhance the properties of high strength concrete. It has been shown that the confinement of the concrete prevents the concrete from expanding and therefore allows the concrete to absorb higher stresses, resulting in a higher load carrying capacity.
- The tests proved that the benefits of confinement could be enhanced by applying multiple layers, which can be seen from the results of testing the concentric loading columns.
- The test results also indicated that the Carbon fibres provides the greatest amount of confinement, and had significantly better results, if the external confinement was achieved by the application of FRP in roll of tape.
- The highest load carrying capacity achieved by Kevlar sheet wrapped column confirms that the wider rolls of the fibre reinforcement can provide a greater confining stress. This also can be concluded that the fibre layout has significant influence on the behaviour of concrete structural members.
- The external confinement with galvanised steel straps improves the strength of the column to a certain extent. The brittle, sudden, soundless failure of the galvanised steel straps wrapped columns shows that the galvanised steel straps had very little effect on improving the ductility of the columns.
- The E-glass proved to be the weakest reinforcing material in this study. The ultimate load achieved by the E-glass wrapped specimen even lower than the internally reinforced column.
REFERENCES


EXPERIMENTAL STUDY ON QUANTITATIVE APPLICATION OF ELECTROMAGNETIC RADIATION EXCITED BY COAL-ROCK FRACTURE

Wenxue Chen¹, Xueqiu He¹, Baisheng Nie¹ and Hani Mitri²

ABSTRACT: A coal-rock uniaxial compression experimental investigation was conducted in laboratory. Electromagnetic radiation (EMR) and acoustic emission (AE) signals were gained during coal-rock fracture under different antenna types and arrangements. The results show that EMR excited by coal-rock fracture are broadband frequency, the EMR and AE signals are from the same source, but their generation mechanism is different. Under the same frequency band of antenna, the EMR amplitude from antenna parallel with crack plane is bigger than from antenna vertical to crack plane. Thus, EMR signals from developing crack propagates along the crack surfaces, which are principle contributions to total EMR signals and the EMR signals from antenna reflect the crack state parallel with receiving direction plane of antenna in coal-rock under uniaxial compression. A quantitative relationship between EMR frequency along the major crack plane and crack was derived by previous studies, which can be used into applications for coal-rock dynamic disaster prediction in the future.

INTRODUCTION

Electromagnetic radiation from materials fractured under stress was first observed by Stepanov in 1933 (Urusovskaja, 1969). The generation mechanism of EMR from rock fracturing and deformation is very complex, and is still unknown very clear today. A number of researchers attempt to explain the EMR mechanism. For example, acceleration and deceleration of dislocations (Perelman and Khatiashvilli, 1981), electro stress effects and electro kinetic effect (Fox6epr, et al., 1985), rupture of bonds effect (Gershenzon, et al., 1985), charge and discharge of electrical dipole (Ogawa, et al., 1985), compress model of atom (Guo, et al., 1988) and synthesis effect of electrical dipole transient, acceleration and deceleration of electric charge (He, et al., 1995). All the results of research show that electromagnetic radiation (EMF) comes from synthetic contribution of electrical dipole transient, variable motion of charge and rupture of bonds and so on. In fact, EMF is affected by internal microcosmic condition (He and Liu, 1995) and properties of materials (Misra and Ghosh, 1980; Frid, et al., 1999). The research of EMR generation mechanism, which can develop the basic rock fracture electromagnetic subject, and another important task is, be applied into fields of industry. For example, prediction of geological dynamic disasters, especially, the coal-rock dynamic disasters in coal mines.

In present, the EMR applications were focused on statistic relationship between EMR and stress in macrocosm (Frid, 1997; Liu, et al., 2001), and a few researches considered the quantitative relationship between EMR and failure degree of rock during deformation and fracture in microcosm or mesoscale. Because of EMR signals are abundant very much, which one kind of EMR signals is available and accurate to be used to predict dynamic phenomenon is key question. This paper try to find out the useful EMR signals and give application guide of the quantitative relationship between EMR signal and crack propagation during deformation and fracture of coal-rock. A coal-rock uniaxial compression experimental investigation was conducted in laboratory. A kind of EMR classified from abundant EMR is presented, which can be seen as useful EMR signal to predict coal-rock failure degree and a quantitative relationship between this kind of EMR frequency and failure degree is derived according to previous studies.

EXPERIMENTAL SYSTEM AND COAL-ROCK SAMPLES

Based on previous experimental system, a novel gas-loading system for coal-rock is designed and which is made of servo-controlled testing system, EMR and AE data acquisition system, load stress recording system, and EMR shielding system. Figure 1 shows a schematic diagram of experimental setup. A servo-controlled 1 700 kN testing system (MTS815.02) was used to load the specimens. The
signal of EMR from coal-rock deformation and fracture is weak, and can be affected by outer environment (wireless broadcast, electric machine and electricity). Thus, the forward gain is set to 40 times, a copper shielding net, the size of grid is 0.5 mm, thickness is 0.5 mm, is as shielding system, is designed and the shielding net is connected to ground. Sketch map for EMR experiment system such as Figure 1.

![Sketch map of EMR experiment system](image)

**Figure 1 - Sketch map of EMR experiment system**

In order to compare EMR signal of different directions from sample, four line type antennas and one circle antenna are selected, and the parameters of antenna are shown in Table 1.

<table>
<thead>
<tr>
<th>No.</th>
<th>Antenna type</th>
<th>Resonance frequency (kHz)</th>
<th>Attenuation coefficient (K)</th>
<th>Remark</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>EMR antenna(Line)</td>
<td>150</td>
<td>61</td>
<td></td>
</tr>
<tr>
<td>2</td>
<td>EMR antenna(Circle)</td>
<td>Broadband</td>
<td>39</td>
<td>R=60 mm</td>
</tr>
<tr>
<td>3</td>
<td>AE sensor</td>
<td>50</td>
<td></td>
<td></td>
</tr>
<tr>
<td>4</td>
<td>AE sensor</td>
<td>150</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The samples of coal-rock were gained from several Chengzhuang coal mine in China. Ten coal-rocks were examined in this study- original coal-rock sample (hard coal-rock) from coal mine. All of cylindrical test specimens were 50 mm in diameter and 100 mm in length, with ends parallel to within ±0.02 mm.

**EXPERIMENTAL DESIGN AND PROCESS**

In order to obtain effects of shielding system, one EMR antennas are fixed in outer of shielding copper net. The four EMR antennas (Frequency is 150 kHz) are fixed in the shielding system, and distance away from the coal sample is 5 cm respectively. Two AE sensors (Frequency is 50 kHz and 150 kHz respectively) are fixed close to the coal-rock sample. The circle antenna is fixed around of the sample, which distance away from sample is about 3 cm, sketch map of the position for antenna such as Figure 2.

![Sketch map of the position of the antenna](image)

**Figure 2 - Sketch map of the position of the antenna**
This test was carried out under ordinary room conditions, and loading with constant strain rate. The contents of test include:

- Study on influence of copper shielding net for EMR when loading system is working.
- Study on EMR and AE rule from deformation and fracturing of uniaxial compression on coal-rock and sandstone samples.
- Study on fracturing type and crack propagation rule of coal-rock.

**EXPERIMENTAL RESULTS AND ANALYSIS**

Ten coal-rock specimens were conducted in laboratory, which belong to middle hard coal-rock. Data acquisition channel 1, 2, 3, 4 and 5 correspond to antenna 1, 2, 3, 4 and 5 respectively. Channel 6 corresponds to circle antenna. Channel 7 and 8 correspond to AE sensor 7 and 8 respectively. Before starting to loading samples, EMR amplitude of line antenna inner and outer copper shielding net under working state of loading machine, results are shown as Figure 3 and Figure 4. According to a lot of group experimental results, Figure 5 to Figure 9 are examples of result for one same coal-rock specimen under uniaxial compression.

![Figure 3 - Changes in EMR amplitude with time (Channel 1 is located in copper shielding net)](image1)

![Figure 4 - Changes in EMR amplitude with time (Channel 5 is located in outer of copper shielding net)](image2)
Figure 5 - Changes in stress and displacement with time

Figure 6 - Changes in AE amplitude and impulse with time (Channel 7-150 kHz)

Figure 7 - Changes in EMR amplitude and impulse with time (Channel 2-150 kHz)

Figure 8 - Changes in EMR amplitude and impulse with time (Channel 3-150 kHz)

Figure 9 - Changes in EMR amplitude and impulse with time (Channel 6: circle antenna-broadband)
Experimental results were classified into several cases to be analysed based on Figures 3 to 9, such as follows:

**Influence of copper shielding net to EMR events**

Figure 3 is EMR amplitude changes with time, which comes from line antenna (channel 1) locating in the copper shielding net. Figure 4 is EMR amplitude changes with time, which comes from line antenna (channel 5) locating in outer copper shielding net. The EMR events from line antenna in copper shielding net is apparent less than from line antenna out of copper shielding net. Figure 5 shows that the peak of EMR event happened at around 20s of time, at the same time, the loading machine was launched. The launch of machine did not work for the EMR event changes. Thus, the copper shielding net can be used as shielding net to prohibit EMR events from outer environment.

**Relationship between EMR and AE with stress**

From Figures 5, 6 and 7, it can be known that the changes in EMR or AE signal are basic accordance with loading stress and displacement with time under coal-rock uniaxial compression condition, which is similar with other rock fracture. And EMR signal and AE signal are pulsed and noncontinuous mode. EMR amplitude and impulse are accordance with AE’s, but not completely accordance. Thus, the events of EMR and AE are coming from the same source and the generations of EMR and AE are different.

**EMR relationship between line antenna channel 2 and channel 3**

Figure 7 and Figure 8 are EMR results from experiment under same loading conditions, but different acquisition positions of antenna (channel 2 and channel 3). Change trends in EMR of channel 2 and channel 3 are basic accordance with stress, but the amplitudes and impulse of EMR from channel 2 are apparent higher than from channel 3. Comparing the channel 2 and 3 antenna positions relative to specimen, channel 2 position was parallel with crack plane of coal-rock (mainly failure direction) and channel 3 position was located in the front of minor failure direction of coal-rock and vertical to crack plane.

**EMR relationship between line antenna and circle antenna**

Circle antenna is a broadband frequency antenna. Figure 9 shows that there are a lot of EMR events happened with variety frequency in the process of uniaxial compression of coal-rock. The EMR signals are obvious more than line type antenna received. Such as Figure 10, the fitted curve of antenna 6 is higher antenna 2 and antenna 3 received by line type antenna. Thus, though the line type antenna only can get single frequency EMR events, it can reflect the change of stress. The circle antenna can get enough EMR events, yet cannot reflect the change of stress obviously yet.

**APPLICATION DISCUSSIONS**

Many researches, such as dislocations and charged electrons model (Misra, 1977; Ghosh, 1980), discharge model (Finkel, et al., 1975) based on crystals splitting experiment, movement of fracture tips (Gershenzon, et al., 1986) and capacitor model (Gershenzon, et al., 1986; O’Keefe, 1995), which attempt to explain the origin of EMR from fracture were, unfortunately, unable to explain all the features of the detected radiation. According to critical analysis of experimental observations, a better satisfied
model for generation of EMR was presented by V Frid and A Rabinovitch, et al. 2003, which resolve the weakness of previous models and presented EMR’s characteristics. According to these models, it can be known that the EMR amplitude increases as long as the crack continues to grow. The EMR waves from developing crack propagates along the crack surfaces, which are principle contributions to total EMR signal and it’s frequency is the same as that of the oscillating ions of the crack sides (Rabinovitch, et al., 1998). Comparing EMR amplitudes of channel 2 and channel 3, the same conclusion is gained under uniaxial compression of coal-rock, the EMR amplitude from the line antenna in parallel with crack plane is far big from the line antenna vertical to crack plane. This law is obvious fitted better before specimen failure (time at around 180s), such as Figure 10. Thus, during the time before coal-rock failure, it is available to calculate relationship between crack and EMR.

Studies show that the atomic perturbation creating the EMR is limited by the crack width ‘b’ (since at both sides of the crack, atomic movements are restricted), then Equation (1) is gained (Frid, et al., 2000; Rabinovitch, et al., 2000).

\[ b \approx \frac{\lambda}{2} = \frac{\pi v_R}{\omega} \]  

(1)

Where, \( v_R \) is Rayleigh wave velocity, \( \omega \) is frequency and \( \lambda \) is wavelength.

Indeed, since the Rayleigh wave velocity \( v_R \) in a material with a given Young’s E, poisson ratio \( \mu \) and density \( \rho \), is given by Equation (2).

\[ v_R = \frac{0.87 + 1.12 \mu}{1 + \mu} \sqrt{\frac{E}{2\rho(1+\mu)}} \]  

(2)

The frequency \( \omega \) of EMR is derived by Equations (1) and (2).

\[ \omega \approx \frac{\pi(0.87+1.12\mu)}{b(1+\mu)} \sqrt{\frac{E}{2\rho(1+\mu)}} \]  

(3)

Equation (3) shows that the frequency of EMR should be inverse proportional to the crack width. Thus, considering application of EMR, the EMR frequency received by antenna in front of coal-rock can reflect the crack size inner coal-rock. The relationship between EMR (major contributions to the total EMR) and cracks can be as one potential approach to predict the state of coal-rock deformation and fracturing in front of working face in coal mines.

**CONCLUSIONS**

(1) EMR and AE signals measured in coal-rock fracture are similar to other rock fracture in laboratory. EMR and AE are coming from the same source and the generation mechanism of EMR and AE are different.

(2) The EMR amplitude from the line antenna in parallel with crack plane is bigger than from the line antenna vertical to crack plane. This law is obvious fitted better before specimen failure (time at around 180s). Thus, the changes in EMR amplitude before coal-rock fracture with time are available information to predict dynamic fracture inner coal-rock.

(3) A quantitative relationship between EMR along the major crack plane and crack was derived by previous studies, which is an approximate equation and may be used into applications for coal-rock dynamic disaster prediction in the future.

The research described in this paper is a preliminary effort to clarify quantitative application of EMR excited by coal-rock fracture. Because of coal-rock’s property and complicated inner structure, further study of quantitative relationship between EMR and failure degree is needed in laboratory.

**ACKNOWLEDGEMENTS**

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EVALUATION OF ROCK SUPPORT PERFORMANCE THROUGH INSTRUMENTATION AND MONITORING OF BOLT AXIAL LOAD

Hani Mitri

ABSTRACT: A practical method for the estimation of roof bolt axial loads in the field as a means for evaluating rock support performance in underground mine openings such as drifts and gate roads is presented. The method is based on attaching an instrumented coupler to the bolt head prior to installation. Once installed, the coupler load cell enables the measuring and monitoring of the bolt axial force exerted on the face plate. Previously developed load cell technologies for rock anchors are reviewed, and their design deficiencies for applications in underground mining are highlighted. On the other hand, the instrumented coupler technique is shown to provide a simple and practical means for the evaluation of rock support performance in mine openings.

INTRODUCTION

The consequences of rock falls in underground mines can be disastrous. Rock falls can cause mine production delays, and can be responsible for serious injuries and even fatalities. Therefore much attention is given to the design and installation of adequate rock support systems. Rock supports such as mechanical rockbolts and resin grouted rebars are installed in almost all mine access areas such as gate roads, drifts, ramps and shafts. Because of their important role as primary rock support, it is necessary to verify that the rockbolt is functioning adequately and is not subjected to excessive load. There are many situations where such a concern may arise especially in development and production areas where the ground response changes constantly due to mining induced stress changes.

The need to measure the rockbolt load with instrumentation methods has been recognized by researchers and new measurement techniques were successfully developed and became commercially available. One of such products is the vibrating wire hollow load cell technology shown in Figure 1. As the load cell is sandwiched between the face plate and the reaction plate, it measures the axial strain inside the cell, from which the axial bolt load is calculated. The disadvantages of this technology are numerous: a) the vibrating wire strain gauge is fragile and often breaks prematurely as it reaches its limit of 2500 to 3000 µs, which is often not enough to ensure the measurement of the bolt yield load; b) the face and reaction plates must be placed perfectly perpendicular to the bolt to capture the correct reaction force, which is not always possible in mining applications; c) the hollow load cell reduces the headroom of the gate road by at least 15 cm; d) surface preparation is required to make sure that the surface and reaction plates are perfectly parallel.

Figure 1 - Design concept of the hollow load cell

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INSTRUMENTED ROCKBOLT

To overcome the drawbacks associated with the use of the hollow load cell technology, Mitri and Marwan (2001) proposed a design that is based on placing the a strain gauge directly in the bolt head by drilling a central blind hole that extends beyond the threaded portion as shown in Figure 2. As the strain gauge is placed in the axial direction, it measures the load induced axial strains, which can be converted to bolt axial load. Unlike the hollow load cell, the embedded strain gauge in the bolt head requires no additional headroom in the drift, and does not require any surface preparation.

Figure 2 - Rockbolt load cell design concept proposed by Mitri and Marwan (2001)

Thus, this technique is not prone to erroneous measurements due to the position of the face plate with respect to the rock surface. Moreover, the metal-based strain gauge has a stretch capacity that is much greater than the vibrating wire strain gauge used with the hollow load cell. Thus, with this design, it is possible to measure the bolt yield load as the strain limit of the metal gauge is larger than the yield strain of the bolt steel material.

It can be seen that this technique is applicable to virtually any type of rock anchor such as mechanical rock bolt, cone bolt, grouted rebar, and forged head bolt. Mitri and Laroche (2005) report the application of this technique in Canadian hard rock mines. Recognizing the need to protect the head connector during transport of the bolt, a protective cap was designed as shown in Figure 3. Mitri and Marwan (2001) and Mitri and Laroche (2005) give more details of this concept.

Figure 3 - Photo of protective metal caps placed on instrumented bolts (Mitri and Laroche, 2004)

In spite of the above mentioned features, the instrumented bolt design has some drawbacks that limit its suitability to mining applications. One deficiency of this design is the need to transport the bolt back and forth to the mine. This results in additional materials handling work and cost. It also exposes the head connector to damage during shipping and handling. Another deficiency is that the hole drilled in the bolt
head reduces its capacity by about 3%. The gauge being installed in the bolt itself is thus unable to measure the ultimate or breaking strength of the rock bolt. Most rock anchors have an ultimate strength that is 10% - 15% greater than the yield load. Thus, it can be said that even if the bolt load reaches the yield limit, it can still offer further load supporting capability before it eventually breaks. It is worth noting that most reported bolt failures occur at the end of the threaded section where the cross sectional area is the least. Snapping of the threaded bolt head in this fashion is due to excessive load at the bolt head and is not uncommon in underground mines near active mine production areas.

INSTRUMENTED COUPLER LOAD CELL

While the design concept of the instrumented rockbolt offers unique advantages over the traditional hollow load cell technology, it is nevertheless not practical for mining applications because of the design deficiencies mentioned above. To overcome such deficiencies, the author proposed a new concept for monitoring axial load on the bolt head was proposed (Mitri, 2011). The new concept, illustrated in Figure 4, is made of a coupler instrumented with a metal strain gauge that is placed in the blind borehole along the axis of the coupler. The coupler load cell is fitted onto the rock anchor, which, once installed in the rock, permits the monitoring of the anchor head axial load. The advantages of this new design concept are numerous. The coupler is designed so that its yield load is greater than the ultimate breaking strength of the rock anchor. This design ensures a complete load path monitoring of rock support performance until failure. The coupler design offers the advantages of light weight in shipping and handling as well as ease of installation in the field. As shown in Figure 4, the upper end of the coupler load cell is a threaded hole that is tapped to fit the desired thread diameter of the rock anchor.

![Figure 4 - New coupler load cell design concept](image)

DATA ACQUISITION

The instrumented coupler load cell circuit design is based on the well-known Wheatstone Bridge. Therefore, a wide range of data acquisition systems can be used to record the bridge output for a given load. The coupler load cell is first calibrated under axial loads ranging up to 50% of the elastic limit of the target rock anchor for it is designed. The calibration chart provides a linear relationship between the axial load (in the desired units) and the Wheatstone Bridge output in mill volts is shown Figure 5. This type of calibration allows the instrumented coupler load cell to be connected to an existing data acquisition system in the mines.
Possible options for load monitoring include a) portable strain gauge reader, b) data logger, c) internet-connected data logger, and d) wireless transmission of data. When a portable strain gauge reader is used, the calibration chart is provided in units of load versus microstrain. The use of portable units is simple and practical in areas that do not require continuous load monitoring. The advantage of data loggers is that they provide load data regularly at a uniform time interval specified by the operator, e.g. one hour. Also, the data logger can be made multi-channel, and typically can monitor 4 bolts at a given location. Data stored in the data logger can be downloaded periodically on a laptop computer underground. Alternatively, the data logger may be connected to an Ethernet cable, if available at the mine, to transmit the load data to a computer on surface. Another option is when the data logger is equipped with a WiFi modem/antenna, whereby the recorded data is transmitted remotely. In all of the aforementioned methods, an instrumentation wire is used to connect the coupler load cell to either the portable readout unit or the data logger. For more information, the reader is referred to the http://www.yieldpoint.com of YieldPoint Inc.

Figure 5 - Typical calibration test result of a coupler load cell

APPLICATIONS

The new coupler load cell has been applied to a variety of rock anchors in different mining applications. Figure 6 shows a typical load cell used to monitor No. 6 (3/4 inch) rebar. The outer nut serves as a jamming nut to enable the bolt spinning to puncture the resin cartridges and allow for mixing. Once the fast resin sets, the outer nut is removed and the bolt is tightened.

Figure 6 - Typical load cell mounted on a rebar

CONCLUSIONS

This is a new method of monitoring axial loads in rock anchors like rock bolts and rebars used as rock supports in underground mine drifts and roadways is presented. It has been shown that conventional hollow cell load technology is not suitable for mining applications. The author developed an alternative method that instruments the bolt head directly. However, such method deemed not practical due to the need to ship and handle the instrumented bolt from and to the mine. The new method, based on a
coupler load cell concept, is shown to be simple, practical and efficient. The instrumented coupler load cell technique is universal in the sense that it can be adapted to any type of rock anchor.

REFERENCES


IMPROVEMENT OF ROCK BOLT PROFILES USING ANALYTICAL AND NUMERICAL METHODS

Chen Cao, Jan Nemcik and Naj Aziz

ABSTRACT: The anchorage capacity of fully grouted bolts has been studied for many years, however the bolt rib profile and its effect on bolt shear resistance is poorly understood. A new development in calculating load transfer capacity between two rib profiles of varying geometries is discussed. The derived mathematical equations presented calculate the stress distribution adjacent to the fully grouted bolt and bolt pull out force needed to fail the resin. The Fast Lagrangian Analysis Continua (FLAC) program was used to verify the calculations of stress within the resin. The novel idea of coupling the bolt geometry with the calculated stress provides another powerful tool to investigate the bolt profile configuration and its effects on the load transfer mechanism for the benefit of the mining industry.

INTRODUCTION

Steel bolts are an essential part of roadway support in coal mining roadways. The effectiveness of bolt reinforcement is a well known and well researched subject; however, little has been done in optimising the bolt profile that directly contributes to the load transfer between the bolt and the surrounding resin. To improve bolt load transfer through the steel rebar design, it is essential to research the details of the bolt profile shape and configuration. Analytical studies, laboratory tests and numerical modelling provide the tools that enable a better understanding of the rebar profile role in increasing the shear resistance during the working life of bolts.

Investigations of load transfer between the bolt and resin indicates that the bolt profile shape and spacing plays an important role in improving the shear strength between the bolt and the surrounding strata. The short encapsulation pull out tests of rock bolt indicate significant variance of shear resistance for various bolt profile spacing, angle, shape and size. Empirical studies can match the graphs of physical tests however these methods cannot describe the exact reasoning why such behaviour occurs. Numerical modelling techniques are much better as they can mimic the physical tests in great detail; however, these methods depend on an accurate knowledge of the physical properties that must be incorporated or added into the model. For example, in the micro-scale world resin properties may not be the same as those indicated in the laboratory tests. The power of the numerical model rests on its ability to compare several models and to establish the optimum solution to the problem. The laboratory testing has its challenges as fabrication of minute differences in bolt profile in the workshop is difficult. Nevertheless the laboratory tests are important to calibrate all the empirical work and the numerical models. At present a mathematical description of the bolt profile and its behaviour during the bolt pull out test is under development to provide better understanding of the physical process that influence the shear strength of the loaded bolt.

The in situ pullout tests are commonly used to examine the shear capacity of rock bolts. Only a few researchers (Blumel, et al., 1997 and Aziz, et al., 2008) have conducted laboratory tests to study various bolt profile parameters and their influence on the bolt anchorage. A typical steel bolt profile configuration is shown in Figure 1. Aziz et al. (2008) found out that the shear load capacity and stiffness vary greatly for various bolt profile configurations and showed that the bolts with profile spacing (PS) of 12.5 mm exhibit very low stiffness while spacing of 37.5 mm had higher peak and residual strengths. The results are presented in Figure 2. Other experimental tests by Aziz and Web (2003) indicated that smooth bolts exhibit very low shear resistance.

Clearly, the mathematical description of the physical problem provides an understanding of the bolt profile behaviour and together with the numerical modelling would constitute a realistic approach to improving the bolt rib profile configuration which is the subject of this study.
Figure 1 - Steel bolt rib profile configuration

Figure 2 - Laboratory studies of steel bolt pull out tests versus the maximum load for various bolt rib profile spacing (after Aziz et al., 2008)

STRESS DISTRIBUTION IN INFINITE ELASTIC MEDIA

Derived mathematical equations enable calculations of the stress tensor at any point within the resin encapsulating the loaded steel bolt. Such detail can make assessment of the bolt profile and its influence on the shear strength possible. Boussinesq (Poulos and Davis, 1974) derived fundamental solutions for various loads on infinite or semi-infinite elastic media.

While loading an infinite strip on the surface of a semi-infinite mass as shown in Figure 3, the stress tensor anywhere within the media can be calculated as a function of the load, position and material properties. For a uniform normal load as shown in Figure 3, the stress tensor can be calculated using the Boussinesq equations while for the uniform shear load, the stress distribution can also be calculated via Cerutti’s equations (Poulos and Davis, 1974).

Figure 3 - Calculated stress tensor at any position given by x and z coordinates within the semi-infinite elastic medium loaded by a uniformly distributed load (p)
MODELLING OF STRESS DISTRIBUTION ADJACENT TO THE BOLT PROFILE

To draw a link between the load transfer system and the bolt profile configuration, a single spacing between two bolt ribs is examined (Figure 4).

Figure 4 - A single spacing between two bolt profiles showing geometry

When the bolt is loaded, the load is applied to the resin boundary as shown in Figure 5. The location of these loads is dependent on the bolt geometry while their magnitudes depend on the bolt geometry and the resin-bolt interface properties. During the bolt pull out tests, the loads can be shown as shear forces and normal forces as shown in Figure 5.

Figure 5 - Load transfer between the bolt and the resin

An assumption was made that the resin-bolt interface forces $S_1$, $S_3$, $S_4$ and $S_5$ are small and when the angle $\theta$ approaches 90˚ the shear force $S_2$ becomes small thus, all shear forces at the interface are equal to zero.

The stress at the bolt interface can therefore be assumed to be as shown in Figure 6a where the normal stress remains acting along the side of the bolt profile. This stress component plays a major role in stress distribution within the resin.

A FLAC model was used to validate this simplification. The model simulated a section of the steel bolt shown in Figure 6b with the angle $\theta = 45˚$. The pull out force was applied to the bolt and the modelled shear stress contours were plotted as shown in Figure 6b. The modelled contours were compared with the shear stress calculated using the Boussinesq and Cerutti’s equations. Both the calculated and modelled contours were in reasonable agreement indicating that the numerical assumptions can be used. Properties used in the calculations and in the FLAC model are shown in Table 1.

<table>
<thead>
<tr>
<th>Material properties used in calculations and FLAC model</th>
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<tbody>
<tr>
<td>Property</td>
</tr>
<tr>
<td>UCS (MPa)</td>
</tr>
<tr>
<td>Shear strength (MPa)</td>
</tr>
<tr>
<td>E (GPa)</td>
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<tr>
<td>Poisson ratio</td>
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NORMAL AND SHEAR STRESS ON A FAILURE PLANE

To investigate where the resin failure will occur, several potential planes of failure can be trialled. As an example a plane of failure that spans between the two rib tops is considered. The Mohr-Coulomb criterion of failure was used to calculate the maximum pull out force needed for the assumed plane of failure.

The equations to calculate the bolt pull out force are derived after linking the bolt geometry shown in Figure 4 and the sum of integrated normal and shear stresses along the failure plane. The lengthy mathematics of the stress tensor solutions and sums of forces along the plane of failure are not presented here, and are outside the scope of this paper. The Matlab program designed for numerical solutions of complex problems was also introduced to obtain specific solutions where derivation of mathematical equations was too complex.

For static equilibrium, the sum of forces parallel to the bolt axis is zero:

$$\sum F_x = 0$$

If the shear forces are assumed to be very small than from Figure 5a:

$$F = pb \sin \theta$$  and  $$p = \frac{F}{b \sin \theta}$$

Where,  

- $F$ = axial bolt pull out force;
- $p$ = Normal load on bolt boundary at the profile inclination $b$.

STRESS DISTRIBUTION IN THE RESIN

The FLAC modelling indicates that the evenly distributed compressive load ($p$) has the major influence on the stress in the resin while all interface shear and tensile stresses are small and can be ignored.

To apply Boussinesq’s stress transformation equations in calculating the normal and shear stress along the studied plane of failure, the coordinate system must be rotated to match the geometry shown in Figure 3. The rotation is shown in Figure 7 where the distance PQ represents the failure plane and the angle $\theta$ is the rib slope. Point (A) is any point on the plane of failure while variable ($h$) is the distance from point P.

Under normal conditions the resin elastic properties are similar to the surrounding strata and the resin boundary can be extended to infinity.
The stress tensor transformation calculations using the Boussinesq equations transform stress to shear and normal stress along the failure plane PQ (Figure 7). To be able to calculate changes in normal and shear stress parallel to the failure plane when the bolt geometry changes, the bolt profile dimensions must be coupled with the stress within the equations. The angles $\alpha$ and $\delta$ shown in Figure 7 need to be substituted with bolt profile configuration parameters $a$, $b$, $c$ and $\theta$ shown in Figure 4. To simplify derivation of the mathematical equations, this step is done after stress transformation.

The normal and shear stress to the failure plane are determined after stress transformation, lengthy integrations, rearrangement and substitutions of each term. The final solutions are as follows:

$$\int \sigma_n \, dh = \frac{F}{b \pi \sin \theta} \left[ \left( \frac{\pi}{2} - \theta \right) \left( c + 3bcos\theta \right) - \left( c + 2bcos\theta \right) \tan^{-1} \left( \frac{c \cos \theta + b \cos 2\theta}{\sin \theta + b \sin 2 \theta} \right) \right. $$

$$+ \left. b \cos \theta \tan^{-1} \left( \frac{c + b \cos \theta}{b \sin \theta} \right) \right]$$

And the sum of shear forces to the failure plane is:

$$\int \tau \, dh = \frac{p}{\pi} \int \sin^2 \alpha \, dh = \frac{F}{\pi} \left[ \tan^{-1} \left( \frac{c + b \cos \theta}{b \sin \theta} \right) + \frac{\pi}{2} - \theta \right]$$

**MOHR-COULOMB FAILURE STUDY ALONG THE PLANE OF WEAKNESS**

Two combined stress fields are considered within the resin. The first one is the initial pre-loading stress tensor at the failure surface and the second one is the load induced stress tensor. Due to bolt installation procedure, the initial pre-loading stress, within the resin, is considered to be small as most of the resin along the bolt cures after pre-tensioning the bolt.

Thus, the failure criterion ($f$) is expressed as net resistant force that can be summed up as:

$$f = (c_w + \mu \sigma_{n0} - \tau_0)L + \left( \int \mu \sigma_n \, dh - \int \tau \, dh \right)$$

Where:

$L = c + 2bc \cos \theta = \text{failure length}$; $h = \text{distance from any chosen point along the plane of weakness from 0 to L}$; $c_w = \text{cohesion}$; $\mu = \tan \phi$, where $\phi$ is an internal angle of friction of the resin; $\sigma_{n0} = \text{initial normal stress}$; $\tau_0 = \text{initial shear stress}$; $\tau = \text{shear stress introduced by pull out force}$; and $b$, $c$ and $\theta$ are bolt profile parameters.

The forces along the plane of weakness are shown in Figure 8, and the failure criterion expression can be written as:

$$f = c_w L + L \mu \sigma_{n0} - L \tau_0 + \mu \int \sigma_n \, dh - \int \tau \, dh$$

Substituting integrated solutions into the equation:

$$f = F_0 + \frac{F_1}{\pi \sin \theta} \left[ \left( \frac{\pi}{2} - \theta \right) \left( c + 3bcos\theta - \frac{b \sin \theta}{\mu} \right) - \left( c + 2bcos\theta \right) \tan^{-1} \left( \frac{c \cos \theta + b \cos 2\theta}{\sin \theta + b \sin 2 \theta} \right) \right.$$

$$+ \left. b \left( \cos \theta - \frac{\sin \theta}{\mu} \right) \tan^{-1} \left( \frac{c + b \cos \theta}{b \sin \theta} \right) \right]$$

\[Figure 7 - Rotated axis of the loading diagram with the assumed plane of failure\]
APPLICATION EXAMPLE

In this example, let failure length \( L = 10 \text{ mm} \) and \( c = 8.5 \text{ mm} \) as constants. Rib height, \( b \sin \theta \), varies from 0 to 4 mm shown in Figure 9 as coloured line (All dimensions in Figure 9 are in mm).

For initial conditions, let:

\[ \sigma_{n0} = 0 \text{ MPa} \quad \text{and} \quad \tau_0 = 0 \text{ MPa} \]

For material properties, let resin cohesion \( c = 16 \text{ MPa} \) and frictional angle \( \phi = 35^\circ \). Then

\[ F_0 = L(c_w + \mu \sigma_{n0} - \tau_0) = 10(16 + \tan 35^\circ \times 0 - 0) = 160 \text{ N} \]

Using failure criteria formula:

\[
f = F_0 + \frac{F \mu}{nb \sin \theta} \left[ \frac{\pi}{2} - \theta \right] \left( c + 3bc\cos \theta - \frac{bsin\theta}{\mu} \right) - (c + 2bc\cos \theta) \tan^{-1} \left( \frac{c \cos \theta + b \cos 2\theta}{bsin\theta + cn \sin 2\theta} \right) \\
+ b \left( \cos \theta - \frac{bsin\theta}{\mu} \right) \tan^{-1} \left( \frac{c + b \cos \theta}{bsin\theta} \right)
\]

\[ = 160 + F \cdot G(\theta) \]

Where \( F \) is bolt pull-out force.

The \( G(\theta) \) may be called the "influence factor" of the rib height. If the influence factor is positive, the failure will never occur on the assumed plane of weakness. If the influence factor is negative, the failure may occur on the weakness line within the resin. In addition, while the influence factor is negative, the larger the absolute value of the influence factor, the easier the failure propagates.

The plot of \( G(\theta) \) versus rib height (\( b \sin \theta \)) is shown in Figure 10.
From the graph in Figure 10 it can be concluded that:

- The minimum rib height is about 0.8 mm to initiate a failure on the proposed plane of weakness. If the rib height is less than 0.8 mm, then \(G(\theta)\geq 0\), and
  \[ f = 160 + F \cdot G(\theta) \geq 0 . \]
  It means that failure will never happen on the proposed weakness line.
- The minimum influence factor \(G(\theta)\approx -0.17\) when the rib height is about 2mm. Therefore the minimum bolt pull out force to cause the resin failure is
  \[ F = 160/0.17 = 941 \text{ N} \]
- For any other rib height the probability of failure along the proposed weakness plane decreases.

CONCLUSIONS

The study of the bolt profile shape presented shows how the mathematical equations were derived. These equations are used to calculate the pull out force needed to fail the resin for different bolt profile configuration. The calculations can be applied to any plane of probable failure within the resin. The important outcome of this study is to show that there is another way to examine resin failure around the bolt for different profile configurations that can be compared with the laboratory tests and numerical modelling. This method can provide better understanding of the bolt-resin interaction with rock reinforcement.

The derived mathematical equations consist of in-built bolt geometry parameters and these can be changed to optimise the bolt shear strength capacity. Various types and bolt geometries and profile configurations can be trialled using the described approach. This method provides another step towards designing more efficient bolt profiles to optimise the support capacity in the mining industry. Future efforts will be made to extend the calculations to three dimensions while the final outcome of this work is to write a computational program to evaluate the pull out force for various bolt profile geometries.

REFERENCES


BEARING CAPACITY OF A GLASS FIBRE REINFORCED POLYMER LINER

Jan Nemcik, Ian Porter, Ernest Baafi and Joshua Towns

ABSTRACT: The development and testing of stiff polymer to be used as part of roof and rib support system for underground mining is discussed. Laboratory tests were conducted to examine the bearing capacity of thin spray polymer liners (TSL) Polymer, which included both puncture test and the optimum size of steel bearing plates. It was found that a 60 mm diameter steel disc carried approximately 60 t, and that a 250 mm diameter steel plate had no major effect on the integrity of the TSL. Further polymer bearing capacity studies on uneven surfaces must be conducted to simulate actual mine site conditions.

INTRODUCTION

Since 1990 thin spray on polymer liners (TSL) have emerged as a viable mine roadway skin protection. Many of the products have specific applications such as weathering protection of the mine roadways, while stiff liners such as shotcrete are used as part of the ground support. Now a fast setting stiff polymeric liner ‘Tough Skin’ is under development to be used as part of the roof support system. These polymeric materials have the potential to allow a highly automated face support cycle that promises substantial improvement in roadway development rates and the removal of personnel from the immediate face area, minimizing associated health and safety risks.

To replace steel mesh, Tough Skin must exhibit several desirable properties such as: high strength and stiffness; good bond to rock or coal substrata under wet or dry conditions; adequate toughness with high elongation and yielding characteristics when loaded to failure and high resistance to puncture loads as Tough Skin must be able to withstand high bolt plate loads.

Tough Skin has the role of reinforcing the coal mine roadway skin and complimenting the bolts that provide reinforcement to rock strata (Lukey, 2008). Roof bearing plates used in conjunction with high capacity roof bolts are utilised in underground mining operations as a means of primary and secondary support. The bearing plates have the secondary objective of securing steel mesh support which provides a passive system guarding against rock falls and minor roof collapse. The load bearing steel plates often yield when severe roof or rib conditions occur. If used as skin reinforcement, the TSL must be strong enough to resist any bearing plate loads without puncture. To minimise the possibility of puncturing or tearing of the polymer due to high bolt forces, various plates with suitable profiles need to be designed with the objective of minimising the compressive load at the plate boundary. Bearing plates are often square with sharp edges, a feature unsuitable for application with polymeric TSL. New bearing plate designs are needed to ensure gentle load distribution at the plate/TSL boundary to minimize any stress concentrations. They need to be of sufficient dimensions and suitable shape to be able to spread the concentrated forces of the roof bolt and rib bolt across a broader area of the mine roof or coal rib. To avoid stress concentrations on the corners of the plate especially where an uneven roof is present a circular plate and a large contact area is suggested to minimize stress at the plate boundary.

PUNCTURE TEST

A puncture test was designed to quantify the bearing capacity of a 5 mm thick polymer sheet reinforced with glass fibre. To measure the load bearing capacity of the polymeric material, a series of steel discs shown in Figure 1 were used to load the TSL material to failure. The 5 mm thick glass fibre reinforced polymer liner used for testing is shown in Figure 2. A 500 kN Instron servo-hydraulic universal testing machine and a 5000 kN Avery compression machine were used to conduct the tests. The test involved nine steel discs of varying diameters, compressed into a polymer sheet sample as shown in Figure 3.
Figure 1 - A series of steel disc sizes used to load the polymer TSL to failure

Figure 2 - Preparation of the polymer based TSL for testing

Figure 3 - Loading of a steel disk compressed into a glass fibre reinforced polymer sheet

As load increased, flow of polymer material caused the plate and TSL to bend upward. As the load approached the maximum bearing capacity of the thin polymer material, separation of polymer layers and shearing occurred below the loaded area. The maximum bearing capacity was identified as puncture of the bearing plate through the polymer sheet with permanent damage. Increasing displacement under a stationary load indicated yielding of the polymer material. As the compression limit of the polymer was reached the next phase of loading indicated a rapidly increasing load with minimal change in displacement. This is attributed to the beginning of the compression of the steel plates and where the tests were terminated.

The test results summarised in Figure 4 below indicate that for disk diameter sizes from 10 mm to 50 mm an approximate linear increase in bearing capacity occurred, increasing 100 kN for every 10 mm increase in the disk diameter. As the outer perimeter of each disk is directly proportional to the disc diameter it appears that the linear relationship between the failure and the size of the disk is a result of the shear failure occurring mainly along the edge of the loaded disk. For larger disc diameters of 80 mm
to 120 mm the bearing performance increased to a higher rate of approximately 400 kN per 10 mm increase in diameter. The smaller load bearing areas exhibited a complete deterioration of the polymer structure while loading of the larger diameter discs indicated that an elastic polymer core was left within the centre of the loaded area enabling a higher bearing capacity of the polymeric sheet. The disc bending may have been part of the cause of the reduced stress along the disc perimeter as the spherical seat that was used to load the 20 mm thick steel disks was only 50 mm in diameter. Once each test was concluded and unloaded, the majority of the smaller discs remained embedded within the polymer sheet. After removal of the disc the compressed polymer region was visibly brittle and broke away easily leaving a hole within the polymer sheet. This effect was minimised in larger discs indicating that the shearing stress at the perimeter was reduced possibly due to both the disk bending and exponentially increasing elastic core below the disk centre.

It must be noted that these idealized tests were performed on perfectly straight and smooth surface areas that rarely exist underground. Nevertheless these tests serve as a good estimation of the minimum load bearing areas needed for the bolt plate sizes. The polymer warping issue under high compressive loads as demonstrated in Figure 3 gives another bearing capacity limit that should not be exceeded, as it may compromise the adhesive integrity adjacent to the loaded bolt plates.

![Figure 4 - Summary of the load bearing capacity tests](image)

Extrapolation of data presented in Figure 5 may indicate the ultimate loads for larger bearing areas however; such estimations may not be accurate.

![Figure 5 - Extrapolated ultimate load capacity for larger disk diameters](image)

ROLE OF BEARING PLATES

Bearing plates are a fundamental and integral part of any rock support solution. Currently used steel plates need to be re-designed so they can eventually be adopted by the TSL system and used successfully on polymer surfaces without any risk of compromising the polymer’s integrity. Early bearing plates were flat square plates of steel with a central hole to accommodate a roof bolt is shown in
Figure 6. Progressive development of high capacity rock and cable bolts has seen the transformation of bearing plates to circular, domed or even triangular designs with varying nut and bolt interfaces.

![Progressive development of high capacity rock and cable bolts.](image)

**Figure 6 - Square steel plates commonly used in coal mining industry**

As traditional plates have been designed specifically for installation with steel mesh systems, they exhibit features not suited to polymer reinforcing skin. When identifying a set of performance criteria for a suitable plate supporting the polymer TSL the following features were addressed. The plate should:

- be of sufficient stiffness to prevent excessive bending;
- be suited to providing compressive resistance to a roof surface coated with polymer TSL;
- not include any sharp contact points that would cause undue stress on the polymer;
- allow for a reasonable degree of deformation to accommodate uneven roof surfaces;
- incorporate drainage outlets to allow passage of water from roof strata to the mine roadway.

After preliminary consideration of the bolt plate, the primary design requirements for a suitable prototype include an appropriate diameter to bear the load of the corresponding roof bolt and a central circular hole that is suitable for the steel bolt design. In addition, the ultimate load capacity of bearing plates should be determined to correspond with the type of bolts, which, they are to be used with. It should be noted that even for fully encapsulated bolts, it should be assumed that high collar loads are still possible due to roof deterioration and load transfer onto the bearing plate. Finally, bearing plates should be designed so that if failure occurs it is progressive and not catastrophic.

From the above reasons it was concluded that an initial design of plate be fabricated to the specifications shown below:

- Circular shape to eliminate undue stress concentrations at plate corners
- Thickness of plate 3 mm to allow for some deformation
- Total diameter 250 mm to ensure sufficient bearing capacity
- Rolled up plate edge to allow for angular installations on uneven surfaces
- Six solid symmetrical ribs to increase central stiffness.
- Drainage holes around collar of plate

To compare whether the circular plate is better than the square design two plates were manufactured, one square and another of circular design, with identical solid wedge type ribbing as shown in Figure 7.

![Square and circular steel bolt plates with identical solid reinforcing ribs](image)
The designed roof plates were used to load the polymer TSL sheet. The primary objective of the testing was to devise a reproducible laboratory procedure to investigate the polymer bearing capacity subject to steel plate loading of various designs. The tests were conducted using an Instron servo-hydraulic universal testing machine equipped with a 500 kN load cell. The arrangement of the test was such that the polymer TSL was placed on a flat steel loading surface with the roof plate compressed on top of it as shown in Figure 8. Each plate was tested under compression on a polymer sheet to allow observations to be made regarding failure characteristics and ultimate load capacity.

![Figure 8](image1.png)

**Figure 8 - Loading of a polymer TSL sheet using the designed steel plates**

When the bearing capacity of both plates were approximately 300 kN the plate deformation caused compression damage to the polymer in the collar zone as shown in Figure 9. Despite that the loading continued to 500 kN the polymer sheet did not suffer any significant damage further away from the plate centre.

![Figure 9](image2.png)

**Figure 9 - Polymer puncture zone due to plate centre deformation**

As can be seen in Figure 10, almost identical loading characteristics were observed for both square and circular plates.

![Figure 10](image3.png)

**Figure 10 - Compressive loading characteristics of a square and circular steel plate on polymer sheet**
CONCLUSIONS

The purpose of this study was to investigate the polymer bearing capacity and the characteristics of polymer failure when loaded with steel plates. A test was devised that utilised loading of circular plate of varying diameters to determine the load bearing capacity of a 5 mm thick glass fibre reinforced polymer sheet. As expected the tests indicated that the load bearing capacity increased with the size of the loading disc. An approximately linear relationship between disc diameter and the load bearing capacity of the polymer indicates that the shear stress along the disc perimeter dominates the failure. Discs of small diameter puncture the polymer sheet while larger discs leave a semi-elastic region of polymer below the centre of the disk.

The fabricated 250 mm square and circular steel plates performed to their design capacity with minimum damage to the polymer TSL sheet. Both the square and circular plate loading characteristics were similar and the difference between them was inconclusive. Testing of the bearing plates determined that the dimensions of the plate ensured a bearing capacity that was sufficient to prevent unacceptable polymer damage due to tensioned bolt forces or excessive loads due to strata failure.

It must be pointed out that all tests were conducted on a smooth surface and do not represent the usual roof or rib conditions in the mine and therefore the results represent the minimum plate sizes that can be used. Further polymer bearing capacity studies on uneven surfaces must be conducted to simulate actual mine site conditions.

REFERENCES

DETERMINING THE ULTIMATE STRENGTH OF ‘TOUGH SKIN’, A GLASS FIBRE REINFORCED POLYMER LINER

Jan Nemcik, Ian Porter, Ernest Baafi and Jeffrey Navin

ABSTRACT: Fully encapsulated roof bolts have been proven to confine the roof and rib strata actively, while steel mesh only provides passive support and protection from falling rocks as they dislodge from the roof. To provide better stability of the mine roadway skin, replacement of the steel mesh with a spray on polymer liner is being investigated. Among the many benefits that may be realised by the application of quick setting spray on polymer liners as skin support are an increase in development rates and improved safety for working personnel. Extensive testing of a range of available polymer formulations is currently being undertaken; this particular study is to determine how Thin Spray on Liners (TSL) behave in conditions encountered in underground coal mines and demonstrates their superior properties over steel mesh as an alternative form of roof skin support.

INTRODUCTION

The role of Tough Skin, a candidate TSL being developed at the University of Wollongong, is to act as a composite material that provides reinforcement to the skin of a mine roadway. To act as a composite with the roadway skin the polymer must have strong adhesion to the strata, but strong adhesion is only beneficial if the polymer is also strong and tough. Strength testing on potential polymer liners is done predominantly on small samples where failure mode or material properties can be examined comparatively in order to evaluate and reformulate the material until an optimum formulation can be determined. Large scale tests are required to predict the potential behaviour of the TSL underground, while an in situ trial is the final confirmation of the product performance. Various tests were performed on a routine basis, including the non destructive loading of a polymer sheet where a load of terracotta pavers was placed onto the sheet to verify its strength. This work extends these tests, loading large sheets of reinforced polymer to failure.

LOADING THE POLYMER SHEETS TO FAILURE

Previous tests involved applying evenly distributed loads of 1 t terracotta pavers to a 1 m by 0.8 m Tough Skin polymer sheet and a similar size section of steel mesh as shown in Figure 1. The measured deflection of the Tough Skin sheet was approximately 40% lower than the deflection of the steel mesh subject to the same load (Nemcik, et al., 2009).

![Figure 1 - Glass fibre reinforced polymer Tough Skin and steel mesh loaded to 1 t.](image)

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154
10 – 11 February 2011
To measure the ultimate load capacity of the polymeric material a series of reinforced Tough Skin sheets were loaded to failure. The one tonne load used in the previous tests was inadequate as a much larger load is needed to fail the sheets. An experimental steel frame was built to hold a sufficiently large reinforced polymer sheet that was then placed in a 500 t Avery compressive testing machine. Due to clearance in the Avery and the size of the steel frame used to clamp the sample, the actual tested area was limited to 800 mm x 600 mm. To place an evenly distributed load onto the Tough Skin polymer sheet a semi inflated air bag was installed on top of the sheet. The airbag was inflated without protruding out of the steel enclosure, covered with a steel plate and loaded as shown in Figure 2.

![Figure 2 - Polymer sheet loaded with the assistance of an air bag](image)

The load applied to the Tough Skin polymer sheet and its deflection was monitored using a 5 000 kN load cell and a LVDT. Strains on the bottom surface of the sheet were monitored with electrical resistance strain gauges glued directly to the polymer surface as shown in Figure 3.

![Figure 3 - Strain gauged Tough Skin polymer sheet loaded to failure](image)

Testing the first sample using an airbag was only able to achieve results up to a certain point, failure of the Tough Skin did not occur as the air bag volume reduced substantially due to the compressive load and the hydraulic press cylinder of the machine reached the end of the stroke. During testing the machine was loaded at 2 mm/min until the end of the stroke at 100 mm was reached. The maximum force applied to the 5 mm thick Tough Skin sheet was 68 kN, while a deflection of 35.3 mm was measured. Unloading the sheet saw a recovery of 23.9 mm with a permanent displacement of 11.4 mm. The large amount of displacement recovery after unloading shows that the sheet was mostly within the elastic region of deformation during testing. The load versus deflection of the Tough Skin polymer sheet and recorded strains are shown in Figure 4.

As the Tough Skin polymer sheet did not fail it was not clear how much force the polymer could withstand at failure due to an evenly distributed load, but a load of nearly 7 t with no indication of failure is substantial and well above that expected in normal mining conditions. The results indicate that the use of the airbag to achieve an evenly distributed load over the entire surface was successful, however, if the bag was filled with a non-compressible fluid such as water the displacements would have been lower.
and the failure load reached. The use of water looked promising and would have solved the problem of using air but unfortunately the air bags available for the experiment were unsuitable for use with water.

Figure 4 - Test 1 a) Load versus deflection b) Strain versus deflection

In the second test a smaller area of the Tough Skin was loaded to failure. A 150 mm diameter steel spherical seat was used to load the centre of the polymer sheet as shown in Figure 5. To lessen the effect of strain concentrations, where the spherical seat would come in contact with the polymer sheet, a dense rubber mat was placed between the two surfaces. The sample was again loaded at a rate of 2 mm/min. As expected the area of maximum strain and hence the location of failure was directly beneath the point of loading. Polymer yielding was characterised by non-brittle failure propagation and no loss of integrity where the polymer sheet had not yielded. Failure of the Tough Skin was reached at a load of 45 kN and a deflection of 52 mm.

Figure 5 - Loading of the Tough Skin polymer sheet to failure with a 150 mm diameter steel plate

The failure load in test 2 was lower than the load applied via the air bag in test 1. This was to be expected as the load during test 1 was evenly distributed over the whole sheet while in test 2 the strains at the point of loading were more concentrated resulting in tear of the polymer.

In the third test the Tough Skin was bonded to a number of terracotta pavers and the pressurised air bag was trialled again. This time the bag was pressurised to 220 kPa in order to minimise deflection during loading. In this test not enough pressure was generated to yield the polymer as it was predicted that airbag damage due to the excessive air pressure would occur, resulting in unsafe conditions. The load developed in test 3 was 100 kN (10 t) with 38 mm of the deflection, as can be seen in Figure 6. A comparison of load versus displacement results for tests 1 and 3 are presented. It is obvious from this figure that bonding of the pavers, test 3, to the polymer had significant influence on the load distribution.

Test 4 was conducted with the same loading conditions as in test 2, but using a layer of pavers as a buffer on top of the Tough Skin as shown in Figure 7. The three layers of loaded pavers were used to distribute the load away from the point of loading, minimising any early polymer tear due to stress concentrations. The pavers were initially bonded to the polymer sheet but unfortunately adhesion was lost between the sheet and half of the pavers during loading. Tests 2 and 4 show similar trends of load versus deflection.
DISCUSSION

The tests presented here indicate that a 5 mm thick reinforced Tough Skin sheet is strong, tough and resilient to compressive or shear failure and tear. As expected, load distribution plays a major role in the apparent strength of the Tough Skin, point loads cause the polymer to tear resulting in failure at lower loads when compared with uniformly distributed loads. The role of Tough Skin is to become a composite member of the strata at the skin level where high adhesion of the polymer to the substrata plays a major role in early reinforcement of fractured roadway skin. A stiff, strong and tough polymer formulation can complement the bolt system not only in the early stages of roadway development, but also in later stages of the roadway’s life where it could also provide sufficient support to broken strata encountered in tailgates and other heavily loaded areas experienced in most coal mines. The strength of a 5 mm thick layer of Tough Skin appears to be comparable to the 5mm heavy duty steel mesh currently used to support heavy strata conditions.

Testing of heavy duty steel mesh at the University of Wollongong indicates a similar strength to unbonded Tough Skin, while Tough Skin that bonds to the substrata promises a better roadway skin support mechanism than steel mesh can provide. When bonded to the substrata, Tough Skin will provide a stiff and resilient coat to the roadway surface, reinforcing the fractured surfaces and strengthening the loose rocks. In most coal mines no significant dynamic loads exist and gradual yielding of strata is usually experienced. Ideally, when yielded, Tough Skin should transit to a post-failure mode of highly elastic but strong material with audible warning that indicates to the mine personnel that excess strata movement is present. This can be possible with an appropriate reinforcing fibre formulation that will remain strong and flexible after the polymer matrix has yielded. It is envisaged that a tear in the Tough Skin liner can be repaired using portable air driven spray on equipment that can patch the Tough Skin to the desired thickness.
CONCLUSIONS

The purpose of this study was to quantify the probable loads that a 5 mm thick polymeric liner can carry while spanning between the bolts. This work has shown that steel mesh is not stronger than the tested Tough Skin. All four tests indicate that the ultimate capacity of the de-bonded Tough Skin spanning small distances is of the order of 4-10 t. Most of the tested steel mesh types do not exceed these values. If the bolted horizon remains relatively intact, the dead loads imposed on the Tough Skin would not exceed its strength. When heavy strata conditions occur with bolts losing their integrity then it is probable that in places the Tough Skin strength could be compromised, this however is no different to the situation where steel mesh has to cope with similar loads.

The tested polymer size was smaller than the average span between the bolts in most coal mine roadways as the loading equipment did not allow full size samples to be tested. The authors believe that larger samples would not outperform or underperform the tested results. As part of further work on determination of Tough Skin mechanical properties, full scale application of Tough Skin is recommended. Full scale trials would enable detailed monitoring of the Tough Skin behaviour and a direct comparison to the test results described here.

REFERENCES

IMPROVED TECHNIQUES FOR HEADING DRVAGE

Stan F Johnson

ABSTRACT: An obvious need exists for more efficient and safer techniques for driving development headings for longwalls and also, in many cases, for bord and pillar operations. A very expeditious solution to this problem is proposed, with the potential to drive such headings two or more times faster than can presently be achieved for any specific set of mining conditions.

INTRODUCTION

The importance of maintaining development ahead of extraction has been emphasised by Mitchell (2010) who pointed out that most mines have little or no lead-time on development. Speed of development has been mentioned by a number of writers, Kathage (2010) has discussed the use of extendable conveyors and Golsby (2010) the introduction of continuous haulage at Clarence Colliery. Place-change mining has been discussed by Howarth and Bevan (1991) and Kathage (2010) and this system has been described as having the effect of reducing delays caused by the need to stop production to erect supports. It is considered that improved rates of development can be achieved by simplifying current procedures.

PROPOSED PROJECT

The following description applies to a typical pair of headings. The belt, which will later carry the longwall production, is in the return airway. The intake road is used for transport of men and materials.

Basic heading machine

Captive at the face would be a “heading machine”, which may be a “conventional” continuous miner, but preferably would be a unit of much simpler and more rugged design. Such a header would incorporate a standard continuous miner ripper head, mounted in a different, more stable way, being incorporated in a beam immediately behind the rotating picks. The beam would be mounted at each end on a heavy duty, but relatively narrow frame, several metres long on each side of the heading. The rear ends of these side members would be joined in a wide, low cross member, incorporating the hydraulic tank, pump, motor and electrical and radio control equipment for the header.

The side frames would be equipped with suitable hydraulic cylinders to raise and lower the ripper head. At ground level at the front of the side frames, they would be connected to a short shovel ramp, incorporating at the centre, the tail end of a chain conveyor, flat on the floor extending a few metres beyond the rear of the header. As is common in many continuous miners these days, the cut coal is swept towards the centre chain conveyor by large, opposing helixes, inherent in the construction of the ripper head.

It will be noted that the header would be constructed of only six large components, able to be transported easily to its commencement site and connected together with only one or two large pins at each carefully designed and constructed junction. Electric cable, hydraulic and water hoses would be incorporated in each component, with well designed junction boxes inherent at each connecting joint.

It will be seen, therefore, that there would be a large empty space immediately behind the ripper head. In this space would be located what may be regarded as a “roof support installation factory”, equipped with all required equipment for convenient, expeditious and simultaneous installation of mesh, roof bolts, rib bolts, cable bolts and mega bolts, without any compromise of hydraulic capacity, whilst providing total safety and convenient amenity for the operators. A bolting machine would be provided for every bolt. All supplies for a pillar length of drivage would be always readily at hand for the operators. The “roof support installation factory” would be connected to the surrounding frame of the header only by two short lengths of chain or wire rope, with which the “factory” could pull itself forward at the conclusion of each module.

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In some mines, propulsion of the header could be by using a system similar to a tunnel boring machine, with hydraulically actuated pads impinging on the walls, but the use of the principle of the “sliding floor” seems generally far more appropriate, as it is completely unaffected by soft or spalling ribs, or indeed even soft, wet ground conditions, being dependent only on its mass, evenly spread from rib to rib and for 20 or 30 m along the headings.

**Sliding floor principle**

This technology was very successfully employed in many large scale hard rock tunnel projects around the world in the late 1950s and early 1960s, until it was superseded by tunnel boring machines. In such projects, the sliding floor carried a very stable by-pass close to the face, on which could stand two muck trains, side by side, each comprising a 25 t locomotive and five 10 m³ muck cars. For about 50 m inbye of the by-pass, the single track was surrounded by a heavy steel floor. After blasting, broken rock covered most of this 50 m, providing ideal conditions for very fast clean up by a high capacity mucker, typically a Conway mucker, which could continue to clean up to two metres beyond the front of the floor, when the floor would be moved forward, with the mucker continuing to operate.

The floor, typically about 150 m long, was divided into three components of roughly equal mass. Hydraulic cylinders, of about one metre stroke, connected the first and second and second and third components. To move forward, the front rams were extended about a metre, the second and third sections providing more than adequate reaction to enable the front section to move forward about a metre. Then, by retracting these rams and simultaneously extending the second set of rams, the centre section moved forward, with the first and third sections providing reaction. Then the third section was pulled forward. The rails of the heavy permanent track extended into the body of the third section, beneath the rails on the floor. Short, tapered rails overlapped the permanent track, enabling rolling stock to travel effortlessly on to or off the floor. As the floor progressed, it was only necessary to slip sleepers under the rails and dog them.

In applying the principle of the sliding floor to the drivage of a pair of headings in a coal seam, it is proposed to use a small sliding floor, about 20 or 30 m long in each heading, constructed of shallow steel tubs, about 2 m wide by 4.6 m long, filled with concrete and pin jointed together.

**Intake road equipment and operation**

Two options are available:

- Using a continuous miner as the heading machine, the disadvantage, however, being that roof support installation would be much slower and less expeditious. In this option, the same machine is also used to drive the cut-through.
- Using a heading machine, as described earlier, and a completely separate machine, called “the cut-through cutter”, solely to drive the cut through.

Where a heading machine is used, it would be connected to the inbye end of the sliding floor by two hydraulic rams, enabling it to sump forward, rip down and retract, cutting the cusp. In this case, the sliding floor would be about 0.3 m thick. A standard shuttle car would ride on it, doing nothing until required to drive the cut-through. Figure 1 shows the intake road equipment with the vent tube on the right-hand side. The cutter for the cut through, the associated shuttle car and the short elevating conveyor are carried on the sliding floor until they are in position to drive the cut through.

![Figure 1 - Intake road heading equipment with the vent tube on the right-hand side](image-url)
Where a continuous miner is employed as the header, when the advancing heading reaches the cut-through location, the continuous miner would turn the corner and drive the cut-through in the conventional way, with the shuttle car dumping on to the chain conveyor of the sliding floor.

Where Option 2 is employed, the specialized “cut-through cutter” would also ride on the sliding floor until required, when it would drive the cut-through, with the shuttle car operating behind it. For both options, it may be noted that the average length of the shuttle car haul is only about 15 m. The layout while driving the cut through is shown in Figure 2. The elevating conveyor that was shown in Figure 1 has been attached to the cutter and loads into the shuttle car.

Figure 2 - Heading machine for a cut-through drivage

At the outbye end of this sliding floor would be attached the tail end of a belt conveyor, of much lighter construction than the belt road conveyor. (The belt road conveyor would be required to haul longwall production of 40 or 50 tpm, whereas the maximum from a continuous miner ripper head is about 20 tpm). As well as the relatively light tail end, this sliding floor would also haul forward a pillar length of conveyor and structure, including the drive head. The structure would be appropriately modified to enable it to slide forward easily.

Included with this belt and structure would be a tripper, similar to the tripper on a skyline stockpiling installation, but in this case, in operation the tripper would be stationary, locked between the floor and the roof while the conveyor is pulled slowly and steadily through it. Incorporated into the construction of the tripper would be the tail end assembly and about two metres of a chain conveyor 700 mm wide. The drive head and about 1.5 m of chain conveyor would be permanently mounted on the belt road sliding floor, delivering directly into the conveyor boot end.

The central section of the chain conveyor (about 40 m long) would be designed to be always handled in one piece, including the top and bottom chains. It would be designed to be dragged out of a cut-through, around a corner protected by a few suitable rollers, outbye along the intake road, then inbye to the newly driven cut-through, around its corner and into it to near its face, to await the arrival of the belt road header and sliding floor.

To achieve this, the conveyor pans would be only about 700 mm long, vertically pin-jointed on only the outbye side, with their floors overlapping so as not to impede the operation of the chain.

Belt road equipment and operation

As indicated earlier, the “header” could be either a conventional continuous miner or a heading machine as previously described.

The sliding floor would be of similar length but of greater thickness, possibly about 700 mm.

This is to ensure that an extremely strong, stable anchorage would be provided for the belt. A chain conveyor 700 mm wide would extend along the centerline of the floor, in a “canyon”, which may be covered, discharging into the conveyor tail end.
The conveyor tail end would be equipped with hydraulic powered adjustments to trim it marginally for line and level. Immediately behind this and also attached to the sliding floor would be a purpose built frame to enable the permanent conveyor structure to be safely and conveniently assembled while the belt is operating.

The heading machine in the belt road, shown in Figure 3, has space for capsules of chain wire mesh carried close to the ripper head. The vent tubes in this heading are located on the left hand side.

![Figure 3 - Belt road heading machine and ancillaries](image)

It would be intended that drivage would occur in only one heading at a time. This would enable all materials for the next pillar length of drivage to be conveniently stocked on the sliding floor or on a trailer immediately behind it. An overhead mono-rail, mounted on the sliding floor would facilitate this materials handling work.

**Operating sequence**

The intake road would be driven as far as the cut-through. The cut-through would be driven beyond the centreline of the belt road.

The intake road would be further advanced, bringing the future position of the tripper and chain conveyor tail end in line with its future position in the cut-through.

The drive head section of the chain conveyor would be disconnected.

Production would now commence in the belt road, provided that 200 m had been inserted in the loop take up.

The tail end section of the chain conveyor would be disconnected.

The central section of the chain conveyor would be withdrawn and relocated to the new cut-through.

Routine maintenance, stonedusting and re-stocking of supplies would now be done in the intake road.

The tripper would be moved forward to its new position. This would be done by clamping it to the top belt, then reversing the belt at low speed.

The exhaust fan and 100 m of telescoped ventube is moved forward to its new position.

When production in the belt road stops, production would now re-commence in the intake road.

**Ventilation**

In both headings ventilation would be by means of a pillar length of telescopic tubing, extending on to the sliding floor and close to the faces.

In both cases, tubing of slightly larger than normal diameter, mounted on a sleigh type sub frame about 0.3 m high, would extend from the rear of the sliding floor back to close to the fan, but not attached to it. A second ventube, of normal diameter, attached to the fan would extend inside the larger tube to the
rear of the sliding floor. As the header and the sliding floor progressed forward, the smaller tube would be left behind on the floor, maintaining ventilation right to the face. Appropriately designed small rollers inside the larger tube would facilitate the telescopic action.

At the conclusion of a pillar length of advance by the header and sliding floor, the smaller tube and the fan would be pulled forward, by a wire rope inside the larger tube, with a winch mounted on the sliding floor below the upward bend in the ventube.

Ventilation of the cut-through would be by conventionally installed ventube from a T-piece on the intake road’s tube.

**Transport of supplies**

Advancing a pair of headings more rapidly would require a corresponding increase in the quantity of supplies required to be delivered from the surface to the working area near the face. Additionally, the last 100 m to the face would be more restricted than for a typical shuttle car operation.

It is proposed to use “gondola” type trailers, with the load carrying gondola about 5 m long by about 1.6 m wide, slung quite low between a pair of wheels at each end, mounted on heavy duty kingpins. When travelling, the forward pair of wheels would be steered by the drawbar, the steering of the rear pair being locked.

Such trailers could be hauled by a pivot steer tractor of about 10 -12 t, preferably also only about 1.6 m wide. Hence, it would be convenient for the tractor to deliver one or more trailers of supplies to the vicinity of the second last open cut-through, leave them on one side of the heading and drive out on the other side, picking up any empty trailers on the way.

For the area close to the face, it is proposed to use two small electric cable powered general purpose pivot steer or skid steer tractors. Such tractors would conveniently move the supplies trailers up close to the sliding floor, where their loads would be moved on overhead monorails, integrally mounted on the sliding floors, to their ultimate destinations.

These small tractors would also be very beneficial in handling the pulling equipment for relocating the central part of the cut-through conveyor from one cut-through to the next.

**Relocation of the cut-through conveyor**

This job of relocating the cut through conveyor would be done when production work in the intake road had been completed and production in the belt road had commenced.

A sledge, fitted with a small hydraulic system powering a “hand over hand” puller would be connected to the outbye end of the centre section of the chain conveyor. The inbye end of a 150 m length of wire rope, say 50 mm in diameter, would be connected to the puller. The outbye end of the rope would be clamped to a sledge-mounted hydraulic powered anchorage between roof and floor.

A group of 3 or 4 appropriate rollers, anchored between roof and floor at the corner of the cut-through would be installed, enabling the conveyor to be pulled from the cut-through, around the corner and 40 m outbye along the intake road.

The corner roller assembly would then be moved forward and installed in a similar position at the corner of the new cut-through. The anchorage sledge would also be moved forward, to the face of the new cut-through and the end of the rope attached to it. The puller would then be hooked to the inbye end of the conveyor and operated until it was close to the anchorage at the face, to await the arrival of the belt road heading machine and sliding floor.

**CONCLUSIONS**

The potential benefits of the system are:

- Generally, the time to install one complete roof support module could be expected to be only slightly more than the time to install one bolt.
- Operation of the ripper head would be independent of roof support work.
- Supplies for face work would be always readily at hand.
- In the drivage of both headings, no labour at all would be required for ventilation. (However, in drivage of the cut-through, the standard method would be required.)
- In the drivage of both headings, coal haulage would be continuous from the picks to the belt.
- The overall capital cost would be low.
- The cost per tonne would be much lower for any specific set of conditions.

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A MAJOR STEP FORWARD IN CONTINUOUS MINER AUTOMATION

David C Reid, Jonathon C Ralston, Mark T Dunn and Chad O Hargrave

ABSTRACT: Progress on a major research and development project undertaken by the CSIRO Mining Technology Group to advance the automation capability of continuous mining equipment in underground coal mining operations is described. The aim is to increase the overall rate of roadway development as well as providing a safer working environment for underground mine personnel.

The outcomes achieved at the half-way mark of this ACARP funded three-year research and development project are reported. Details of the technical developments undertaken towards demonstration of a “self-steering” capability to enable a Continuous Miner to automatically maintain a given mining heading and mining horizon under production conditions are provided. Reported outcomes include the means to accurately determine both the location and orientation of a Continuous Miner in real-time using a combination of a navigation-grade inertial navigation unit, Doppler radar and optical flow technologies. Comprehensive performance evaluations have been conducted using a scaled skid-steer mobility platform and results achieved to the present stage of the project indicate that the required automated self-steering functionality is achievable under production conditions. The project outcomes represent an important move towards achieving a step change improvement in underground roadway development practice.

INTRODUCTION

Continuous Miner (CM) automation has been identified by the Australian coal industry as essential to achieve a step change improvement in roadway development productivity. The research project reported in this paper covers one component of a larger research and development effort referred to collectively as CM2010.

Advances in longwall coal mine production in Australia have put pressure on roadway development rates, which have become a limiting factor in the coal production supply chain. Due to new technology and equipment, production rates from longwalls are increasing rapidly while roadway development improvements have generally been limited and incremental in nature. The Roadway Development Task Group (RDTG), established in 2005 by the Australian Coal Association Research Program (ACARP), is tasked with addressing this production bottleneck by means of research and development projects that will lead to new processes and technologies. The RDTG carried out a review of existing processes and technologies and charted a path forward for roadway development based on the introduction of new systems that could deliver the necessary improvements in production for both new generation longwalls and for existing mines. Based on this review, an extended research and development programme was initiated with broad industry support. This was formalised as the CM2010 roadway development strategy in 2008 with four major technology categories: remotely supervised continuous miner, automated installation of roof and rib support, continuous haulage and integrated panel services.

Current CSIRO research and development is focussed on the first of these technologies, a remotely supervised Continuous Miner. The primary goal is to deliver a “self-steering” capability that will enable a Continuous Miner to maintain 3D position, azimuth, horizon and grade control within a variable seam horizon under remote monitoring and supervision.

This research builds on previous research which demonstrated the practical application of advanced inertial navigation techniques for longwall automation (Reid, et al., 2001; Reid, et al., 2006). Despite the inherent time-dependent position drift associated with all inertial-based solutions (Savage, 2000), the longwall automation research delivered a commercial-grade system that achieved sustained position accuracy under full production conditions. The use of this enabling-technology for underground mining...
applications is covered by international patent and is targeted as an area of strategic research by the CSIRO Mining Technology Research Group (Hainsworth and Reid, 2000).

Progress to date in this CM2010 CM automation project, the technology solutions that have been developed, the experimental setup to evaluate the performance and the results achieved is described.

INERTIAL NAVIGATION TECHNOLOGY FOR MINING GUIDANCE

Central to achieving a practical inertial navigation solution for CM automation is the performance of the underlying inertial sensor technology – in particular the gyroscopes and accelerometers. The quality and technical sophistication of commercially available inertial technologies covers a wide range from Micro-Electro-Mechanical Systems (MEMS)-based devices used in mass consumer products such as mobile phones, automotive application such as air-bag control and satellite navigation aiding, through to fibre-optic and laser-based devices which are essential for high performance air, land and sea navigation systems.

Common to all inertial navigation systems, irrespective of the gyroscope and accelerometer technologies used, is the fundamental requirement to compute a positional translation by means of the numerical double integration of acceleration (as measured by the accelerometer sensors) and angular rotation by the single integration of angular rate (measured by the gyroscope sensors).

In recent decades a large body of strapdown navigation theory, that builds a theoretical framework for optimally combining the inertial sensor data to compute 3D position and thereby a navigation solution has been developed. Even with the highest performance sensing devices, the nature of numerical integration means that position errors will accumulate and grow with time. In a free-inertial mode, where only purely inertial information is used, this position error will grow quickly even for a high performance system (Savage, 2000).

Given this inherent limitation to inertial sensor performance, practical inertial navigation solutions operate in an aided-inertial mode to limit the growth of these errors by taking advantage of external (non-inertial) information. The most convenient and commonly used strategy is to periodically correct the integration error build-up by taking advantage of times when the inertial system is stationary (i.e., in a non-moving position relative to the earth) to correct and recalibrate the internal velocity calculations. This simple and quite robust aiding strategy known as Zero Velocity Updating (ZUPTing) can be very effective but requires relatively frequent stops (typically every few minutes) for a short duration (typically about 10 s). With ZUPTing it is possible to reduce the position errors for a typical high performance system from nautical miles per hour to metres per hour.

Further improvements can be made by incorporating external aiding, for example, the addition of velocity sensing to internally allow the inertial navigation system to continually correct for sensor noise and integration error build-up by comparing internally computed velocity to the external source. This arrangement is shown in the block diagram of Figure 1. Conceptually, this approach can be thought to extend the ZUPTing strategy to non-zero velocity updating and is generally referred to as Vehicle Motion Sensor (VMS) aiding. VMS-aiding is a key requirement necessary to achieve a practical navigation solution for automated CM guidance.

VMS-aiding is commonly used with vehicle-mounted inertial navigation systems by utilising odometry signals from rotary encoders fitted to the vehicle wheels or drive train. This approach works well when the vehicle is travelling on a hard surface where wheel slip is minimal. On rough terrain wheel slip will quickly degrade the sensor performance to the point that it may be worse than without any VMS-aiding. Early in this project it was concluded that it would be necessary to develop accurate, reliable and practical non-contact odometry technology for CM automation, that is, a means of measuring vehicle motion relative to the surrounding environment without mechanical linkage from the vehicle or contact with the surface over which the vehicle is travelling.

A number of non-contact odometry technologies were considered, taking into account performance, robustness and general suitability to operate and survive in the hostile mining environment. Candidate technologies including scanning laser, optical flow and Doppler radar were identified as providing individual and complementary advantages.
Figure 1 - Block diagram showing the relationship between the IMU sensors and the aiding source used to compute the navigation output

NAVIGATION SOLUTION PERFORMANCE: EXPERIMENTAL EVALUATION METHODOLOGY

The underlying performance of navigation-grade inertial navigation systems can be confidently determined from the technical specifications of the internal gyroscopes and accelerometers. Navigation system performance is often expressed in terms of nautical miles per hour position drift for pure-inertial operation and pointing accuracy which measures the ability of the system to resolve the gravitational vector and the rotation of the earth about the central axis.

The achievable navigation performance is much harder to analyse or predict when the motion of the mobile platform (CM in our case) is unconstrained and the motion of interest is small relative to the erratic motion resulting from significant background vibration and jolting. In this case the achievable performance depends greatly on the performance of the VMS-aiding sensors and the tuning of the internal signal processing filter parameters to match the vehicle motion and dynamics. For these reasons the performance of the complete navigation system needs to be assessed under realistic operating conditions.

Routine prototype testing on underground coal mining equipment is impractical due to the logistics and statutory regulations governing the installation of electrical equipment in explosive atmospheres. For this reason a skid-steer remote-control vehicle, referred to as the Phoenix mobility platform, was adapted to provide a suitably realistic scaled mobile test platform. The Phoenix as shown in Figure 2 captured some of the CM dynamics in terms of motion profile, skid steer manoeuvring, wheel slip and jolting/vibration characteristics. In this figure one of the INS units under test can be seen mounted on top at the rear and the Doppler radar mounted on the front far corner angled down towards the ground. The optical flow sensor is mounted to the rear of the vehicle and is not visible in this view.

The Phoenix is also fitted with a high-accuracy RTK GPS using a CSIRO-located base station, which provides an absolute ground-truth position reference updated at twenty times per second with an absolute position accuracy of better than 2 cm RMS. These high accuracy absolute position data are used as a base line reference for all the navigation experiments on the Phoenix. In addition to the navigation system under test, the Phoenix is fitted with an embedded computer so that the vehicle can navigate to a mission plan under closed-loop control.

The design of the Phoenix means that the navigation experiments can be conducted on natural (i.e., un-paved) and rough terrain which provides more realistic conditions for evaluating the navigation and speed sensors.

Trials of the integrated CM navigation system are conducted along a 55 m natural bush track located on the grounds of the CSIRO research facility. The track has a loose-gravel surface in places and moderate uphill and cross-track grades. A satellite image of the test track is shown in Figure 3 with the start and end points shown and the straight line target path for the experiments indicated by the white line. The RTK GPS system is used to survey the absolute 2D coordinates of the start and end points used during these trials.
At the start of each trial the reference point on the Phoenix is placed on the start marker and the inertial navigation system is initialised with the absolute coordinates of the start location. The Phoenix is then instructed to autonomously navigate to the end coordinates using only vehicle position and attitude information provided by the inertial navigation system. The vehicle travels at a slow walking pace and in accordance with a pre-programmed mission plan periodically stops for a short duration and reverses for a short distance before moving ahead again. This motion profile approximates the motion of the CM production cycle and allows the inertial measurement unit to take advantage of Zero Velocity Updates.

Around the mid-point of the mission the vehicle is taken out of closed-loop control and is driven under manual (remote) control through a sharp left turn and a short distance off track. It is then driven back on track where closed-loop control is resumed. This manoeuvre simulates some aspects of a niche or cut-through construction which is another less frequent component of the CM production cycle. The vehicle automatically stops when it reaches the mission end point.

Multiple navigation trials as described above have been conducted over an extended period using various combinations of inertial and speed-sensing hardware.

**EXPERIMENTAL RESULTS**

Representative results from a recent navigation system performance trial follow. The start and end points for this experiment being approximately 55 m apart are shown superimposed on the satellite image of the track in Figure 3. As previously described, during each experiment navigation data from the navigation system under test is recorded and also used in real time to steer the Phoenix in a straight line between the nominated start and finish points including the periodic stops and reversing manoeuvres. For ground-truth validation, the actual path of the Phoenix is also recorded from the on-board RTK GPS system and this is shown as the blue trace in Figure 3 where the short departure from the straight path can be seen around the midpoint of the mission. Apart from this intentional departure it can be seen that the Phoenix tracked closely along the desired path.

More detailed analysis is presented in Figure 4 where it can be seen that the position error of the Phoenix throughout this mission is generally less than 0.1 m. Larger errors during the programmed reversing manoeuvre are due to deficiencies in the closed-loop control algorithms and not the navigation system. This is confirmed by observing that the vehicle regains correct track alignment once it begins to travel in the forward direction again. Relatively larger position errors near the end point reflect control system error as the vehicle experiences increased track cross-grade and vehicle slide-slip in this portion of the track.
DISCUSSION

Extensive practical evaluation of inertial system performance using the Phoenix mobility platform has yielded important information towards delivery of the final system. Firstly, the results obtained indicate that an inertial based approach indeed represents a valid means for deducing real-time vehicle position. This sensing capability is an essential component for any robust closed loop automated control system, and a very practical requirement in the development of a guidance solution for underground mining equipment. Secondly, the results obtained clearly demonstrate that the performance of navigation-grade inertial measurement systems can be greatly improved by taking advantage of position or motion information that is available within or outside the inertial frame of reference.

Figure 4 - Analysis of 2D position error as measured by the on-board RTK GPS equipment. The position of the Phoenix under INS-only guidance is in close agreement with the RTK GPS supplied position. Note that the excursion in the centre of the plot was an intentional manually instructed departure from the nominal desired path to demonstrate correct control system return behaviour.

Further developments of the guidance system have yielded a novel approach to inertial navigation aiding using vehicle speed sensing, which has greatly improved the overall performance of the inertial navigation system. This has involved the use of non-contact speed sensors which are integrated with the inertial navigation algorithms to provide accurate ZUPTing and VMS-aiding to achieve significant performance improvement.

Extensive field campaigns have provided a measure of understanding and confidence with regards to achievable position measurement with the current system. This has been undertaken using the Phoenix vehicle which has provided a convenient experimental test platform to evaluate the complete navigation system performance under CM-like motion and operating conditions. The test vehicle is fitted with on-board RTK GPS to provide sub-centimetre ground truth reference data. Importantly, the experimental results to date have shown that sub-decimetre 2D position accuracy can be achieved with a suitably-aided inertial navigation system over a 55 m track length.

Research is continuing on the further development of the non-contact speed sensing technology to ensure accurate and robust operation over the wide range of operating conditions encountered in roadway development. Further intensive evaluation is required to assess the assumption that there are periods when the CM is sufficiently stationary to permit ZUPT aiding.

While it can be expected during roadway development that there will be periods during the mining/bolting cycle when the CM is not tramming, the platform is generally not strictly stationary. Background vibrations, platform jolting and slippage due to reaction forces all constitute low level motion to the inertial measurement unit. Under a strict ZUPTing interpretation this low-level motion will either...
One way to deal with the practical realities is to adjust the internal signal processing filters so that the INS is less sensitive to this background “noise”. The trade-off here is that the navigation system will be equally less sensitive to legitimate small motion and this will lead to some degradation in navigation performance. Further testing and evaluation will be necessary to better understand and quantify the possible system degradation due to this non-ideal ZUPTing condition.

SUMMARY

With the support of the Australian coal industry, CSIRO is currently involved in a major continuous miner automation research and development project. Effort so far has demonstrated a guidance capability based on high performance inertial navigation technologies coupled with novel aiding strategies. Practical performance limits for core instrumentation have been identified. Extensive evaluations have been conducted on a scale mobility platform to identify and validate real-time system behaviour. Work continues to improve the underlying accuracy and performance towards an integrated solution.

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REMOTE TELE-ASSISTANCE SYSTEMS FOR MAINTENANCE OPERATORS IN MINES

Leila Alem¹, Franco Tecchia² and Weidong Huang¹

ABSTRACT: Complex technologies such as fully automated and semi-automated equipment and teleoperated machines are being introduced to improve productivity in mines. Consequently, the maintenance and operation of these complex machines is becoming an issue. There is a growing interest in industry in the use and development of technologies to support the collaboration between a local worker and a remote helper to deliver expertise to guide the local worker in undertaking maintenance and other activities. The productivity of the future mine relies on the effective delivery, of remote guidance. ReMoTe (Remote Mobile Tele-assistance), a mobile augmented reality system for remote guiding, has been developed at CSIRO as part of the work in the Transforming the Future Mine Theme.

INTRODUCTION

In the industrial and mineral extraction fields, complex technologies such as fully automated and semi-automated equipment or teleoperated machines are being introduced to improve productivity. Consequently, the maintenance and operation of these complex machines is becoming an issue. Operators/technicians rely on assistance from expert in order to keep their machines functioning. Personnel with such expertise, however, are not always physically located in close proximity to the equipment/machine. They are often in a major metropolitan city while the technicians maintaining equipments are in rural areas, where industrial plants or mine sites may be located. There is a growing interest in industry in the use and development of technologies to support the collaboration between a local worker and a remote helper. For example, in telemedicine, a specialist doctor may guide remotely a non-specialist doctor or a nurse (Palmer, et al., 2007); in remote maintenance, an expert may be guiding remotely a technician through the task of repairing a piece of equipment (Kraut, et al., 2003). Communication means that have been used for this purpose include telephone, email and basic video conferencing. It is generally accepted that augmented reality technology is very useful in maintenance and repair applications (Lapkin, et al., 2009).

ReMoTe is a remote guiding system developed for the mining industry. ReMote was designed to support the mobility aspect of maintenance workers. In ReMoTe, the expert, when guiding remotely a worker, uses his/her hands not only to point to remote location - “grab this” - but also to demonstrate how to perform a specific manual procedure. The potential of applying a non-mediated hand gesture communication, a proven effective technique of communication, in the field of wearable augmented reality is explored. A review of the literature on augmented reality (AR) remote guidance systems used in industry is followed by some initial results of ReMoTe testing and a short description of future work.

AUGMENTED REALITY REMOTE GUIDING SYSTEMS FOR MAINTENANCE

Automated AR based remote guiding systems

Augmented reality (AR) systems have been developed since 1990 to assist maintenance workers in various industries in conducting their tasks. In order to minimize the risk of errors, relevant information was projected onto the machine in real time (using AR) to assist operators in repairing the machine. One key benefit of the use of AR is that the attention of the operator is on the maintenance task not on the system delivering the help. Many studies were conducted in the early 2000 to evaluate the benefits of AR in the area of maintenance. Identifying the exact location of the required intervention helps reduce the transition between tasks (Henderson and Feiner, 2009). AR based guiding is better than paper based instruction for guiding an assembly task (Wiedenmaier et al., 2003), leading to a reduction in the number of errors. When comparing paper based instruction and AR based guiding, the AR based guiding system allow users to stay on task; there is no need to switch attention to a piece of paper to look for specific information and hence a reduced cognitive load (Henderson and Feiner, 2009).

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One of the early AR guiding systems being developed for the maintenance of laser printing machines is the KARMA system (Feiner Macintyre and Seligmann, 1993). The system used an optical see through display. Boeing in 1992 developed its own AR guiding system to help their technicians in the electric cabling of Boeing planes (Caudell and Mizell, 1992). This system was based on real time annotation of videos based on operator tasks. The ARTESA project (ARVIKA, 1999) at Siemens started in 1999 and aimed at further exploring the use of AR in industrial applications. As in the Boeing project, ARTESA relied on instrumentation of the workspace of the operator in order to localize him/her. Augmented information in the form of text (Figure 1) and 3D images based on the specific context of the operator’s task were generated (Weidenhausen et al., 2003).

![Figure 1 - Augmentation in the form of text in ARTESA (ARVIKA, 1999)](image1)

Subsequent efforts at Siemens (2004 to 2006) have been focusing on developing marker-less tracking as well as looking at ergonomic considerations. BMW has also explored the use of AR for guiding its maintenance workers (Platonov et al., 2006) using a see through system (Figure 2). The system uses a database of images of a system to detect specific features, which are then registered onto a CAD model. The guiding system detects features from the video of maintenance worker and compares them with the preregistered features in order to determine the orientation of the worker.

![Figure 2 - BMW AR system (after Platonov et al., 2006)](image2)

In project ARMAR (Augmented Reality for Maintenance and Repair), Henderson and Feiner (2003, 2010) have been interested in exploring the extent to which AR can increase the productivity, the precision and safety of maintenance personnel. The AR system uses a binocular video see through system (see Figure 3).

![Figure 3 - ARMAR system (after Henderson and Feiner, 2010)](image3)
The last two systems are the more developed AR systems to date for guiding a maintenance worker in performing a standard procedure. These systems cannot guide the worker in situations where there is no predefined way of solving the problem. In such a situation, there is a need to involve a remote expert.

**Tele-supervised AR remote guiding systems**

Kuzuoka *et al.* (2004) developed a system for supporting remote collaboration using mobile robots as communication media. The instructor controls the robot remotely and the operator receives instructions via the robot. In this system, the robot is mounted by a three-camera unit for the environment of the operator. It also has a laser pointer for hitting the intended position and a pointing stick for indicating the direction of the laser pointer. The movement of the robot is controlled by the instructor using a joystick.

Sakata and Kurata (Sakata, *et al.*, 2003; Kurata, *et al.*, 2004) developed the Wearable Active Camera/Laser (WACL) system that involves the worker wearing a steerable camera/laser head. WACL allows the remote instructor not only to independently look into the worker's task space, but also to point to real objects in the task space with the laser spot. As shown in Figure 4 the laser pointer is attached to the active camera-head and it can point a laser spot. Therefore, the instructor can observe the environment around the worker, independently of the worker's motion, and can clearly and naturally instruct the worker in tasks.

![Figure 4 - The WACL (left, after Kurata, *et al.*, 2004) and the REAL system (right, after REAL)](image)

Previous work in the area of remote guiding of mobile workers has mostly focused on supporting pointing to remote objects and/or remote area using a projection based approach, such as the laser pointing system in WACL (Sakata, *et al.*, 2003; Kurata, *et al.*, 2004) or using a see through based approach, such as in REAL; (see Figure 4). While pointing (with a laser or a mouse) is an important aspect of guiding, research has indicated that projecting the hands of the helper supports a much richer set of non-verbal communication and, hence, is more effective for remote guiding (Li, *et al.*, 2007; Kirk, *et al.*, 2006; Fussell, *et al.*, 2004).

The next section presents ReMoTe a remote guiding system developed for the mining industry. In Remote, the expert, when guiding remotely a worker, uses his/her hands not only to point to remote location - “grab this” - but also to demonstrate how to perform a specific procedure “you grab this way and push it this far from the wall”.

**THE REMOTE SYSTEM**

The ReMoTe, system has been developed to address the above needs. In particular, ReMoTe captures the hand gestures of the helper and projects them onto a near-eye display worn by the worker. It is composed of 1) a helper user interface used to guide the worker remotely using a touch screen device and an audio link, and 2) a mobile worker system composed of a wearable computer, a camera mounted on a helmet and a near eye display (Figure 5).

**Helper interface**

A participatory approach for the design of the helper interface was adopted. The aim was to design a system that would fulfil the users’ needs and be as intuitive to use as possible. The initial step consisted of observing maintenance workers and developing a set of requirements for the helper-user interface (UI) based on our understanding of their needs including:
• The need for supporting complex hand movements such as: “take this and put it here”, “grab this object with this hand”, and “do this specific rocking movement with a spanner in the other hand”.

• Mobility of the worker during the task, as they move from being in front of the machine to a tool area where they access tools, to the back of the machine to check components, such as valves.

• The need to point/gesture in an area outside the field of view of the worker, hence the need to provide the helper with a panoramic view of the remote workspace.

![Figure 5 - Worker interface](image)

Subsequently, a first sketch of the interface was produced consisting of a panoramic view of the workspace and a video of the worker’s view. The video provides a shared visual space between the helper and the worker that is used by the helper for pointing and gesturing with their hands (using unmediated gesture). This shared visual space augmented by the helper’s gestures is displayed in real time on the near eye display of the worker (image + gestures). The helper UI consists of:

• A shared visual space which displays, by default, the video stream captured by the remote worker’s camera. This space occupies the central area of the touch table.

• A panoramic view of the worker’s workspace, which the helper can use for maintaining an overall awareness of the workspace. This view can also be used by the helper for bringing the worker to an area that is outside their current field of view. The panoramic view occupied the lower end of the touch table.

• Four storage areas, two on each side of the shared visual space, to allow the helper to save a copy of the shared visual space. For instance, a particular instruction/gesture on a particular object may be reused in the collaborative task at a later stage of the collaboration.

Remote technical specifications

The platform draws on previous experience in the making of the REAL system, a commercial, wearable, low-power augmented reality system employing an optical see through visor (LiteEye 750) for remote maintenance in industrial scenarios. In particular, ReMoTe makes use of the XVR platform, a flexible, general-purpose framework for VR and AR development. The architecture of the system is organized around two main computing components: the worker wearable device and the helper station, as seen in Figure 6.

Wearable computers usually have lower computing capability compared to desktop computers. To take into account the usual shortcomings of these platforms, software has been developed using an Intel Atom N450 as a target CPU (running Microsoft Windows XP). It presents reasonable heat dissipation requirement and peak power consumptions below 12 watts, easily allowing for battery operation. A Vuzix Wrap 920 HMD mounted on a safety helmet was used as the main display of the system. The arrangement of the display is such that the upper part of the workers field of view is occupied by the HMD screen. As a result, the content of the screen can be seen by the worker just looking up, while the lower part remains non-occluded. With such an arrangement, what is displayed on the HMD gets used as a reference, but then the worker performs all his/her actions by directly looking at the objects in front of him/her. CMOS USB camera (Microsoft Lifecam HD) is mounted on top of the worker’s helmet (as seen in Figure 6). This allows the helper to see what the worker is doing in his/her workspace. A headset is used for the worker-helper audio communication.
The main function of the wearable computer is to capture the live audio and video streams, compress them in order to allow network streaming at a reasonable low bit rate, and finally deal with typical network related issues like packet loss and jitter compensation. To minimize latency a low level communication protocol based on UDP packets is used, data redundancy and forward error correction, gives the ability to simulate arbitrary values of compression/decompression/network latency, with a minimum measured value around 100 ms. Google’s VP8 video compressor is used for video encoding/decoding, and the Open Source SPEEX library is used for audio, with a sampling rate of 8 kHz. It should be noted that at the same time the wearable computer also acts as a video/audio decoder, as it receives live streams from the helper station and renders them to the local worker.

The main component of the helper station is a large (44 inches) touch-enabled display. The display is driven by NVidia GeForce graphic card mounted on a Dual Core 2.0 GHz Intel workstation (Windows XP). The full surface of the screen is used as a touch-enabled interface, as depicted in Figure 7.

Occupying the central portion of the screen is an area that shows the video stream captured by the remote worker camera: it is on this area that the helper is using his/her hands to guide the worker. On the side of the live stream, there are four slots, initially empty, where at any moment it is possible to copy the current image of the stream. This can be useful to store images of particular importance for the collaborative task, or snapshots of locations/objects that are recurrent in the workspace. Another high-resolution webcam (Microsoft Lifecam HD) is mounted on a fixed support attached to the frame of the screen, and positioned to capture the area on the screen where the video stream is displayed in Figure 9: the camera capture what is shown on the touch screen (see arrow 1) and the hand performed by the helper over that area (see arrow 2). The resulting composition (original image plus the hand gesture on top) is once again compressed and streamed to the remote worker, to be displayed on the HMD (see arrow 3). The overall flow of information is represented in the diagram of Figure 8.

Remote design and initial testing

Four design iterations of our UI were performed, testing and validating each design with a set of representative end users on the following three maintenance/repair tasks (Figure 9):

- Repairing a photocopy machine;
Over 12 people have used and trialled the system, providing valuable feedback on how to improve the helper UI and more specifically the interactive aspect of the UI: the selection of a view, the changing of the view in the shared visual space and the storage of a view. The aim was to perform these operations in a consistent and intuitive manner, for ease of use. The overall response from the representative end users pool is that the system is quite intuitive and easy to use. No discomfort has been reported to date with the near eye display of the worker system.

**FUTURE WORK**

The next step in the development of the augmented reality system is to investigate the expansion of the current system to a mobile helper station. In the remote guiding system currently developed the gesture guidance is supported by a large touch table. A fully mobile remote guiding system using similar technologies for the two parts of the system, the expert station and the operator station, will be easily deployable and adaptable in the mining industry.

Currently a rugged version of the system is being engineered for initial field deployment and field studies. Industry deployment and the study of the system in use in its real context is crucial in understanding the human factors and issues prior to prototype development and commercialisation of the system.

The deployment of a rugged ReMoTe system to a mine site would allow investigation of the following questions:

- What is required for mining operators to use the system effectively?
- What measurable benefits can be achieved from the system use in a mine, such as, productivity and safety?
• What ROI on maintenance cost could be obtained by means of a large deployment of several similar units?

REFERENCES


AN INTEGRATED APPROACH TO IMPROVING SAFETY AND EFFICIENCY THROUGH COMMUNICATIONS, TAGGING AND COLLISION AVOIDANCE SYSTEMS

Brian Nicholls¹ and Tony Napier²

ABSTRACT: The use of tracking and collision avoidance devices coupled with better communication facilities for personnel are shown to be important not only for reducing accidents but also for managing emergencies.

INTRODUCTION

Recent vehicle to operator and vehicle to vehicle accidents (some resulting in serious injuries or fatalities) have highlighted the need to introduce better communication systems into the mining industry.

Systems currently available include methods of collision avoidance, tracking devices and facilities for personnel communication.

UNDERGROUND PERSONAL COMMUNICATIONS

The availability of adequate communications is of benefit in the effective deployment of machines and personnel but is of particular importance in dealing with emergencies that may involve evacuation from the mine.

Some aspects of such Communication systems as shown in Figure 1 are:

- Operator to operator communications are now available through the two way radio systems. This equipment enables mine wide communications to be achieved via a base station, amplifiers and leaky feeder cable run throughout the main transport roads in the underground mine complex. Other vital areas of the mine (main conveyor drives / underground workshops) can be accessed via branches off the main leaky feeder line.

- Hand held Intrinsically Safe (IS) two way radios can be installed with up to 16 independent channels. These units are suitable for segregated voice communications for various mine operations longwall/development/maintenance or can be used for mine wide emergency messaging.

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Where required, using specifically located antennae, two way communication can be achieved in roadways without the installation of the leaky feeder cable.

Information can also be relayed from the leaky feeder system via access points through the mine fibre optics cables installed for high speed data transmission shown in Figure 2.

Radios can also be used as independent units without the necessity of installing the leaky feeder cables in specific areas of the mine such as during longwall relocation tasks.

Figure 2 - Intrinsically safe “hotspots” to provide access on to the Network, for VoIP, high speed data and video monitoring

TRACKING AND TAGGING

When tracking and tagging components are used the following advantages can be assured:

- The location of all personnel and mobile machines (including equipment ancillaries) can be available in real time on a constant 24/7 basis in the mine operations control room, as shown in Figure 3.

Figure 3 - Pantha screenshots allow unlimited options for displaying tracking information and system maintenance/performance data

- Increased operational efficiencies and particularly in the event of accidents or an emergency. The importance of knowing the location of individuals within the mine workings in case of a major
emergency requiring evacuation cannot be overstated. Combining this with reliable two-way personal radio communication will make dealing with any emergency much more effective.

- When mine control officers know the location of all personnel and machines in the mine, (including within the various operating panels much more effective deployment of people and machines can be achieved, thereby improving overall utilisation of these resources.

**COLLISION AVOIDANCE**

Collision avoidance devices are an important aspect of accident prevention not only from the point of view of personal injury but also the costs involved.

The following comments apply to the application of such devices:

- The multi-technology collision avoidance system is designed to facilitate bi-directional notification and alert warnings against potential collisions.

- It is recognised that no single detection technology is currently capable of providing all of the required information needed to predict a dangerous proximity in a reliable and optimal manner. It follows then that the use of multiple detection technologies concurrently will provide the maximum protection envelope required within the confined underground mine environment. Figure 4 shows the multiple warning zones provided by multiple detection technologies.

- The systems used have to be reliable, repeatable and must ensure that the information conveyed to the vehicle /machine operator and the miner is in such a format as to minimise the annoyance factor. If this is not achieved, it may result in the operator ignoring the warning or possibly turning off the information source.

- While seeming complex, the use of various technologies does result in an effective zoning of alarm or information levels between the machines and adjacent personnel.

- It is believed that collision avoidance and proximity detection technology will be mandated by the mining inspectorates in the major mining states of N.S.W, Queensland and W. A. within the next two years.

![Image: Multiple warning zones provided by multiple detection technologies](image)

**CONCLUSIONS**

Improvements in the use of control equipment for mining machinery and the availability of better communication facilities are considered to be essential for efficient management and maintenance of safe working conditions.
DIGITAL NETWORKS AND APPLICATIONS IN UNDERGROUND COAL MINES

Denis Kent

ABSTRACT: An increase in remote monitoring and automation of many aspects of mining operations has seen the demand for more reliable and higher bandwidth communication systems in coal mines.

Digital networks have allowed the convergence of many applications onto a single communication backbone in general industry. The opportunity for a similar consolidation of communication infrastructure has driven the introduction of IP based networks into underground mines. One important application has been tracking personnel and equipment via RFID Tags. This tracking application has in turn formed the basis of a Proximity Detection System or a Collision Avoidance System.

This convergence of technologies is discussed together with the experience in coal mines of the implementation of new applications, including:

- Wi-Fi devices, such as intrinsically safe VoIP Telephones and RFID Tracking;
- Network design considerations;
- PDA’s, for paperless reporting systems.

The growing industry push for Proximity Detection Systems and the initial implementation of Proximity Detection Systems, and in the underground mining environment is discussed together with work on a Longwall Tracking System.

INTRODUCTION

The constant push for productivity and safety improvements has driven the development and adoption of more advanced automation, remote monitoring and control systems. The implementation of these systems and successful management of the operation as a whole is dependent on quality communications to and from the underground areas.

Though emerging technologies do need to take into account the limitations imposed by a working environment that is encased in solid rock, Mine Site Technologies (MST) identified the advantages of the so called “convergence of technologies” that digital and IP based communications have been giving general industry and which could offer similar benefits to the underground mining industry.

DEVELOPMENT OF DIGITAL TECHNOLOGY

MST’s roadmap was focussed onto a digital technology path about seven years ago, beginning with the development of a digital backbone for underground hard rock mines.

Rather than run a number of separate data backbones, the ability to use an Ethernet based data highway with wireless access through the Wi-Fi 802.11 b/g protocols was an obvious approach to take. This backbone not only provided a significant step change in bandwidth from more traditional leaky feeder systems or copper data networks, but also:

- Leveraged off the optic fibre network some mines had installed but were not fully utilising.
- Provided an optic fibre network for mines that did not have any fibre installed.
- Developed a network topology suited for mines, particularly in ease of installation and maintenance over time. Basically a digital network that didn’t need rocket scientists to extend or replace hardware underground.

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• Allowed the use of the many available industrial devices using Wi-Fi and Ethernet protocols, such as PDA’s, IP Video Cameras, VoIP Telephones, Lap Tops, basically any Wi-Fi or Ethernet connectable device.

• Through MST’s development roadmap not only provided a suitable network but some of the specific devices in a mine worthy form, including intrinsic safe versions for coal mines. These include Wi-Fi RFID Tags, VoIP Telephone handset, and the Wireless Access Point and Network Switch.

• Developed Mine specific applications such as Personnel and Asset Tracking. Other applications included a Vehicle Intelligence Platform for remote, real-time diagnostic uploads, event reporting and alarming, and production cycle monitoring. Then the user interface for the various applications was consolidated into a single web based user software package known as MineDash, for viewing, reporting and managing the digital systems and their respective applications.

Summarising, the development path for the digital systems, now known as our ImPact technology suite, included the digital backbone and specific devices and applications for mines.

In particular the use of digital 802.11 based technology gives an open protocol system with the signal quality and bandwidth to meet increasing levels of remote control and automation planned over the coming decades. This process set the platform for refinement of a general tracking system into a Proximity Detection System (PDS).

Considerable evidence has been presented in the form of statistics by others, including by Queensland Department of Employment, Economic Development and Innovation (Qld DEEDI) during a series of Proximity Workshops in 2009 to clearly demonstrate that people-vehicle and vehicle-vehicle collisions are a major factor in the many fatalities in the mining sector in Australia. These statistics are similar in other mining countries, including the USA and South Africa. Hence the justification for an extra level of risk control around vehicles is well acknowledged. To this end MST has been working with several industry partners over the last 5 years to develop a PDS. Key partners include Australian Coal Association Research Program (ACARP), Xstrata Zinc and Xstrata Coal.

Collision Avoidance Systems (CAS) have been developed, most industry progress has been made in the surface environment, particularly for heavy-vehicle and light-vehicle interactions. This led to MST’s decision to focus on their core market in underground mining where no proven or widely deployed system existed.

The development has been in two stages, firstly for hard rock mines and most recently for underground coal mines.

**Hard rock mines**

As a way of introducing the development of a proximity system for coal mine applications, a brief background development of systems used in hard rock mines is worthwhile. The basic concept is to use active Tags worn by miners underground to be detected by vehicle mounted Readers. The Reader is an existing Vehicle Intelligence Platform (VIP) Wi-Fi enabled data logger module. This VIP module is interfaced to a display unit to alert the driver of a person encroaching within the vehicles vicinity. The Display Unit will also provide the interface between the operator and the system and a means to acknowledge Tags and other necessary controls, such as alert outputs.

Outer Zone – 60 to 120 m:

• Gives a first warning to operator that there is someone around.
• The detection range is roughly adjustable from 60-120 m.
• Importantly, it can detect personnel around corners and blind-spots.

Inner Zone – 5 to 15 m:

• Uses a Very Low Frequency (VLF) magnetic field to give a stable electromagnetic field as the detection zone around the vehicle.
• The detection range adjustable from 5-15 m, adjustable in 1m increments.
- Triggers a higher-level audible and visible alarm.

These two levels of warning are shown in Figure 1.

![Figure 1 - The proximity detection system is based on two warning zones](image)

The first installed system was at Xstrata Zinc's George Fisher Mine. This involved a staged approach of adapting the technology tested on outbye vehicles in coal mines to the hard rock mining environment.

The project team consisted of a broad range of skills, including:

- Xstrata loader operators
- Xstrata electrical and maintenance engineers
- Xstrata management
- MST electronic and software engineers, communication technicians and project manager.

### Coal mines

The on-vehicle equipment used in the hard rock PDS were always going to present a challenge for coal mine certification. In addition, the complex situation and control needed around the mining face with continuous miners and shuttle cars required additional technology development.

Before developing this additional technology MST did a review of other PDS and CAS technologies to determine if a technology existed that could be directly applied or adapted to coal mine requirements. This extensive investigation has led to an alliance with Frederick Mining Controls out of the United States for integrating their HazardAvert® System into MST's overall PDS.

**The need for a proximity detection system:**

Since the introduction of remote controls in the mid-1980s, the United States mining industry has experienced 31 crushing or pinning type fatal accidents associated with the operation of remote control continuous miners as shown in Figure 2. Remote controls offered increased safety and health benefits to continuous mining machine operators by removing them from the noise and dust exposure of on-board operation, but subjected the operator to new crushing and pinning hazards.

![Figure 2 - Crushing or pinning fatal accidents – MSHA Statistics (Chirdon, 2009)](image)
In Figure 2 the circles represent operators, the squares represent helpers and the greyed areas indicate fatalities during maintenance operations.

“MSHA conducted a review of all mining-related fatal accidents from the last five years. It was determined that approximately 20% of all mining-related deaths could be prevented through the use of proximity detection.” (Chirdon, 2009)

The United States’ National Institute of Occupational Safety and Health (NIOSH) has complied statistics for “Struck by Accidents” between 2004 and 2008 for underground coal. The data indicates many partial and permanent disabilities (Bartels, et al., 2008).

In Australia 2009 statistics released by the Qld DEEDI revealed that 35% of fatalities within Queensland mines involved vehicle interaction. A sampling of incidents in Australia is highlighted in the Mines Inspectorate Safety Alert 237 (Qld DEEDI, 2009).

HazardAvert® is an adaptation of a PDS which was originally developed by NIOSH-PRL personnel in the US to warn underground coal miners whenever they get too close to continuous mining machines. Since its creation, HazardAvert® has not only been applied to continuous miners but also to Shuttle Cars, Load-Haul-Dumps, Haul-Trucks, Roof Bolters, Drag-Lines, Feeder-Breakers, Light Duty Vehicles, Fork Lifts, and other machinery.

**The most important requirement for an effective PDS:**

Areas around machinery where injuries have occurred need to be highlighted and marked in some way so that the workers and the vehicle operators are made aware of the danger. The danger zone marker must be robust and should not change, even when in open space or close to coal pillars and other equipment. Marker zone changes would reduce confidence which is not conducive to an effective safety system.

A marker signal in effect is limited to low frequency signals. The frequency chosen must be low enough that it does not propagate. Signals above around 100 kHz start attaching to any piece of conductive material and can propagate for miles. Determining the central focus of a danger marker would be extremely challenging at signals greater than 100 kHz. Low frequency signals can penetrate almost anything, including; coal, rock, dust, water sprays, metalliferous ore, as well as metal. This fact obviously makes low frequencies an excellent marker choice.

The frequency of the marker however must be high enough so that it is out of range of the high energy, low frequency electromagnetic noise always present in mines. The levels of the electromagnetic noise generated can overwhelm the best designed electronic system in close proximity to the source. Fortunately the magnitude of the electromagnetic noise decreases rapidly at short distances from the source. After many measurements in mines near various machines (using a spectrum analyser) 30 kHz was found to be about the high end of expected electromagnetic noise near heavy machinery. Hence, it follows that the best choice for a marker signal would be at the sweet spot between 30 kHz and 100 kHz.

**HazardAvert® system components:**

The basic system components of the HazardAvert® system are a Generator, a Personnel Alert Device (PAD), and a Vehicle Alert Device (VAD). Other peripheral devices are added depending on the environment in which the system will be used and the type of vehicle to which it is attached. A select group of these components will be described.

A Generator consists of microprocessors, an electromagnetic marker circuit, a warning module, a wireless data link, and can contain a VAD. The Generator is contained in a mine worthy rugged enclosure. There are at least two packaging arrangements for the generator; one an FLP housing (see Figure 3), and the second non-FLP housing for most other applications.

The Generator marker signal frequency is 73 kHz, which is at the sweet spot for marker signals. The size of the HazardAvert® marker field is based on the amount of energy put into magnetic marker field. HazardAvert® can project a marker field up to 30 m from its centre. The size of the field can be adjusted to the requirements of the application (see Figure 4). Additionally, the size of the field can be dynamically adjusted if accurate vehicle speed information can be acquired. This lends its use to mine operations where many vehicles work in close proximity to one another.
Figure 3 - HazardAvert on-machine components

Figure 4 - A number of stable detection fields around a continuous miner form the basis of the HazardAvert® system

Also contained in the Generator is a wireless data link which provides communications with PAD’s and other devices. The system warning module is placed within the operators viewing range in the vehicles cab. In many cases, a single Generator can provide the protection needed around a vehicle. Larger vehicles however can be accommodated using two or more Generators.

A PAD is worn by a worker on foot. It is a highly accurate multi-axis magnetic field measurement device with a wireless data link. The magnitude of the Generators magnetic field as measured by the PAD determines the distance to the Generator. The PAD includes a warning module which can be attached to the brim of a hard hat. The PAD is calibrated at the factory to provide a Warning Alert at one distance from a Generator and a Danger Alert at another distance from a Generator. The alerts provided are audible and visual and are programmed to suit the needs of the application. For Australia PAD’s are currently being integrated into the ICCL cap lamp.

To provide vehicle to vehicle detection a VAD needs to be installed on all vehicles. A VAD is a PAD integrated into the housing of a Generator. Its purpose is to detect when another vehicle, which contains a Generator, is within its Warning/Danger zone. Subsequently, the operators of both vehicles will be alerted to the other vehicles presence.

HazardAvert® installations:

A number of trials and demonstrations of the HazardAvert® System have been undertaken in Australia, one of the most recent at Anglo Coal’s Grasstrees Coal Mine. A longer term trial is also on-going in a hard rock mine, at BHPB’s Cannington Mine. However, longer term trials and full installations at coal mines have not taken place in Australia, due to certification still pending. With that in mind, a brief description of the most extensive HazardAvert® deployment to date, at Sasol coal mines in South Africa, follows.

PDS activity in South Africa began with Sasol’s ZERO harm mining strategy. A company team was assembled to investigate the risks for workers in proximity of mining equipment and to determine what solutions were available to address the risks. The team created a list of basic performance requirements for the system. The system had to:
• Work on all underground machines;
• Warn all miners and operators of the dangers;
• Slow down and then stop the machine;
• Work with multiple machines with multiple people;
• Have every machine respond to every miner;
• Individually warn every miner;
• Indicate to the operator the highest priority warning/danger;
• Be active with two-way verification;
• Be inherently safe;
• Provide robust marked zones.

HazardAvert® was further developed to meet all the requirements. Of the vendors reviewed, the HazardAvert® system was chosen for trial. The trial was conducted at Sasol’s Twistdraai Mine and was concluded in January of 2009.

The Trial Findings (Duvenhage, 2009) were:

• System very reliable
• Equipment damage negligible
• System consistency very good
• No false indications
• Operator feedback very good
• Data collected very useful to address operational concerns
• Interlocks worked very well
• Machine coverage very good
• Dramatically improved overall safety awareness

The success of the Sasol trial resulted in a complete mine roll-out strategy, where approximately 65 working sections are fitted with the HazardAvert® System. HazardAvert® is now on a few hundred underground machines including continuous miners, shuttle cars, load-haul-dump vehicles, and rotary breakers.

**Proximity detection and HazardAvert® activity in Australia**

MST and Frederick Mining Controls have formed an alliance to integrate the HazardAvert® System into MST’s Proximity Detection System. Though particularly relevant for coal mine use, this integration also allows another option for underground hard rock mines where confined area machine control is required.

The status of the activities in introducing the HazardAvert technology include:

• Continuing work with BHPB Cannington on extending beyond the initial trials into a longer term deployment on all vehicles.
• Integration of the PAD into the Integrated Communications Cap lamp (ICCL) to ensure the PAD module is always with a miner underground and part of their Personal Protective Equipment (PPE).
• I.S. Certification of the PAD/ICCL unit.
• Certification of the flameproof components of the on-vehicle devices (VADs, Generators, and Controller).
• Integration of MST’s existing Wi-Fi based proximity system with HazardAvert®. In particular incorporating the “Outer Zone” detection system with the HazardAvert® “Inner Zone” detection capability.
• Integration of MST’s Vehicle Intelligence Platform (VIP). This will allow real-time reporting of proximity events and alarms via a Wi-Fi communication link with a mine’s digital network, where this digital backbone is installed.
• A Functional Safety Assessment to determine and confirm the Safety Integrity Levels (SIL) and Functional Safety requirements for such a system, particularly as the system does have the capability to stop or slow vehicles where required.
LONGWALL TRACKING

Of interest to coal miners is a current project to provide tracking of personnel moving along a longwall face. The need for such a system is being driven by the developments in automation of longwalls and the need to know where people are or, more precisely, that they are in a designated safe location for the remote or automatic operation of shields and face alignment pushes.

MST is working with a longwall manufacturer for the delivery of a longwall tracking system for Narrabri North’s longwall. The basic principle is to monitor the location of personnel as they move along the face from support to support. This is done by detecting a RFID Tag worn by each miner (integrated into their ICCL cap lamp). This RFID Tag is already used for general tracking of personnel as they move from surface to underground and through the mine.

For Longwall Tracking a special detection field generator or exciter is installed on each support along the face. This field is very similar to the field used as the “Inner Zone” detection field in the general hard rock PDS. Figure 5 shows the principle of the detection fields along the face that monitor the movement of people to the nearest roof support. However, in the longwall application it is used to raise an alert when a person leaves the field, not when they enter it as in the case of proximity detection.

![Image](image1.png)

**Figure 5 - Longwall tracking**

In the Narrabri North configuration, the safe zone is between the supports legs and the rear of the shield (see Figure 6).

![Image](image2.png)

**Figure 6 - Longwall safe zone**

CONCLUSIONS

The development and deployment of a Proximity Detection System has been seen to offer additional levels of control for people-vehicle interactions underground. The benefits of such systems need to be carefully introduced into a mine’s current systems and procedures. In particular their initial introduction should focus on their use as an adjunct to, and not a replacement of, existing procedures. This will ensure their benefits are realised and not compromised by individual complacency and/or de-sensitisation.
REFERENCES


OPTICALLY-POWERED UNDERGROUND COAL MINE COMMUNICATIONS

Garry Einicke¹, David Hainsworth¹, Lance Munday¹ and Tim Haight²

ABSTRACT: Emergency underground coal mine communication networks are discussed. Emergency communication systems should continue to operate following an explosion, major fire, roof collapse, or water inundation without relying on internal batteries or underground mine power. Optical fibre cables support Ethernet communication networks and power underground communication devices are proposed. The results of some tests with power-over-fibre technology have confirmed that 40 mW to 35 mW of power can be made available over a 0.5 km to 1 km length of fibre optic core, respectively. This is sufficient to power an underground SMS or email device.

INTRODUCTION

During 2006, a Mine Safety and Health Administration (MSHA) committee in the USA evaluated communication and tracking system technologies for use in underground mines. This effort was in response to the Sago and Alma mine accidents which indicated that functioning communication and tracking systems would benefit search and rescue efforts. MSHA received more than 100 proposals and selected six systems for underground mine testing. The committee reported that the tested systems lacked communication range and were susceptible to radio interference.

In 2008, NIOSH, ACARP, Australian mining industry and CSIRO representatives attended a meeting and agreed to support mine emergency communication collaborative R and D. NIOSH subsequently advertised a Broad Agency Announcement that solicited research proposals for communication systems that could result in improved safety for mine workers. NIOSH advised that the communication system should satisfy the following requirements: the system should operate in underground coal mines, and can be used during routine operations as well as continued operation following an explosion, major fire, roof collapse, or water inundation; and the system should provide sustained operation without relying on internal batteries or underground mine power.

Currently installed underground communication systems and underground power supplies are not designed to withstand explosions, fires, roof falls or floods. Battery-powered communication devices are unlikely to operate for more than a week, whereas in the Beaconsfield Mine collapse of 2006, two miners were rescued two weeks after being trapped. Most recently, 33 Chilean miners were rescued after been trapped underground for 69 days.

In response to the above-mentioned NIOSH announcement, CSIRO researchers were tasked to investigate options and conduct lab tests for underground coal mine communication networks. This investigation established that direct-burial fibre optic cable buried at a depth of 0.6 m and subjected to pressures many times that exerted by a loader did not exhibit performance degradation. This observation was confirmed independently by NIOSH. That is, the same direct burial cable that is ploughed-in by telecommunication providers across Australia can be trenched within underground drives.

The trenched cable would be protected from fires, rock falls and floods. Trenching is done routinely within longwall mines for water drainage. For example, Hydrapower can supply a rock saw for developing 70-mm-wide underground trenches at a typical rate of 200 m per hour. A rock saw for cutting 600 mm deep trenches is estimated to cost about AUD $25,000. An approximate operating cost is AUD $20 per hour for wear and fuel usage.

The results of CSIRO tests with new power-over-fibre technology established that approximately 30 mW of power can be safely delivered to a communication device over a 1 km length of fibre optic core. The maximum optical power of the light source was sufficiently low such that any light energy that escaped the fibre (due to an unterminated cable or breakage) would be incapable of igniting explosive

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gas mixtures or coal dust (refer IEC 60079-28) which is consistent with intrinsically safe design. The authors are proposing a system for underground SMS or email communication that is powered exclusively by a few fibre optic cores connected to equipment at the surface.

This paper summarises the results of the underground coal mine communication investigation done for NIOSH (Einicke, et al., 2009). The communication system requirements are canvassed in Section 2. Options for underground and surface communication infrastructure are discussed in Sections 3 and 4, respectively. Section 5 outlines some candidate underground communications devices. Some remarks about the feasibility of the proposed approach are provided in Section 6 and the conclusions follow in Section 7.

COMMUNICATION SYSTEM REQUIREMENTS

Copper cable requirements

It is envisaged that separate copper cable is used to distribute power to emergency mine communications equipment. The cable’s conductor size would be chosen such that it could provide sufficient voltage and current to all connected emergency communications equipment, some of which may be kilometres from a power source. Non-emergency communication equipment would need to be independently powered.

During mine emergency incidents, in which the underground AC supplies are switched off, an IS supply having an internal uninterruptable power supply (UPS) may sustain connected communication equipment for up to several hours. Suppose instead that AC supplies together with IS power supplies were installed at the surface. Assume also that a satisfactory copper conductor was available and the cables were installed down bore-holes to support underground communication equipment. Under these conditions, underground emergency communication devices could continue to operate indefinitely during emergencies.

Optical fibre cable requirements

A mix of cable types is advocated, such as a multi-core optical fibre cable for Ethernet communication, and a two-core copper cable for power. Optical fibre cable is proposed due to its inherently low signal loss per kilometre. In comparison, copper Ethernet cable is limited to 100m in length, which is not consistent with the communication distances that exist within underground mines.

Optical fibre is considered intrinsically safe, provided the light energy transmitted down the fibre is at or below a certain power level. For intrinsically safe optical devices in Group I hazardous areas, the IEC 60079-28 standard applies. It specifies a maximum optical power of 150 mW for continuous-wave radiation in the 380 nm to 10 μm band. Note that further research is being done into single-mode fibre. Due to its extremely small core size and the nature of the light exiting the fibre, optical power density can significantly higher than in multi-mode fibre. Hence only IS-certified devices with an optical power output at or below 150 mW and in the specified wavelength range may be attached to the optical fibre. The optical fibre cables will need to be suitably armoured so they remain operational under the following conditions: routine underground vehicular traffic; underground roof collapse; underground water inundation; and pressure waves resulting from underground explosions.

Power-over-fibre (POF) technology can be used to power small electrical loads. That is, laser diodes at the surface could be used to transmit power via fibre-optic cables installed down bore-holes to support IS emergency communication devices situated underground. Access at the surface should be possible in flat rural areas where mine operators have arrangements to enter leased or neighbouring farming land. Cable length design trade-offs may preclude the supply of power over copper cables for the deeper bore-holes within inaccessible terrains. The bore-holes would require sealing to minimise impact on underground ventilation performance.

Self-escape and mine rescue requirements

Communications are required to support two rescue phases. The first is the self rescue phase when people underground become aware of an incident and they use their own resources and systems available to them in the mine to leave the mine. The second is the aided rescue phase, where people are trapped in the mine by a physical impediment, fire or injury. Communication technology is required to
support an emergency management team's underground navigation and rescue activities. In the event of an underground fire or flooding, the underground mine power supply will almost certainly be switched off. Therefore, the technology of choice is that which supports two-way communication and operates independently of underground mine power.

**UNDERGROUND COMMUNICATION INFRASTRUCTURE OPTIONS**

A configuration involving underground node switches and power supplies is shown in Figure 1. Briefly, a trunk cable, which includes a fibre optic core bundle, is connected to underground nodes having IS Ethernet switches and IS power supplies. It is envisaged that the trunk cable would be routed to the core Ethernet switch situated next to the control room at the surface. The lengths of gates roads within longwall mines can vary from 3 - 5 km. Thus, the distance from a control room to the current working face may be 10 km.

*Figure 1 - A star configuration of underground unmanaged switches and power supplies. The network is susceptible to failure unless the underground trunk is protected from fire, floods and roof-falls.*

Underground patch panels are suggested for connecting mine-powered communication devices such as 802.11x WiFi equipment. That is, high-bandwidth communication is supported during routine production conditions. In the event of mine emergency conditions involving localised fire, flooding or roof-fall, equipment within some regions may be destroyed. However, the trunk will need to be protected so that communications elsewhere within the mine are supported. For example, as discussed previously, the trunk could be buried in the floor of roadways.

A 110 V AC supply is usually available near existing underground infrastructure. An off-the-shelf 110 V AC to 12 V DC IS power supply could be used to support an IS Ethernet switch. For example, the entity parameters of a 7 AH Holville IS power supply are $U_o = 12.6$ V, $I_o=2.34$ A, $P_o=29.48$ W, $C_o=9.6$ µF and $L_o=0.05$ mH. The entity parameters of an Ampcontrol IS Ethernet Switch are $U_i = 15$ V, $I_i = 2.5$ A, $P_i = 37$ W, $C_i = 0$ and $L_i = 0$. Suppose that this Ethernet switch draws 2 A at a minimum input voltage of 12 V and that the resistance of available copper cable is 5 Ω/km. Then the maximum permissible length of power cable is $0.6 \text{ V} / (2 \text{ A} \times 0.005 \text{ Ω/m}) = 60$ m. Assume that the IS Ethernet switch was installed underground and the IS power supply was located 500 m away at the surface. Then the available voltage would be $12.6 \text{ V} \times 6 \text{ Ω} / (6 \text{ Ω} + (500 \text{ m} \times 0.005 \text{ Ω/m})) = 8.9$ V, which is insufficient to power the switch.
To minimise the voltage drop over the copper, the cable resistance would need to be very low (less than 1 Ω/km), which would require a cable with a large cross-sectional area (25 mm² or greater). Such a cable would have a very high mass and would not have sufficient tensile strength to support its own weight over hundreds of meters. Additionally, the L/R ratio could be too high to be safely connected to any IS power supply. These example calculations demonstrate that an underground IS Ethernet switch requires its power supply to also be situated underground. This limitation applies irrespective of whether the power cable is installed within conduit or trenched under the drive.

**SURFACE COMMUNICATION INFRASTRUCTURE OPTIONS**

It is common practice to install IS Ethernet switches underground where other communication equipment is situated. The power supply cable to the IS Ethernet switch is typically installed in conduit. Although burying this cable under a roadway would provide extra physical protection, it is still within an explosive risk zone and so the power supply is required to be IS. The example calculation (above) demonstrates that a switch cannot be installed underground and powered by an IS supply on the surface. Therefore, it is proposed instead that the switches are located on the surface.

The IS certification process for communication equipment can take 2 – 3 years. Consequently, the capabilities of IS communication equipment tend to lag state-of-the-art technology. For example, managed IS switches having dual power supplies are not yet available in the marketplace. A ring topology involving managed switches at the surface is shown in Figure 2, which is more robust than the star topology of Figure 1. For example, if failure occurs within a fibre optic cable or a switch, a rapid spanning tree protocol (RSTP) can automatically reroute all communication traffic via the redundant paths. Switches supporting dual power supplies are advocated since they can tolerate a failure in one supply.

![Diagram of managed switches having dual power supplies](image)

*Figure 2 - Configuration of managed switches having dual power supplies. The node communication network will remain operable in the event that a single failure occurs within a connection or switch.*
COMMUNICATION DEVICE OPTIONS

VoIP telephone

Suppose that an IS VoIP telephone that plugs into an underground patch panel has been developed. The VoIP telephones would enable underground personnel to communicate with personnel in the control room or make outside calls. In the event of an emergency incident in which underground power is not available, the VoIP telephones should be operational for the time it takes able-bodied personnel to walk out of the mine. The phones could remain operational for up to several hours, depending on the state of the UPS. However, any personnel that are confused, stranded, injured or trapped for a longer time would not be guaranteed communications availability.

Assume that an IS VoIP telephone requires 500 mA at 12 V, i.e. 6 W. A 500 m cable with a 1.5 mm$^2$ cross-sectional area has a typical DC resistance of 7 Ohms and DC inductance of 0.4 mH. For a current of 500 mA, the voltage drop will be 3.5 V at 500 m over 500 m of cable. Therefore the cable could deliver 13.5 V @ 500mA from a 17 V power supply. IS power supplies with suitable entity parameters are available at these power levels. On the surface, an alternative to the IS power supply is to use a large-capacity battery (charged from any non-IS source). The battery would be connected to safety barriers, which would provide IS outputs of appropriate power levels.

Wireless messaging

Underground mine equipment relies heavily on Ethernet communications. An underground wireless local area network (WLAN) could be established by connecting wireless access points to underground patch panels, provided that: the WLAN equipment is IS; fibre optic Ethernet connections are used; and independent power supplies are installed. Northern Light Technologies offer IS wireless access points and a cap-lamp-powered wireless messaging system. An IS messaging capability would aid underground workforce productivity and safety. The access points need to be installed underground. After the loss of underground mine power, the WLAN operating time will depend on the state of their IS UPS. Similarly, the availability of the messaging system will depend on the state of a worker's cap-lamp battery.

Wired messaging / Email devices

Currently, underground communications equipment relying on batteries are unable sustain operation for a week or more. There is clearly need for a survivable communications system to respond to such situations with indefinite operational duration. Consequently, the authors are undertaking the development of an IS email/messaging device.

Figure 3 - Depiction of the proposed low-power email/messaging device. It is envisaged that the device would be powered via a small number of fibre optic cores.
FEASIBILITY CONSIDERATIONS

Cable crushing tests

The ground within underground drives tends to include rocks. Thus, a trencher possessing a rock saw will probably be needed for direct burying of cable in mines. It is also envisaged that a hopper that discharges bedding material may additionally be required for rocky environments. Therefore, the efficacy of candidate bedding material needs to be investigated.

Six 5-m-long direct-burial 6-core SMOF loose-tube cables were obtained from Optical Fibre Systems for crush testing. The cores were pre-terminated with ST connectors. A test rig was manufactured so that a cable could be buried at different depths with different sample material and loaded within a press. The main body of the rig consisted of a 720-mm-high vertical section of rolled steel pipe having wall thickness of 5 mm and an outer diameter of 270 mm. The bottom of the cylinder was bolted onto the bench of a 250 kN (i.e., 25 t) Instron 1342 press.

Samples of the following material were selected for testing: crusher dust, 1.59 t/m³ density; 5 mm gravel, 1.33 t/m³ density; 15 mm gravel, 1.30 t/m³ density; and 50 mm road-base, 1.26 t/m³ density. Photos of these samples are provided in Figure 4. For each sample material and load setting, an FLS-600 light source (set to 1310 nm) was connected to one end of each core and a WG OLP-18B optical power meter to the other end. The tests were stopped when the piston reached a depth of 200 mm within the bedding material.

Crushing could be heard continuously throughout the tests, and surprisingly, no core failures occurred. The cable crushing test results are plotted in Figure 5. The variations in the attenuation measurements occurred because dust accumulates on the exposed 9 µm fibre cores with each disconnection and connection. The pressure produced by a loader was estimated to be 23.5 t-f/m² (33.4 psi). Thus, with a safety factor of greater than 10, burying the cable together with crusher dust at a depth of 0.6 m should afford adequate protection. The results presented here are spot tests. It is suggested that decision makers need to conduct repeated testing with the actual mix of bedding material and excavated material available at mine sites.

Ball-park cost estimates

Boreholes of 100 mm diameter can be drilled with ±1 m cross-track accuracy for depths up to 300 m and ±3 m accuracy for depths up to 600 m. Anything deeper than 600 m requires “steering” to ensure that the drill accurately reaches the target region. The cost of drilling a 100 mm borehole is about AUD $100/m. Hence, for our example with an average depth of 300 m, the borehole costs are $30 000 * 5 = AUD $150 000. Typically, 100 m boreholes can be grout filled around cables for approximately AUD $50/m. Thus, the grouting costs for five 300 m boreholes are about $50 * 300 * 5 = AUD $75 000.

It will not always be possible to drill a hole to every desired location. For example, infrastructure and other obstructions may exist at the surface or geological faults may be present underground. In this case, some cable may have to be buried within an underground roadway. Since burying cable underground is not common practice, estimating its cost is difficult. An approximate estimate for burying
cable can be obtained by adding a levy of $250/m for underground coal mine work to the $100/m drilling costs, i.e., $350/m.

Figure 4 - Crusher dust sample (top left), 5 mm gravel sample (top right), 15 mm gravel sample (bottom left) and 50 mm road-base sample (bottom right).

The cost of optical fibre cable depends on the number of cores, armouring and whether it is multimode or single-mode. For example, typical retail costs of an 8-core single-mode armoured riser cable (that is suitable for aerial use and for direct burial) is AUD $2.60/m. Industrial-quality managed 24-port Ethernet
switches and 100BaseTX to 100BaseFX media converters can cost around AUD $2 000 and $500, respectively. Thus, the estimated cost of two surface switches and 14 media converters is about AUD $11 000.

The low-powered communications devices are required to be IS and it is expected that they may cost about AUD $4 000 each. It is estimated in (Einicke G, McPhee R, Munday L and Hainsworth D, 2009) that the start-up cost for installing the proposed network at a new longwall mine is around AUD $400k. The switches, low-power communication devices and the patch panels can be reused from one longwall to the next.

CONCLUSIONS

This paper has discussed options for emergency underground coal mine communication networks. Underground coal communications systems are required to be intrinsically safe, meet communication needs during both production and emergency conditions, and survive underground mine power outages. Intrinsic-safety design considerations prevent the use of copper cables to supply power from the surface to underground Ethernet switches. Therefore, it is proposed that Ethernet communication networks are installed at the surface.

Some theoretical estimates and the results of some tests with power-over-fibre technology have confirmed that 40 mW to 35 mW of power can be made available over a 0.5 km to 1 km length of fibre optic core, respectively. This is sufficient to power an underground SMS or email device. The surface equipment would employ enclosed 150 mW laser sources, which are no more hazardous than a laser printer. It is expected that similar low-power design techniques could subsequently be applied in the development of IS VoIP telephones. This would involve an end-to-end delay design trade-off, which is consistent with push-to-talk telephones and intercoms that are currently in use.

REFERENCES

Intrinsically Safe Communication and Tracking Technologies for Underground Coal Mines

L Munday, S Addinell, J Thompson and E Widzyk-Capehart

ABSTRACT: The CSIRO, with funding assistance from Japan Coal Energy Center (JCOAL) and The Australian Coal Association Research Program (ACARP), has developed, tested and secured IEC Ex ia certification via Simtars for an Ethernet Switch and via TestSafe for a Serial To Ethernet Converter (STEC) and a Paging and Location System (PLS). The STEC provides translation between serial data and Ethernet packets. Any device with a wired (i.e. copper) serial interface can be connected to the STEC; the device then becomes part of an optical fibre-based Ethernet network and is controllable and addressable using standard Ethernet communication protocols. The PLS is comprised of two devices: tags and nodes. The tags, small portable units that communicate wirelessly, can be attached to personnel and/or equipment within a mine. The tags have a paging function, enabling two-way messaging between personnel and the remote server (control room operator). The tags communicate with the remote server via the nodes; fixed units that communicate with each other and the control room via optical fibre. Tag locations are continuously tracked within the mine, providing the control room with the locations of all tagged personnel, vehicles and equipment. Prototypes of the technologies developed by CSIRO were tested and demonstrated at three mines.

INTRODUCTION

Underground mining is hazardous and no more so than in coal mines, with the risk of fire, methane gas and coal dust ignition. The best means of minimising these hazards are improved risk management, hazard control, personnel communication and personnel/equipment tracking. But none of these measures can be effectively implemented until there is a reliable communication network in place.

Such a communication network must be able operate at all times, even during loss of mine power, which is critical when the network is part of a safety system. In underground coal mines, equipment operating within an Explosive Risk Zone (ERZ), and that can operate independently of mine power must be approved as Intrinsically Safe (IS). This means that the equipment itself will not provide an ignition source, even if it develops a fault. The CSIRO’s Mining Technology Research Group, has developed a set of intrinsically safe devices that enable Ethernet communication and personnel and equipment tracking in underground coal mines. These devices are:

- Ethernet Switch;
- Serial to Ethernet Converter;
- Paging and Location System;
- Server and software algorithms.

Figure 1 shows a typical configuration of these components. The Ethernet Switch provides the core of the communication system. It routes all packets of information over multi-mode or single-mode fibre, using Ethernet-over-fibre protocols. The STEC is a translation device; it converts data between any serial format and Ethernet packets. The PLS nodes and tags provide messaging and tracking functionality and have unique IDs, which allows them to be addressed individually by the server. The PLS Charger is used to recharge the Tags’ batteries; it is not an intrinsically safe device and must remain in a Non-Explosive Risk Zone (NERZ).

The Server (in a NERZ or on the surface) runs the monitoring and control software, such as CSIRO’s fully-featured risk management and monitoring application known as Nesysx™. PLS functionality is integrated with Nesysx™, allowing node and tag positions to be displayed on a mine map. Tag positions are constantly updated and each tag may be associated with a person or with a piece of equipment. For more details on Nesysx™ refer to Haustein et al., 2008a and b.
A variety of media are used to provide the communication links: single-mode optical fibre for the long distance and high speed links, multi-mode optical fibre for the shorter range and lower speed links, copper wiring for the short range and low speed links, and radio frequencies for the wireless links.

Data can be sent to and from any device in the network using this software.

**Figure 1 - Communication system overview**

**ETHERNET SWITCH**

Figure 2 shows the assembled Ethernet Switch, that has been approved as intrinsically safe (IEC Ex.ia Group I) making it possible to use in underground coal environments. It is housed in an IP20 stainless steel enclosure, with dimensions of 270x220x82 mm (excluding protrusions).

**Figure 2 - Ethernet switch**

The Switch is layer-2 device used to route Ethernet packets. Use of the TCP/IP and UDP protocols over optical fibre provides many advantages: long transmission distances at high data rates, redundant data paths (i.e. spanning tree protocol) and an effectively unlimited number of devices that can be connected to the network. The Switch has five full-duplex optical fibre ports; ports 1, 2 and 3 adhere to the 100BaseFX specification. These ports operate at 100 megabits per second (Mbps) and may be
connected to multi-mode fibre (50/125 \(\mu m\) or 62.5/125 \(\mu m\)) that uses the 800 nm range or to single-mode fibre (9/125 \(\mu m\)) that uses the 1300 nm range. Ports 4 and 5 adhere to the 10BaseFL specification. These ports operate at 10 Mbps and may be connected only to multi-mode optical fibre (50/125 \(\mu m\) or 62.5/125 \(\mu m\)) that uses the 800nm range. Any combination of fibre connector types can be fitted to the unit at the time of manufacture. Two optical fibre cores are required per port; one for transmission and one for reception.

The Switch should be used in conjunction with an uninterruptible IS power supply (11-13V DC). This allows the Switch to operate independently of mains power, which is critical if the Switch is part of a safety system that works reliably, even in emergency situations. The production version of the Switch should be available in 2011 through CSIRO’s commercial partner, AmpControl.

**SERIAL TO ETHERNET CONVERTER**

Figure 3 shows the Serial To Ethernet Converter (STEC), which has been approved as intrinsically safe (IEC Ex.ia Group I) making it possible for use in underground coal environments. It is housed in a DIN-mountable IP20 enclosure, which has dimensions of 150x110x75 mm (excluding protrusions).

![Figure 3 - Serial to ethernet converter](image)

The STEC has one wired serial port and one optical Ethernet port; it translates between serial-over-copper packets and Ethernet-over-fibre packets.

The optical fibre port is full duplex and operates at 10 Mbps (10BaseFL). The port is fitted with ST connectors for multi-mode fibre (62.5/125 m) and uses an 850 m wavelength. Two optical fibre cores are required; one for transmission and one for reception. The STEC has its own MAC address and IP, which means the serial device connected to it becomes addressable using standard TCP/IP or UDP communication tools. The copper serial port may be configured as RS422 or RS485. The serial data rate can range from 300 to 115200 bits per second and any asynchronous serial protocol can be used. In particular, the STEC has been tested extensively with the Modbus-RTU protocol and the proprietary PLS serial protocol. The STEC’s user interface consists of an LCD and keypad, which allows the operator to configure and test the STEC in-situ. Many parameters can be set, including the MAC and IP address, the serial speed and the serial format.

The STEC should be used in conjunction with an uninterruptible IS power supply (9 to 13 V DC). This allows the STEC to operate independently of mains power, which is critical if the STEC is part of a safety system that must work reliably, even in emergency situations. The production version of the STEC should be available in 2011 through CSIRO’s commercial partner, AmpControl.

**PAGING AND LOCATION SYSTEM**

The PLS consists of two primary hardware devices: tags and nodes. Both devices are approved as intrinsically safe (IEC Ex.ia Group I), making it possible to use them in underground coal mine environments. Tags are mobile units that communicate with the nodes wirelessly. The nodes communicate with each other (and the server) via serial-over-fibre links. A third component of the PLS is the Charger. It is used to recharge the Tags’ internal batteries and can only be operated in a non-explosive risk zone.
Tags and nodes communicate wirelessly with each other and the server on a continuous basis; the use of unique device IDs allows tags and nodes to be addressed individually or in groups. Tags will generally be in wireless range of only 1 or 2 nodes at any time. This allows the server software to determine coarse tag locations.

Wireless communication is realised by a half-duplex radio frequency (RF) link, with the carrier set to either 433 MHz or 916 MHz (selectable at the time of manufacture). Most countries recognise one or both of these frequencies as falling within an ISM (Industrial, Scientific and Medical) band. These bands are license-free and may be used with little restriction. In Australia, the only restriction of these ISM bands is that (for a digitally-modulated, fixed-carrier transmitter) the maximum effective isotropic radiated power (EIRP) is 25 mW. The tag and node radios have maximum EIRP of 10 mW.

The 433 MHz and 916 MHz radio frequencies were also chosen as they exhibit good propagation characteristics in coal mine tunnels (Emslie, et al., 1975). Additionally, these lower UHF frequencies propagate a given distance using less power than higher UHF frequencies (such as 2.4 GHz). Low power usage is particularly important for the tag, since it is battery powered.

**PLS tag**

Figure 4 shows a photograph of a PLS Tag. It is housed in a small IP55 plastic enclosure, which has dimensions of 100x52x25 mm. The enclosure is translucent, allowing internal LEDs to illuminate the case.

The tag’s user interface consists of several parts, which are detailed below in Table 1.

![PLS Tag](image)

**Figure 4 - PLS tag**

**Table 1 - Tag user interface**

<table>
<thead>
<tr>
<th>Part</th>
<th>Description</th>
</tr>
</thead>
<tbody>
<tr>
<td>LCD display</td>
<td>Messages and status information is displayed on the LCD.</td>
</tr>
<tr>
<td>Soft buttons</td>
<td>Three membrane keys can be used for any purpose; e.g. for specific responses to incoming messages. In the current version of the firmware, the three buttons always correspond to a “Yes”, “No” or “OK” response.</td>
</tr>
<tr>
<td>Status LED</td>
<td>This dual-colour LED displays battery charging information and communications status information.</td>
</tr>
<tr>
<td>Vibration and Perimeter LEDs</td>
<td>A small vibration motor and a set of blue perimeter LEDs indicate communications and battery charging status.</td>
</tr>
<tr>
<td>Accelerometer</td>
<td>A 2-axis accelerometer senses vibration and tilt, which can be used to detect injured or incapacitated personnel.</td>
</tr>
<tr>
<td>Reed switch</td>
<td>This internal switch can be activated with a magnet in order to turn off the Tag.</td>
</tr>
</tbody>
</table>

On receipt of a message, the tag vibrates, the case illuminates and the message text is displayed on the LCD. The user can then respond to the message via the soft buttons. Additionally, a “Mayday” message can be sent by holding down any two buttons for five seconds.
PLS node

Figure 5 shows a prototype PLS Node. It is housed in an IP55 powder-coated steel enclosure, with dimensions of 300 x 200 x 100 mm.

A node has three serial ports and one wireless port. Data transits the ports at 19 200 bps using the 8N1 format. Ports 2 and 3 are optical fibre ports (multimode 62.5/1125 m), while Port 1 can be configured as either a multi-mode fibre port or as copper port (RS422/RS485). All port cabling passes through glands and is terminated inside the enclosure.

A node has eight LEDs which are used to indicate activity on the communications ports; there is a transmit and receive LED for each port. When power is applied, they will flash LEDs briefly to indicate successful node initialisation.

Packets are routed between nodes using a simple scheme: any valid message received on a node's serial port that is not intended for that node, is retransmitted on all ports except for the port it was received on. This ensures that packets propagate through the network and will reach the desired destination/s.

A node should be used in conjunction with an uninterruptible IS power supply (11-13 V DC). This allows a node to operate independently of mains power, which is critical if the node is part of a safety system that must work reliably, even in emergency situations.

PLS charger

Figure 6 shows a prototype PLS Charger; a production unit would feature a more robust style of charging dock and would have facility for charging many tags simultaneously. The charger is housed in a plastic enclosure with dimensions of 240 x 160 x 90 mm.

An inductive charging method is used, which allows tags to be simply placed into a slot for charging. Additionally, there are no exposed charging contacts, thus increasing safety and reliability. Charging a tag may take up to 6 hrs if the tag’s battery is near-flat.

FIELD TRIAL

Prototypes of the Ethernet Switch and STEC have been tested and demonstrated at several underground coal mines: Xstrata Beltana, Anglo Coal Grasstree and JCOAL Kushiro. The entire communication suite of devices and software was successfully installed and demonstrated at Kushiro mine; hence discussion of field trials will focus on the testing at this mine.
In November of 2006, the development team undertook a field trial of the Switch, STEC and PLS at the Kushiro coal mine in Japan. The system consisted of 1 Ethernet Switch, 2 STECS, 6 PLS tags, 3 PLS nodes and 1 PLS charger. The hardware was setup in a similar configuration as that shown in Figure 1. Figure 7 shows the underground locations for the nodes. Node 1 (not shown) was placed at the muster area on the surface. Node 2 was placed 1577 m down the main drift. Node 3 was first placed 1790m down the main drift (labelled as position “Node3A”). Node 3 was later moved to the position labelled “Node3B”.

![Figure 7 - Underground node locations](image)

The aim of the trial was to test the performance of the Switch, STEC, PLS and server software. Since the PLS relied on all the other components operating correctly, its performance became the focus of the trial. Particular attention was paid to the RF performance of the PLS, since little information was known on its RF propagation characteristics. All tags and nodes were configured to operate at 433 MHz. The average effective isotropic radiated power for all the tags and nodes was between 7 mW and 9 mW.

### Stage 1 testing

The primary aim of Stage 1 was to verify system operation and to test RF range in an above-ground office environment.

Note that the PLS tags can be configured to continuously display an “In Range of Node” or “Out of Range” message on their LCD, which was updated every few seconds. However, each tag’s EIRP and RF sensitivity is slightly different. This is inherent in the circuit design and primarily due to the RF chip used. Hence, it was important to determine an average RF range for the tags. To achieve this, the tags were moved to various locations until one tag of the six showed “Out of range”. The tags were then moved further away from the node until all tags showed “Out of range”. Hence, the average RF range was taken to be approximately half-way between these two locations. Average RF range will be hereafter simply referred to as “RF range” to simplify explanation.

The tags were first activated in the control room. All 6 tags showed “Out of Range”, which was not surprising considering that node 1 was two floors below and through several layers of steel and concrete (at least 50 m away in a straight line). The tags were then walked around the building, traversing several levels and passing close to node 1 on several occasions.

The result of this test showed that RF range varied from 30 m to 100 m and was highly dependent on the intervening materials between the tags and the node. With the tags on the same level as node 1, it was possible to walk up to 100 m away and still be in range. When traversing two levels above node 1, the RF range dropped to approximately 30 m. The building was constructed of steel-reinforced concrete so this result was not surprising, given that the radio frequency of interest (433 MHz) is readily absorbed by electrically conductive items.

Additionally, several messages were sent to the tags during the test. When the tags were in range of a node 1, or when they re-entered range, all messages were successfully received.
Stage 2 testing

Figures 8 and 9 show the underground environment in which the tags and nodes were operating. As can be seen, there is a significant amount of steel reinforcement and heavy copper cabling. The tracks shown in the photos are used by the drift train, which transfers workers between the muster area and the working area of the mine.

![Figure 8 - Main drift, near Node 1](image)

![Figure 9 - Side tunnel, near Node 3B](image)

Initially, the tags were walked up and down the drift, to traverse a few hundred metres either side of Node 2 and Node 3A. The tags were held in the hands of various CSIRO and Kushiro personnel, who were never more than 10m apart. The same method for determining RF range was used as detailed in stage 1. After several traversals of the drift, the average RF range for this test was found to be 120 m.

For the second test, the personnel holding tags boarded the drift train and the train traversed a few hundred metres either side of node 2 and node 3A. Some of the train doors were left open, while others were closed. Again the tags were hand-held and never more than 10 m apart. The RF range for this test was found to be 80 m. The lower RF ranges as compared to the first test can be explained by the shielding effect of the train's metal chassis.

The final test of stage 2 was designed to measure the overall communication latency when communicating with PLS tags. To achieve this, one of the CSIRO staff used the underground phone system to stay in contact with the server operator (situated in the control room). The operator then initiated a tag message and communicated this fact over the phone. Using a stopwatch, the underground personnel calculated the time for a message to be delivered. The tag user then responded to the message by pressing a button. Similarly, the time for the response to appear back at the server could be timed.

This procedure was repeated several times. The highest latency observed was 15 s, with tag messages and responses typically delivered within 5 s.

Stage 3 testing

The primary aims of stage 3 were to measure the RF range in an underground environment that contained lots of steel, and to test RF range in a non-line-of-sight situation.

As with Stage 2, a walk-test was performed a few hundred metres either side of the node (in this case node 3B). For this test all tags were placed on a plastic tray, and carried by one person, with the tray about 1m above ground level and held horizontally.

In the straight section of tunnel, the RF range was 110 m. In the other direction from the node (i.e. around the tunnel bend), the RF range was 90 m. Additionally, at one point a string of coal carts passed by on the track, which temporarily increased the RF range to 110 m. This 20 m effective increase in range was due to the coal carts acting as a waveguide.

As a final test for stage 3, the tags were taken into a side passage and through several brattice stoppings. The RF range was found to be 30 m. This was not surprising given the smaller aperture of the side passage and the presence of the stoppings.
Summary

Table 2 summarises the RF ranges observed during the various stages of the field trial. Several conclusions can be made from the Kushiro field trial. Firstly, the RF range of the tags and nodes is highly dependent on the materials in the surrounding environment. In a line-of-sight situation with no obstructions, the RF range was typically 110 m. Moving a tag out of sight of a node (i.e. moving slightly around a bend in tunnel) reduced the RF to approximately 90m. Moving into a side tunnel, which had a smaller aperture and non-metallic stoppings, significantly reduced the RF range to 30m. Conversely, the RF range actually increased by about 20 m when there was conducting structures or vehicles nearby.

**Table 2 - RF range results**

<table>
<thead>
<tr>
<th>Stage #</th>
<th>Test description</th>
<th>RF range (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Same level as node 1</td>
<td>100</td>
</tr>
<tr>
<td>1</td>
<td>Two levels up from node 1</td>
<td>30</td>
</tr>
<tr>
<td>2</td>
<td>On foot</td>
<td>120</td>
</tr>
<tr>
<td>2</td>
<td>On train</td>
<td>80</td>
</tr>
<tr>
<td>3</td>
<td>In straight tunnel</td>
<td>110</td>
</tr>
<tr>
<td>3</td>
<td>Around the bend</td>
<td>90</td>
</tr>
<tr>
<td>3</td>
<td>Side passage</td>
<td>30</td>
</tr>
</tbody>
</table>

**CONCLUSIONS**

This paper discussed three intrinsically safe communication devices, all approved as Ex.ia Group I and targeted for use in underground coal mines. The Switch provides core routing capability for an optical fibre network and enables Ethernet devices to be readily utilised underground. The STEC provides Ethernet-over-fibre to serial-over-copper translation, enabling a wide variety of serial devices to be connected into a mine's existing Ethernet fibre network. The PLS enhances productivity and safety with its messaging and tracking functionality. All of these communication devices, when used in conjunction with uninterruptable IS power supplies, will operate independently of mine power. This is critical if these devices form part of a safety system that must work reliably, even in emergency situations.

**REFERENCES**


THE NEXSYS™ REAL-TIME RISK MANAGEMENT AND DECISION SUPPORT SYSTEM: REDEFINING THE FUTURE OF MINE SAFETY

Kerstin Haustein¹, Eleonora Widzyk-Capehart¹, Peter Wang², Dean Kirkwood³ and Ricky Prout³

ABSTRACT: Underground coal mine control rooms are inundated with data but there remains a lack of information enabling timely decision making. Control room operators’ cognitive abilities are stretched beyond their limits; processing of the vast array of data sourced from multiple, non-compatible proprietary systems coupled with old communications systems makes the job of a control room operator extremely challenging, if not impossible, particularly in emergency situations when speed and accuracy are of great importance.

The CSIRO has developed a real-time risk management software called Nexsys™. Nexsys™ is a decision-support system designed for the collection and integration of disparate mine data, real-time analysis of safety critical data and real-time risk profiling using rules-based trigger action response plans. The system allows access to a wide range of risk management data in an easily interpretable format and in real-time, such as, availability of messaging and tracking data to precisely determine the location of all personnel, vehicles and equipment at all times. Through compatibility with an optical fibre underground communication network, which uses intrinsically-safe equipment and keeps the data network alive during power shutdown, an access to sensor data during emergency conditions (power shutdown) is available. The software system has a risk preventive and predictive capability, ability to track people and equipment underground, and the provision of 2-way communication via the operator’s interface. The system is designed to make risk management-related data immediately available to the operator and to reduce the amount of irrelevant and unnecessary data, such as false alarms.

Nexsys™ has the potential to radically reshape safety in the underground coal mining industry and has the future potential to be adapted to surface coal mining, metalliferous mining and non-mining applications, where safety and decision support are critical operational characteristics.

INTRODUCTION

The dangers associated with underground mining operations raise a compelling need for risk management and accident prevention. It is certainly true that the coal mine control room of 2010 is infinitely more sophisticated than its predecessors; however, during each shift, millions of bits of data can be transmitted into a control room from all areas of mine operation and covering everything from gas levels to temperature, movements of mining equipment, and personnel location. While significant advances in solving problems associated with transmitting data within mines has been made over the last several years, the challenge remains in the ability to analyse these massive amounts of data and convert it into useful information for both production and safety management. In particular, emergency situations place extreme demands on effective information management, both in the response to the development of a potentially safety-critical situation and, if unavoidable, during an incident itself.

Many incident and accident evaluations have shown that although predictive data was available, it was often too ambiguous, incomplete or scattered across a number of disparate proprietary systems to effectively deliver vital information to mine site personnel in a form that would allow appropriate pre-emptive responses (Einicke and Rowan, 2005). If this data had been properly managed and interpreted, it is likely that many of the incidents or accidents could have been prevented or their consequences reduced. Cliff and Grieves (2010) stated that “the control room in particular is a key area where accurate information is required during an incident especially in the early stages until a senior mine official can take charge. The control room remains the first point of contact during an incident for most personnel. Speedy evacuation and in-seam response is predicated upon knowing what is

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happening and where everyone is located”. They went on to say that currently it was often “impossible for him (the Control Room Operator) to carry out all his designated tasks in a timely and effective manner.” This is especially true during emergency situations.

To help control room operators and other mine officials provide accurate, timely information and deliver a Decision Support System for the mining industry, the Commonwealth Scientific and Industrial Research Organisation (CSIRO) in collaboration with Japan Coal Energy Centre (JCOAL) and with sponsorship from Australian Coal Association Research Program (ACARP) has developed the Nexsys™ Real-Time Risk Management and Decision Support System. Nexsys™, within its risk management analysis system, is capable of integrating and interpreting data from various proprietary systems to provide safety-critical hazard analysis in real-time, with reference to relevant industry standards.

Prototypes of the Nexsys™ software developed by CSIRO were demonstrated at three mines: the Xstrata Beltana mine, the Anglo Coal Grasstree mine and the JCOAL Kushiro mine, with a commercial version of the software planned for release in 2010 through CSIRO partner, Mining Logic Solutions.

The main characteristics of the Nexsys™ system are described and the outcomes of the system deployment at an underground coal mine in the Bowen Basin, Queensland through the partnership between CSIRO and Mining Logic Solutions are presented.

**NEXSYS™ SYSTEM**

In September-October 2002, Rowan et al., (2002) undertook a series of discussions with the representatives of the mining industry at Kestrel, Crinum, Oaky No. 1, Oaky North, Moranbah, Grasstree, Central Collery, North Goonyella and Newlands mines, which focused on the current status of the control room monitoring systems. Subsequently, a number of recommendations towards new developments were compiled addressing the following issues:

- Interface to CITECT, SafeGas and RSVIEW monitoring systems;
- Interface to real-time gas monitoring equipment such as Trolex;
- Reduction in the number of nuisance or false alarms;
- Improvements to the reconciliation/interpretation of underground data;
- Provision of information about the possible courses of actions in the event of critical alarms;
- Provision of appropriate Trigger Action Response Plans (TARPs), in the event of critical safety alarms, and
- Automatic generation of reports to alleviate the workload of control room operators, where possible.

This study formed the basis for the CSIRO-led Nexsys™ Project. Nexsys™ is a mine-wide hazard reporting system that monitors real-time critical data to detect potentially hazardous mine conditions. It integrates data from a range of proprietary systems and independent sensors within a single concise system, providing real-time safety-critical hazard analysis and enabling operators to make informed decisions in safety-related areas. Whereas other mine monitoring systems typically focus on specific aspects of a mine’s health and provide relatively simple data analysis and low-level decision support through alarms, Nexsys™ can draw information on the condition of the entire mine and provide a fully interpretative and preventive analysis. Through its data analysis capability over a multitude of domains, Nexsys™ is designed to reduce the uncertainty and variability in the interpretation of this data. It provides continuous monitoring, evaluation and reporting of mining conditions, supports operators in the decision-making process by providing information relevant to activities being undertaken, and allows rapid communication between mine site personnel.

**Nexsys™ characteristics**

There are several key features of the Nexsys™ system including risk profiling and rule engine, anomaly detection, data analysis for decision support, and mine wide communication and reporting. The Nexsys™ system has been developed using modern C#.NET programming methods. In general, the
system consists of a Server, a Client and a Database Management System, each of which can reside on separate machines located at various sites around the world. In mining applications, the system is usually located in a mine control room. The Client, however, can effectively be located anywhere on the Internet. A typical Nexsys™ installation at an underground coal mine might include a Nexsys™ server in the mine control room connected to and importing data from various proprietary mine computer systems and sensors. Nexsys™ Clients might be located in the mine control room, in offices, or on the laptops of various personnel on and off site. Nexsys™ could also be connected to a personnel and equipment locating system, which would transmit location information back to the Nexsys™ Server and hence the Nexsys™ Clients.

Risk profiling - rule engine

A key feature of the Nexsys™ system is its unique risk profiling matrix, dynamically populated by automated rules based on the mine’s TARPs, with each rule having a condition (potentially a rule or hierarchy of rules) that must be met to trigger a particular response. The Nexsys™ Risk Profile matrix (Figure 1), which is based on a standard mine risk profile template, is displayed in real-time on the likelihood-consequence diagram with a user-enabled view of the change in risk over a specified time period. A multitude of rules can be used to generate the overall risk profile.

![Risk Profile](image.png)

Figure 1 - Nexsys™ risk profile view for underground coal mining applications

The Rules Engine determines a set of actions to be taken based on the state of the data in the Nexsys™ database and a particular sequence of events. This engine can interrogate any data in the system using Boolean logic comparisons. Rules can be defined for each installation of the system at different mines and may be grouped to enable testing of a particular rule in relation to the outcome of another rule. The grouping feature is particularly important when an event, such as equipment stoppage, triggers another event, such as stoppages of equipment down the line.

Nexsys™ provides various alarms triggered by the Rules Engine or the system. When an alarm is raised, a diagnostic analysis is performed, which generates appropriate response plans. This information can be forwarded to the appropriate personnel by way of email/SMS, messaging personnel using a mine messaging system or by updating the Nexsys™ Risk Profile, which compels the personnel to undertake defined actions. For example, if gas sensors readings are above a specified level triggering an evacuation action, an alarm is raised and an evacuation message (as per the appropriate TARP) is sent to miners carrying paging devices in a designated area. The alarms are divided into three categories:

- New Alarms, which have not been acknowledged by the operator,
• Action Alarms, which have been acknowledged and also require a response from the operator, and
• No Action Alarms, which have been acknowledged but are only alerts requiring no response from the operator.

Each Alarm is categorised as high or low, gas or equipment-related, and, if applicable, its TARP-level is listed. The control room operator can initiate automated responses from the Action Alarms, such as notifying appropriate personnel of their required actions. All triggered alarms are displayed on the operators Alarms View (Figure 2).

**Figure 2 - Nexsys™ alarms view**

**Anomaly detection**

Another unique attribute of the Nexsys™ system is its predictive and anomaly detection functionality, which uses historical mine data to predict future hazards, evaluate associated risks, and eliminate false alarms. This feature enables pre-emptive and preventive actions to be initiated before an event reaches a critical state or, in case of false alarms, it can indicate whether a certain state of events is, in reality, a non-dangerous event despite above-threshold sensor readings (Figure 3). The results of the predictive analysis and anomaly detection are used to trigger alarms and update the Nexsys™ Risk Profile. The Anomaly Detection module is integrated with the Rules Engine enabling prediction of sensor values and anomaly detection in the available data.

**User interface and decision support**

The user interface (UI) development for the Nexsys™ Real-Time Risk Management System focused on the design that would allow the system operator to monitor human and machine activities throughout the mine, display sensor information, and provide the operator with information to allow rapid assessment of emergency situations and guidance for corrective actions. The design of the user interface took into account human capabilities and limitations to ensure ease of use and improve operational performance as well as to enhance safety and user satisfaction while reducing operation errors, operator stress, and user fatigue.
The Nexsys™ UI design process followed typical steps of iterative design (Carrol, 1997) based on the concept of iteration within the usability engineering lifecycle design (Porter, 1964), that is:

- An initial interface design was completed based on the designers’ knowledge of Nexsys™ system requirements and capabilities (both Human Factors and mining experts were involved).
- The design was presented to several test (end) users, a process during which various mine site personnel were interviewed. These interviews were conducted at the mine site to enable assessment of the current end-users working environment in terms of their cognitive abilities. Initial interviews of mine personnel were conducted to: (1) obtain information on the use of the existing user interface, (2) determine deficiencies in the current system, and (3) learn of desired features for the future interface. Control room operators, engineers, and supervisors were interviewed to determine the specific needs of personnel for the delivery of information.
- Any problems identified by the end-users were used in the development of user interface requirements. This material was used to produce a set of drawings of the UI, which were subsequently presented to the relevant mine site personnel during follow-up interviews to solicit further feedback and refine the design (several iterations).
- During the final user interface-related site visit the acceptance of the design by site personnel was sought. Final minor refinements were then incorporated within the final UI design.
- Assessment of user interface compatibility with Nexsys™ software was then undertaken by software engineers.
- The final user interface design was completed and transferred for implementation within the site-deployed Nexsys™ software.

The resultant UI provides an integrated view of mine-wide data via Alarms, Risk Profile, Charts, Reports and Mine Plan views. The Mine Plan View (Figure 4) can display information, such as gas and equipment sensor readings and locations, personnel locations (including messaging), views from video cameras, mine ingresses and egresses and airways. This concise view allows the operator to make quick decisions in emergency situations. For example, they are able to locate personnel, exits, gas leaks and clean airways and then message these personnel to notify them of a safe exit in the case of evacuation. All the information the operator needs to make these decisions is spatially located in the same place so that the relationships between all the objects and features in and of the mine can be easily identified and correlated.

In addition, Nexsys™ can provide mine-wide reporting to all levels of personnel, offering many mediums of communications from messaging personnel via Northern Lights messaging to their cap lamps, SMS capability and email. This reporting occurs by way of Alarms, automated Trigger Action response Plans (TARPs) and standard Reports. The location function on the Northern Lights tracking and messaging system assists with this task, as the operator can select which staff member to contact based on their location within the mine.
The Nexsys™ system was originally developed for the purpose of increasing coal mine safety and, as such, a prototype system was successfully demonstrated in 2005/2006 at three mine sites: Beltana and Grasstree Longwall Coal Mines in Australia and the Kushiro Longwall Coal Mine on the northern Japanese island of Hokkaido. During the original Grasstree trials of the Nexsys™ prototype system, Nexsys™ was connected to Citect’s Supervisory Control and Data Acquisition (SCADA) system and Location and Monitoring for Personal Safety (LAMPS), a CSIRO-developed system for personnel location and communications. During testing, Nexsys™ was able to send a signal to miners located underground through LAMPS, notifying them to contact the mine control room. Sensors detected gas levels (carbon monoxide, carbon dioxide, oxygen and methane) and ventilation status. Rules were created to trigger alarms and send emails to appropriate personnel if the risk produced by defined combinations of sensor readings exceeded safety thresholds. The research prototypes of the Nexsys™ system at all three mines continued to run for up to several months providing invaluable data for further development of the system.

Nexsys™ deployment at Grasstree Mine

In September 2009, Mining Logic Solutions and CSIRO commenced a 12-month trial of the Nexsys™ pre-commercial system at Grasstree Mine. The overall aim of the trial was to confirm the Nexsys™ Real-Time Risk Management System as a state of the art safety software product feasible for commercial deployment within underground coal mining conditions. To determine the viability of the system, the trial had two distinct goals:

- to demonstrate that vital parts of the software operated correctly in a full operating environment, and
- to create and evaluate a specific user interface that contained all the required information for different levels of personnel employment status.

Nexsys™ trial - functionality

The functionality of the following components of the Nexsys™ system in full operating underground mining conditions was tested:

- connectors for man tracking and SCADA database;
• real time mine plan viewer;
• rules engine, and
• anomaly detection and predictive data analysis.

The Nexsys™ system received data from two proprietary monitoring systems used at the mine site: the Citect’s Supervisory Control and Data Acquisition (SCADA) System and the Northern Lights Technologies (NLT) Man and Asset Tracking System.

For the Citect Connector, the Nexsys™ Server collected data from over 40,000 sensors, distributed throughout the mine, in real time without error. In addition, service properties were successfully set to restart automatically on service failure. The criteria for successful implementation of the NLT Connector were to import and export location and message data to and from the NLT system as well as into and out of the Nexsys™ system. This was completed and verified by the mine site personnel. The connector service automatically reconnected to the NLT Connector and Nexsys™ Server after a server machine restart.

The successful implementation of the Real-Time Mine Plan Viewer was proven through mine plan updates when a new mine plan became available and/or changes to the mine plan were detected through the system. In addition, the location of most recent tag reader that personnel and equipment came into range were shown on the Mine Plan and the Mine Plan speed and accuracy were acceptable to the end users.

The development of the Rules Engine was undertaken during the second half of the trial and, by the end of the 12 months period, mine site personnel testing the system witnessed and confirmed effective functioning of the standard rules, the Anomaly Detection rules configuration as part of the standard rule set and execution of omission of the rules depending on the hierarchical set up of the rule set. Furthermore, the Anomaly Detection (AD) algorithms were successfully trained using historical data from methane, carbon monoxide, carbon dioxide and oxygen sensors and the anomaly detection rules were configured as part of the standard rule set. In addition, the AD module could process new sensor readings at run time from the methane, carbon monoxide, carbon dioxide and oxygen sensors based on knowledge obtained from previous training and graphical display of AD inference and likelihood results.

**Nexsys™ trial - user interface**

The second goal of the Nexsys™ trial was to create and evaluate a specific user interface that would contain all the required information for different levels of personnel employment status and thus addressing their needs and responsibilities. The user interfaces were designed for the Control Room Operator and the Maintenance Superintendent, however, during the 12-month trial, the interface design and implementation were primarily tailored towards the requirements of the Control Room Operator (CRO).

The user specific interface is intended to minimise the amount of effort the user must expend to interact with the system i.e. provide input for the system, quickly interpret the output from the system, and learn how to perform these functions. Accordingly, a successful interface implementation was linked to specific users’ requirements determined through user interviews and agreed to with the mine management. The main criteria for the interface acceptance by the site personnel included:

• Gas Alarms sent to correct person via NLT system displaying;
• Description of alarm;
• Sensor that triggered alarm, including its location if available;
• Alarm acknowledgements sent to CRO via Nexsys™ UI and stored in alarm history;
• Non-acknowledged alarms could be acknowledged by CRO after verbal confirmation with appropriate personnel;
• Alarms acknowledged via NLT system and displayed on the CRO UI;
• Successful initiation, display, and acknowledgement of alarms using Nexsys™;
• Personnel and equipment tag reader location (NLT) displayed on mine plan;
• Ability to locate personnel and equipment using a search function;
- Gas sensor value displayed next to the gas sensor icon on the mine plan;
- Acknowledgement of TARPS messages by various personnel displayed on UI;
- TARPs available for display on the main screen when gas sensor trigger level bridged to alleviate need for manual searching of hard copies of TARPs (Figure 5).

![Image]

**Figure 5 - Nexsys™ TARPs display: window becomes available to CRO for Level 3 Alarm**

**CONCLUSIONS**

The Nexsys™ Real-Time Risk Management decision support system is able to improve mine safety through its continuous monitoring of the state of a mine, integrating critical mine data from various systems and sensors and notifying the appropriate personnel using a variety of decision support tools.

Through the system trial, Mining Logic Solutions (MLS) and CSIRO were able to demonstrate that the Nexsys™ Real Time Risk Management System could store data from multiple propriety databases and use that data to create four dimensional rules that took into account anomalies and false alarms. Information was shown on a specially designed user interface that enabled the user to locate all the information required to solve the alarm on one screen. The information provided to the operator included TARPs and the locations of the alarm, people and equipment within close range of the alarm. The successful completion of this trial has convinced both MLS and CSIRO that the product is now ready for commercial deployment. The Nexsys™ Real Time Risk Management System will be available to market from November 2010.

Due to the importance of effective risk management in mine safety and the lack of similar systems, Nexsys™ has a great potential to contribute to safety improvement in the international mining community. With its unique features, Nexsys™ is ideally suited for export to many other countries including India, China, the USA and Canada. In addition, the Nexsys™ application is not limited to the underground coal domain; future directions for Nexsys™ implementation include surface mines (coal and metalliferous), underground metalliferous mines, and non-mining domains wherever the assessment of risk and risk-based analysis is critical to the health and safety of the operations and personnel, such as emergency and rescue services.

**ACKNOWLEDGEMENTS**

Nexsys™ development would not have been possible without JCOAL support, which contributed substantial financial support, participated in the initial definition of the research goals and provided a demonstration site in Japan. JCOAL has also expressed willingness to contribute to the future propagation of the technology in coal producing countries.
ACARP is also acknowledged for recognising the importance of a real-time risk management system for underground coal mining in the early days of the project and providing CSIRO researchers with funding towards system development.

Special thank you is extended to the management and personnel at the Anglo Coal Grasstree Mine for their continuing support and faith in Nexsys™ capabilities.

REFERENCES


VENTILATION SURVEYS AND MODELLING - EXECUTION AND SUGGESTED OUTPUTS

J A Rowland

ABSTRACT: The need for a pressure quantity survey is usually brought about by a lack of knowledge of the actual value of various roadway, branch and infrastructure resistances around the mine. If resistances are not known or are suspected to be erroneous then it is difficult or impossible to accurately replicate the circuit performance, on a ventilation modelling software program.

The challenge for mine operators is to carefully decide what level of survey detail is required so that the appropriate adjustments can be made to an existing model or the appropriate data collected so a model can be built in the first instance.

The level of survey detail required is totally dependent on the end use of the data and the criticality of the accuracy of the dataset, to the safety and viability of the business. Mine technical personnel should think clearly about the report outputs required that will help them better understand their mines and better prepare them to manage their own ventilation circuits, on an ongoing basis.

Some methods of data collection and circuit analysis required for surveys up to a high order of accuracy are detailed.

Some of the issues which are important when formulating a strategy to carry out the appropriate ventilation survey for the required purpose are summarised.

Some details that should be furnished within any survey and some survey/report outputs and/or omissions, which are clearly not acceptable, are highlighted.

INTRODUCTION

Most mines in Australia, at least in the underground coal sector, maintain a validated model to assist with the process of minor to major ventilation circuit adjustments. They also routinely utilise validated ventilation models to track ventilation system performance and thus effectively plan and budget for major ventilation infrastructure, as the mine expands. The modelling program of choice in Australia is “Ventsim” software which is now available in a fully graphical 3D configuration as well as the widely utilised 2D version, the models of which are operable on the newer format. The wide availability of this software, coupled with changes to legislation in both NSW and Queensland in the last decade, has ensured that most site ventilation personnel are well equipped to utilise the technology.

Probably the most vexing issue for site personnel is just when the model actually needs a professional “tune up” and to what use will the rebuilt model be put. There is arguably little or no value in carrying out a detailed mine pressure/quantity survey if the results are not going to be transposed onto a model interface and, more importantly, the model must be religiously and periodically updated, pursuant to such works.

Considerable thought should be put into the survey scope to obtain the desired result dependent on site needs. As such the scope of the work is an extremely important component.

There is a diverse range of survey possibilities that may be considered depending on the use to which the data will be put.

The overall challenge for management is to decide the appropriate level of survey detail that is required to satisfy current and future site needs. It is possible that a working model could be built remotely from the site using raw experience, site supplied survey data and empirical values to build a rudimentary model that actually works. At the other end of the spectrum is the “full PQ survey” which may mean

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determining the resistance of all open areas of the mine from the bathroom to the fan blades and closing survey loops “a la” survey level run fashion. Obviously the time and dollar cost of two such contrasting assessments could vary ten to twentyfold. Management really need to consider the work that the ventilation model needs to be put to before asking the consultant to supply a quote for a “full PQ survey”.

The challenge for mine operators and ventilation personnel is to decide when such a high level of detail is required and what options are available if a less accurate dataset will suffice, for the job at hand. All survey and model outputs should be carefully scoped according to the particular requirements of the mine at that time.

SURVEY SCOPE

It is routinely the case that the client is not familiar enough with the model assembly process to specify, or scope, what is exactly required from the ventilation survey process. Whilst they may be familiar with what the final outcome will be, they are sometimes unclear on the actual steps required to reach that objective.

It is prudent for the client to discuss with the ventilation surveyor: 1) what they hope to get out of the final assembled model; and 2) how and for what they plan to use it. The survey process may be dependent on the layout, age and gassiness of the mine along with numerous interrelated factors. Considerable thought and discussion should surround the intended purpose for which the model will be used. An appropriate survey strategy can then be devised accordingly. Often clients will request a full closed Pressure/Quantity survey when considerably less may be required. The concept of a fully closed survey loop of all mine roadways for all jobs is a simple way to oversupply a service that is often not needed. Ironically many fully closed loop surveys result in the provision of a basic stick figure schematic model. The value of such a model should be closely scrutinized by mine site staff.

Summarily the nature and complexity of the survey, and subsequent assembled model must reflect the needs of the client at all times but assisted by the experience of the ventilation surveyor to achieve a satisfactory outcome for all concerned.

SURVEY PLANNING

The initial survey preparation is of the utmost importance. Unless the surveyor intimately understands the circuit design, then the allocation of the appropriate station locations is virtually impossible. It is of prime importance to study the mine-plan sufficiently to understand all circuit flows, and considerable discussion and questioning of the client and/or his representatives is required unless the ventilation surveyor intimately knows the mine.

Decision on method

After a thorough knowledge of the ventilation circuit is gleaned, a decision is made on the intended survey method. Hopefully this should be known before the reconnaissance survey is carried out but often the findings during the reconnaissance survey alter the planned survey method. Direct survey methods rather than indirect are generally favoured as the chance of unacceptable errors using the indirect method is higher for whole mine surveys and results are often more difficult to reconcile. Having said that it is prudent to utilise barometers for specific or pointed resistance back calculations especially in areas of limited access and they are ideal for lengthy splits with very low pressure drops per unit length which may make the hose drag method unsuitable. Summarily though, a combination of direct and indirect pressure determinations may be recorded depending on the particular situation, to achieve an acceptable result.

On occasions, the lack of appreciable roadway pressure drops may mean that the model is assembled using only empirical methods. That is to say the model roadways are assigned measured parameters of dimension, and k factors can be allocated based on experience. This can result in models of adequate accuracy and overall flows and pressure drops can be refined using differential pressures only. Whilst this is acceptable, there is no real way to guarantee the actual pressure drops in either the intakes or returns, only in the combined sets of intakes and returns. In such cases circuit and roadway knowledge is of paramount importance if individual branch model resistances are to replicate real underground values. As such, considerable exploratory work and discussion with experienced mine site operational
staff is required, and any shortcomings of the development of an empirical model should be properly explained.

Having due regard to the above comments, the final survey method may be ultimately determined by a number of factors including the ultimate planned use of the model in conjunction with client preferences, the speed of assembly, the final accuracy required, available equipment, and budgetary constraints.

Many circuits can be delineated with measured flows and a combination of frictional and differential pressures and the number of determinations required will depend on pressure drops per unit length, roadway conditions, access issues and intake and return conditions.

**Preparing underground plans**

Up to date underground plans are critical to a ventilation survey. They need to include details of existing driveages and those intended to be driven along with cross-cut numbers, panel names, air flow direction, location and description of various appliances along with all natural and artificial resistances or regulation. Obviously the plans need to be printed and prepared for use during the underground survey and should be both regularly updated and closely guarded throughout the project.

**Identifying falls and other obstructions**

Considerable discussion will be required with the appropriate staff to assess the impact or level of underground obstructions such as falls, water lodgements, stowage emplacements and the like. This information is pivotal in the selection of the process used to delineate roadway resistances.

**Identification of probable station locations**

Once the circuit is appropriately understood and a generic survey plan has been devised, the surveyor needs to plan where quantity survey stations and pressure station locations should be established. Whilst there is some flexibility in the establishment of these, they are primarily dictated by the circuit design and contribution of flows from and into adjacent air splits.

The overarching requirement of the number of stations required and their locations is that all circuit flows, whether intentional flows or uncontrolled leakage, can be identified and delineated.

**Underground reconnaissance survey**

A reconnaissance survey is of the utmost primary importance after the surveyor has chosen the selected survey method. The survey enables the following functions to be completed:

- Stations must be clearly and legibly marked up at the appropriate locations giving due regard to roadway velocity profiles, roadway uniformity, proximity to obstructions and the like.
- Quantity station areas must be accurately measured. This is a very time consuming process and should not normally be done during the survey. The survey should be carried out in the shortest and most efficient way possible and to measure and mark up stations during the survey is not normally a viable option. Bridge access across conveyors and door access into returns must also be identified which may impact on station locations.
- During reconnaissance attention must be paid to the collection of pressures as tubes may need to be run across regulators and other obstructions to collect differential and/or frictional pressure data. The reconnaissance survey is often the hardest work involving the longest days but is arguably the most valuable part of the process, without which, satisfactory results are highly unlikely.

**CIRCUIT LAYOUT/PECULIARITIES ON DAY OF SURVEY**

It is critical to know the exact circuit layout at the time of the actual pressure/quantity survey and face area peculiarities, are of paramount importance in this regard. For example, if an auxiliary fan is tightly bratticed up as in a weekend ventilation arrangement to push air through the exhaust ducting, then it is very important to be aware of such a scenario. This would then be resolved by measuring the pressure
and flow and the resultant back calculated resistance could then be inserted into the model as a fixed resistance. Circuit knowledge is always a prime consideration.

It is obviously critical to ensure that the mine site does not alter circuit parameters or design during the survey. They must appreciate the need to ensure it is left in a stable state throughout the survey duration.

Survey execution

The following actions should be taken to ensure a successful survey.

- Team discussion should focus on the planned method and goals to achieve during the shift and the intended work schedule adjusted according to the groups input. Ideally the route of travel should gather the intended data in the shortest possible time frame.
- Assemble the appropriate tools/survey aids including two way radios, anemometers, manometer, barometers, hose, watch, tape, sling psychrometers, paint, chalk, pro-formas and plans.
- Remember key items such as ample water and food because days often end up much longer than expected.
- Any new stations must be clearly and legibly marked up at the appropriate locations giving due regard to roadway velocity profiles, roadway uniformity, proximity to obstructions and the like. They should also be added to the mine plans.
- When frictional resistances need to be determined it should be done sequentially by measuring the branch pressure drops and determining the average branch flow at that time for resistance back-calculation. Obviously appropriate strategies for the determination of densities and the elimination of differential velocity pressures at each survey station must be exercised during this process. The author believes that many specific branch resistances may be determined in isolation to the survey as long as they are static splits and not likely to change. Examples of such branches would include airflows across overcasts with indeterminable shock losses, longwall faces, heavily timbered roadways and areas of fallen roof or stowage. The resistance of these can then be deduced at any suitable time.
- When the required frictional branch resistances are determined along with the corresponding differential pressures into adjacent splits such information should allow the skeletal framework of the ventilation model to be later assembled, ensuring roadway resistances in the model closely replicate those determined during the survey.
- Pursuant to the frictional pressure/quantity survey it is wise to carry out a full mine wide quantity/differential pressure survey in the shortest possible time measuring data at all selected flow and differential pressure stations. This data gathered then reflects a concise and precise “point in time” data set that can be used to tune the model after the aforementioned specific attributes and predetermined resistance sets are included in the initial model assembly process. Speed is of the essence when gathering this “point in time” data and the observations must be sufficient to delineate all circuit flows and leakage paths.
- Any ancillary data that can be gathered during the survey is also collected. If the mine is gassy, it is a simple process to collect gas concentrations at every station and a complete mine gas balance can be determined. Such information can then be included in say a bar chart format in the final report.

ONGOING STATION USE

The pressure and quantity stations used during the survey should then be utilised by mine site staff to allow them to update the model on a regular and ideally monthly basis. Most model assemblies should result in a pro-forma of sorts for ongoing routine site validation. The biggest mistake most consultants make is data overload. Let Kirchhoff’s laws work for you and use the laws to establish the minimum number of stations to supply the maximum data output for both the pressure/quantity survey and ongoing site staff validation.
FAN PERFORMANCE

Fan performance during the survey must be measured and the results compared against expected operating points according to available supplied curves. If no curves are available, it is prudent to identify at least a couple of measured operating points on the curve. This can be done pre or post survey by loading up the mine resistance or by reducing the mine resistance by opening pit bottom doors and determining corresponding PQ operating points.

It is important not to be overly concerned about the expected operating point at the fan itself. Some adjustment of shaft bend resistance may need to be factored in to get pit bottom pressures the same on the model as they are in the mine. More often than not, the supplied manufacturers curve as stated will just not be suitable.

Often, for a myriad or reasons, the measured operating points of fan pressure and total fan flow lie well off the manufacturer’s curve. The most important thing however is to get the pit bottom ventilation pressures the same in the model as those measured in the pit. The balance of pressure losses in the shaft and shaft top appurtenances is of little consequence unless discrete changes or improvements are planned to them, which is unlikely.

Consider a situation where there is 2000 Pa available across intake to return at the bottom of the shaft and there is 250 m³/s of available airflow. If the model can be tweaked to emulate this, both at that point, and all pressure and quantity stations inbye, then it can be reliably utilised. The fan curve issue can be sorted out at the first available opportunity.

RESULTS TABULATION

After all readings are taken, the data then needs to be reconciled. The process may highlight errors or omissions of data that need to be further explored before winding up the survey. To achieve this all data should be reviewed on each day of survey. If data is found to be questionable, it should be checked as soon as possible, preferably the next day. It is a little late to go and collect data a week later if there were important omissions and the circuit has since changed markedly.

Results are best tabulated on a spreadsheet and a complete analysis of the circuit can then be carried out and discrete leakage magnitudes assessed. Figure 1 shows the raw results data and Figures 2 and 3 shows the distribution of intentional and unintentional flow respectively.

![Figure 1 - Raw survey data](image1.png)

GRAPHICAL DATA REPRESENTATION

The final results are best transposed onto a plan type schematic of the circuit. This is paramount to the model update process. The schematic allows an iterative update of the model to be done by cross-referencing measured results against the changing model values during tuning and validation.
The use of a schematic also allows site personnel to better understand their own circuit peculiarities. Further to this it can be used as an ongoing statutory report inclusion and serves as an ideal circuit training tool for mine officials and the workforce in general. Microsoft Excel is perfect for this task.

Figure 4 shows an example of a circuit schematic generated in Excel which ideally should be fabricated during the survey and report process.

Figure 2 - Intentional flow distribution

Figure 3 - Unintentional flow distribution

Figure 4 - Vent circuit schematic

**MODEL ASSEMBLY AND TUNING**

**Assessment of generic roadway attributes**

There is nothing more ridiculous than assembling a ventilation model that has grossly erroneous roadway attributes and for no logical reason. Models have been previously sighted with roadways with designated cross-sectional areas 10 times that of actually measured underground only to have unrealistically resistive k factors assigned to the branches to achieve the appropriate pressure drops. Whilst such a strategy does nothing in relation to confidence of the finished product the biggest risk with
such a plan is ongoing validity when others attempt to update the model. If the true roadway area is used and realistic k factors and shock losses are designated the longevity of the model will be maximized. Considerable effort should be put into determining the approximate cross sectional areas of various roadways or branches of roadways and this is of far greater importance if a full mine wide pressure/quantity survey is not envisaged, as empirical resistances of various branches need to be determined. The colliery mine surveyor is usually a key resource in this case and can also assist with accurate values of shaft depths and diameters which are obviously required for the model assembly process.

**Physical model assembly tasks**

Centerlines from a dxf drawing file need to be imported into the modeling software assuming that all or most roadways intend to be utilised in the finished model.

There is no reason why virtually all open mine roadways cannot be included in the model. Having said that it may be a benefit to make irrelevant roadways schematic by design if they have no real bearing on the model. (Examples of this may be an open inaccessible goaf area that is flowing air but cannot be accessed to assess the roadway status or flow distribution. A further example may include multiple flood intake or flood return roadway sets that may be simplified in discrete locations).

It is important however to at least make the model circuit look like the mine circuit.

There is arguably little value in turning a 2000 branch mine circuit into a 50 branch ventilation model. The main reason for not utilising all or most circuit branches in bygone days was the inability to be able to properly view the network model. This was particularly relevant in metal mines where plan and elevation views were almost always confusing. Thankfully modern software such as the “Ventsim Visual” 3D screenshot shown in Figure 5 makes such viewing simple and practical from any conceivable angle. If the consultant has the ability to fabricate the model the technology is now available to easily view and understand the circuit.

**Figure 5 - 3D view in “Ventsim Visual”**

What really is the point of turning the mineplan shown in Figure 6 below into a stick figure style ventilation model such as that depicted in Figure 7?

The minimalistic skeletal model depicted in Figure 7 would add very little value when trying to add on the five year mine plan as shown. Worse still, how would you then model what happens when discreet changes are made such as modelling the effect of segregating the total conveyor system, or trying to model emergency escape protocols according to contamination velocity?
To further this point and considering a more crucial scenario how could anyone go about sequentially recovering the mine in practice if it was sealed due to say a fire and then needed to be recovered?

There is no conceivable reason why the model supplied pursuant to the survey cannot contain most of the branches as detailed on the mine plan shown in Figure 6. An example of such a model which would prove far more useful to mine staff is shown in Figure 8.

It is also important to ensure both the model and the mine plan utilise the same alignment to North and the same coordinate origin. In this way future drivages can easily be imported as the mine expands.

**Figure 6 - AutoCAD mine plan of driven and future working**

**Figure 7 - Skeleton model of arguable value**

**Input of resistance determinations and final tuning**

The results from the “hose drag data” or “barometric survey” are then interpolated to determine specific split resistances. The corresponding roadway sets in the model are assigned the appropriate k factors.
and/or cross-sectional areas to achieve the desired resistance whilst maintaining believable and substantiative roadway attributes.

Assigning all known attributes is then carried out. All known appliances should be inserted with approximate resistance values depending on condition of them noted underground.

Initial adjustment of appliance resistances is carried out according to leakage rates and magnitudes measured across the site during the survey.

Continued model refinement is then carried out to enable the assembled model to properly mimic the measured underground data.

**REPORT ASSEMBLY AND STANDARD**

The report should be assembled in an easy to read manner so that mine-site staff can easily understand it. There is really no value in overcomplicating the survey. Whilst a discussion on the limitations of the survey may be valid there is no value padding out the report with theoretical issues such as generic reasons for survey errors, advantages of indirect over direct methods, anemometer yaw and related sources of error. The client needs functional practical solutions to the physical problems found at the site.

The report should ideally contain:

- Introduction
- Agreed scope according to pre survey meeting.
- Executive summary listing key issues.
- Interim practical solutions to circuit problems/ restrictions identified.
- Raw data results of all measured or determined values.
- Schematic layout of the circuit detailing all flow and pressure results.
- Model displays of both flow and pressure to demonstrate validity against the assembled schematic.
- Graphical comparison of measured versus modelled values in the case of a new model. In the case of a rebuild a comparison of the both the client supplied model and the refurbished model should be made to the measured data. These would take the form of:
  - Flow validity of a model supplied by the site when compared to the underground data measured. (Similar to that shown if Figure 9).
  - A demonstration of improved model flow correlation after the survey and model tuning process is completed. (Similar to that shown if Figure 10).
  - A similar approach should be utilised to demonstrate pressure validation also.
- Sector chart of all intentional and mine leakage flows
- Gas make/temperature/dust or other contaminant contributions if applicable.
- Long term hit-list of improvement initiatives to improve circuit performance.

The last item (k) is probably the least done by ventilation consultants but is the sort of information the client needs and appreciates. A snapshot of the circuit in time without any immediate improvement suggestions is of little value to the client in practical terms, especially if the circuit is severely constrained at the time of survey.

**FORWARD PLANNING**

If centrelines of intended driveages have been made available then future capacity simulations can be run as and if required. It is fair to assume that the previous performance of the circuit and appliances as determined during the survey can be replicated in future. Having said that though, if appliances have
been measured to perform poorly during the survey it is reasonable to elevate performance in line with industry standards as long as that point is made to the client. Any number of ventilation iterations can be modelled as required.

Figure 8 - All roadway ventilation model

Figure 9 - Supplied model validation

Figure 10 - Rebuilt model validation

ASSISTING MINE-SITE PERSONNEL TO OWN THE PROCESS

The mine should be encouraged to then take the “appropriate readings” on a regular (ideally monthly) basis to properly maintain the new tool whilst minimizing assistance from outside consultants.

The “appropriate readings” are those required to allow an accurate ongoing delineation of the circuit such that the consultant could remotely tune the model from offsite if required.

Obviously the mine should not have to rely on outsiders to maintain the model *ad infinitum* and thus management should strongly consider resourcing the ventilation department such that they can effectively tune and maintain their own model into the future. Whilst this may be somewhat problematic in the early stages, numerous mine site models have been adequately maintained by relative journeyman, as long as some personalized training is provided. Like any skill the more the site staff are exposed to the software the more capable they become at tuning the model. What is often a sketchy update in the early days quickly transforms into confident routine tuning of the ventilation model by site staff.
INAPPROPRIATE VENTILATION SURVEY/REPORTING PRACTICES

The following practices are simply not appropriate:

- Non disclosure of all measured survey data.
- Model validation of only one unknown. There is little evidence demonstrated of model validity if only the flows or pressures from the model are compared against the measured survey data. If there is insufficient validity in either the flows or the pressures then the model resistances are clearly incorrect. If both are not disclosed then this should prompt some questioning.
- An ad hoc approach to an over simplistic model which bears little resemblance to the mine layout.
- Not utilising mine site survey department dxf co-ordinates in the model assembly process. (This is important for future additions of longer term mine plans.)
- Apportioning inappropriate time and resources into measuring and detailing microscopic and thus irrelevant values of natural ventilation pressure. Such work should be only included according to the sensitivity of that phenomenon.
- Using unrealistic roadway $k$ factors or dimensions to adjust branch resistances.
- Not getting sufficient data to properly assess all intentional and unintentional flow paths and magnitudes.
- Furnishing the client with a report filled with scholastic type issues such as generic instrument or method errors, advantages and disadvantages of chosen processes or research findings, and the like.
- Collecting insufficient data to allow the suggestion of simple common sense solutions to the ventilation problems that are immediately at hand.

CONCLUSIONS

The execution of a ventilation survey and the fabrication and/or maintenance of a properly tuned ventilation model are intricate components of modern underground safety management systems. Such systems in Australia revolve around the routine use of risk based logic. Obviously the assessment of risk with regard to ventilation changes and ventilation circuit designs utilise ventilation modelling software as a key tool to assist in this structured process.

As such the measurement of the underground circuit and its replication on a working model is extremely important work. To organise such important work without carefully scoping the job to suit the particular needs of the operation is less than adequate.

There is no set rule on either the level of detail or which survey method should be adopted. Such parameters need to be decided by management dependent totally on site needs at that time.

Considerable expense is involved in the execution of such work and it is vitally important that the final outputs complement the mine requirements and ultimately improve the safety of the operation for all persons employed therein. It really is a responsibility of management to ensure the scope of works complements the needs of the mine at any particular time and the ventilation survey is done to a level of detail that complements those needs. Considerable discussion and scoping is required before the call goes out for a “full PQ survey”.

REFERENCES

THE EFFICIENCY STUDY OF THE PUSH-PULL VENTILATION SYSTEM IN UNDERGROUND MINE

Xichen Zhang, Yutao Zhang and Jerry C Tien

ABSTRACT: Auxiliary ventilation refers to the systems that are used to supply fresh air to the working faces in dead end including the use of a push-pull arrangement. There are many advantages of using such a system when compared to other methods. It has been shown that for headings longer than 30 m, auxiliary fans are the only practicable means of delivering the required air quantities. Besides air quantities, air quality is another critical issue for evaluating the ventilation efficiency in underground mines. Forcing and exhausting ducts used in the push-pull system are closely associated with the ventilation efficiency. This paper focuses on the efficiency evaluation for a push-pull ventilation system by using two methods, the dead zone and the mean age of air. By using the CFD technology, the air velocity and air quality are calculated and compared in four different cases. The results of evaluation will be identical by using these two methods. It is concluded that in the push-pull ventilation system, the position of the forcing duct plays a major role on the ventilation efficiency. Also, when the forcing duct position is determined, there must be a particular position of the exhausting duct to provide best ventilation efficiency.

INTRODUCTION

Mine ventilation is critical in all underground mining operations. It provides the miners with fresh air at proper temperature and humidity, and more importantly it removes pollutants out of the mine. However, due to increasing resistance and leakage from stoppings, the main ventilation system may not be capable of ventilating remote or more localized areas underground and auxiliary ventilation is needed to assist the main ventilation system.

An auxiliary ventilation system can usually be classified into three basic types, line brattice, fan and duct systems, and "ductless" air movers. Previous researches have shown that the fan and duct systems are the only practicable means of producing the required airflows for headings with length greater than 30 m (McPherson, 1992). A fan-duct combination can be forcing, exhausting or a combination system of them. Considering that the forcing system will potentially add pollutants to the airstream at the working face, and the exhausting system can cause uncontrolled recirculation, a more common approach is to have a push-pull system, as shown in Figure 1. By comparing the advantages of forcing and exhaust duct, this push-pull ventilation system, especially the forcing system with exhaust overlap, is in practice adopted widely during the process of mechanised advancing because of the following advantages.

Figure 1 - Overlap systems of auxiliary ventilation (McPherson, 1992)

Firstly, the entire cost of the system for (a) might be cheaper, because the cheaper flexible duct can be used for the long forcing duct, which is also easier to transport and enable leakages to be detected more readily. Secondly, a dust filter or a cooling system can be added in the exhaust duct for (a) in order to
purify the air and keep a comfortable environment for the miners. Finally, in some emergent cases, the auxiliary fans may stop running, the layout like (a) can make sure that there is still a little fresh air going into the working face (Wang, 2007).

In the push-pull auxiliary ventilation system where the forcing system with exhaust overlap is used, the fresh air is delivered to the working place through the forcing duct, forced to flow through the working area where mixing with the pollutant air takes place, and then exhausted out of the blind heading through the exhaust duct after filtering out the dust by the depurative device at the end of the exhausting duct. Many factors, such as the pollutant air distribution in the working place, selection of fans and ducts, forcing and exhausting device capacities, and duct layout, will all affect the ventilation efficiency. This paper focuses only on the effect of duct layout.

THE EFFECT OF PUSH-PULL SYSTEM LAYOUT

Problem description

In the push-pull ventilation system, airflow behavior through blowing out from a forcing duct into an exhausting duct is influenced heavily by the surroundings. Meanwhile, for the two push-pull ventilation systems in Figure 1, the recirculation or zone with low velocity may exist depending on the ducts layout, as shown in Figure 2 (Niu, et al., 2006; Wang, 2008). Lf and Le represent the distances between the working face and the forcing duct outlet and the exhausting duct inlet, respectively. So, to avoid local recirculation, the position of the forcing duct and exhausting duct must be carefully determined. The effective range (Lr in Figure 2), which is defined as the distance between the duct outlet and the place where the velocity goes down to the required value by the regulation, is critical in placing a vent duct. To meet the minimum airflow requirement on the working face, the distance between the duct inlet or outlet and the working face should be less than the effective range.

![Figure 2 - The air flow distribution in the push-pull ventilation system](Based on Niu, et al., 2006; Wang, 2008)

For a forcing duct, the effective range of a semi-confined jet can be determined by using the formula below (Bai, 2005):

\[
\bar{x} = \frac{a \cdot x}{\sqrt{S}}
\]

\[
\frac{\bar{v}}{v_0} \cdot \frac{\sqrt{S}}{d_0} = 0.177 \cdot (10\bar{x}) \cdot e^{(10.7x - 37x^2)}
\]

Where: 
- \(\bar{x}\) is the non-dimensional effective range;
- \(x\) is the effective range, in m;
- \(a\) is the turbulent coefficient;
- \(\bar{v}\) is the average velocity of backward flow, in m/s;
- \(v_0\) is the jet velocity in outlet, in m/s;
- \(S\) is the cross-section area of the roadway, in m²;
- \(d_0\) is the duct diameter, in m.

There is no comparable equation available to determine the effective range for an exhausting duct in a confined space, because the situations are more complicated. Neither theoretical nor experimental
equations have been successfully used to determine the effective range. The only approach would be through computer simulation using such package as FLUENT to optimize the exhausting duct location.

Geometric model

Considering a rectangular entry with 2.5 m in height and 3.5 m in width using a push-pull auxiliary ventilation system with 0.6 m diameter forcing duct and a 0.45 m diameter exhausting duct. Both ducts are hung parallel to each rib at half height between the roof and floor. The exhausting duct is 30m in length. The geometry model is built as shown in Figure 3. To avoid the influence of the far end on the flow field, the model length L is set to 100 m. Lo represents the overlap areas.

Numerical model

The standard $k - \varepsilon$ model is used to calculate the turbulent flow distribution. The air age, which is defined as the time since gaseous elements enter into a domain through inlet, is introduced as an important index of evaluating the confined space environment (Sandberg, 1981). Since the transport equation for calculating air age is not an available model in FLUENT, the equation is incorporated in the CFD simulation by using user defined functions (UDF) which are developed into a separate code based on the platform of FLUENT and then are compiled into executable functions in the solver. An energy equation is not considered because heat transfer is ignored in this simulation. The air density is also assumed to be constant. The transport equation of air age has the same form with the standard $k - \varepsilon$ model shown below:

$$\frac{\partial}{\partial x_j} \left( \rho u_j \phi - \Gamma_\phi \frac{\partial \phi}{\partial x_j} \right) = S_\phi$$  \hspace{1cm} (3)

Where $\rho$ and $u$ are the density and the velocity of the air, respectively. Universal variable $\phi$, the diffusivity coefficient $\Gamma_\phi$, and the source term $S_\phi$ are defined in Table 1.

<table>
<thead>
<tr>
<th>Equation Type</th>
<th>$\phi$</th>
<th>$\Gamma_\phi$</th>
<th>$S_\phi$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass Continuity</td>
<td>1</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>X-Momentum</td>
<td>$u$</td>
<td>$\mu_u$</td>
<td>$\rho u_j \frac{\partial}{\partial x_j} \phi$</td>
</tr>
<tr>
<td>Y-Momentum</td>
<td>$v$</td>
<td>$\mu_v$</td>
<td>$\rho v_j \frac{\partial}{\partial x_j} \phi$</td>
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<tr>
<td>Z-Momentum</td>
<td>$w$</td>
<td>$\mu_w$</td>
<td>$\rho w_j \frac{\partial}{\partial x_j} \phi$</td>
</tr>
<tr>
<td>Kinetic Energy</td>
<td>$k$</td>
<td>$\mu_k$</td>
<td>$\rho u_j^2 \frac{\partial}{\partial x_j} \phi$</td>
</tr>
<tr>
<td>Dissipation Rate</td>
<td>$\varepsilon$</td>
<td>$\mu_\varepsilon$</td>
<td>$\rho u_j \phi \frac{\partial}{\partial x_j} \phi$ - $\rho \phi^2$</td>
</tr>
<tr>
<td>Air Age</td>
<td>$\gamma$</td>
<td>$\mu_\gamma$</td>
<td>$\rho$</td>
</tr>
</tbody>
</table>
In these equations, $u$, $v$, and $w$ are the velocities in $x$, $y$, and $z$ direction; $k$ and $\varepsilon$ are the turbulence kinetic energy and its rate of dissipation, respectively; $\tau_p$ is the average air age; $P$ is the pressure; $\rho$ is the density; $G_k$ represents the generation of turbulence kinetic energy due to the mean velocity gradients; $g$ is the gravity; $\mu_{eff}$ is the effective viscosity, equal to the laminar viscosity $\mu$ plus turbulent viscosity $\mu_t$, which is calculated with $\mu_t = \frac{C_\mu \rho k^2}{\varepsilon}$; $C_1$, $C_2$, $C_\mu$, $\sigma_k$, $\sigma_\varepsilon$ and $\sigma_t$ are constants in the model.

**Boundary conditions**

Fresh air is supplied at the entrance to the forcing duct with an average velocity of 12 m/s (or an air quantity of 3.4 $m^3$/s), delivered to the working face with a temperature of 300 K (27°C). The pollutant air then enters the exhausting duct at a velocity 12 m/s and is delivered out of the exhausting duct. The pressure in the airway outlet is set equal to that of the atmosphere. The boundary condition for the air age equation is set to zero at the forcing duct inlet. No slip velocity is present along the walls and ducts, which means flow velocities are set to zero.

**Results and discussion**

To simplify the simulation, the distance $L_f$ (Figure 3) between the forcing duct outlet and the face is constant at 10 m based on the effective range equations (1) and (2). Four different cases, in which the distance $L_e$ between the exhausting duct inlet and the face varies from 2 m to 5 m, 8 m, and 12 m, are designed and calculated for comparison. The simulation results are summarized and discussed below.

**The effect of ducts layout on dead zone**

To guarantee a healthy and comfortable environment during operation, mining regulations always set a minimum air velocity in the working place (Wang, 2008; Anon, 2001; Parra, et al., 2006); air velocities in the face area are different but varied little. In this paper, the minimum velocity value is set at 0.3 m/s. This value is used to evaluate the ventilation efficiency by using the dead zone method. Dead zones are defined as those regions where velocity is under a minimum value. As mining operations are performed in the vicinity of the working place, this region shows the greatest pollutants’ concentration. So, to make sure the environment is suitable for mine workers, air renewal should be ensured at this area. By assessing the dead zones’ shape and distribution in this area, it is possible to evaluate air quality of a ventilation system.

![Figure 4 - The dead zones percentages in four cases](image-url)

Figure 4 shows the dead zones percentages on the cut-planes (normal to X-direction in the model) which are parallel to the working face. In the simulation, the distance between the forcing duct outlet and the working face is constant (i.e. $L_f = 15$ m). While the distance between the exhaust duct outlet and the working face is variable (i.e. $L_e = 2, 5, 8, 12$ m). The dead zone percentage is defined as the dead zone area divided by the area of the whole section. In all four cases, percentage value follows the same pattern. In the area close to the working face, the percentages are lower than 10% until about 14 m from face, then a sudden increase to the peak before it starts to drop. This sharp increase of the low velocity area is a result of a recirculation region formed in the area between plane 14 m and 21 m. The percentage of dead zone increasing from 10% to 60% means a sharp decrease in ventilation efficiency.
Close examination of the data shows that the recirculation region is formed in the overlap area. The range of the region is determined by the duct layout. The forcing duct outlet determines where the recirculation starts, while the exhausting duct inlet determines where the recirculation ends. The recirculation zone in the overlap areas has the worst ventilation in the entire simulation area.

**The effect of duct layout on air age**

Sometimes, ventilation velocity alone does not tell the whole story since air velocities can be over the required value by the law because of the recirculation, quality of air may not meet statutory requirements. Using “air age” may be appropriate to complement system evaluation. Lots of researches on studying local mean age of air have been done previously (Roos, 1999; Bartak, *et al.*, 2001; Karimipanah and Awbi, 2002). The governing equation of the air age follows the characteristics of the transport equation (Sandberg, 1981). At the flow inlets the air age starts with zero, which is used as a reference or starting point. It increases along the streamlines. If there is recirculation or zone with very low velocity in airflow, the air age will be higher in that region compared to areas outside of the recirculation zone.

Figure 5 shows mean air age on various cross-sections in the entry with the horizontal axis starting at the working face. Generally, as the distance moves away from the working face, the air age tends to increase. The air age increases steadily from the very beginning and faster at 10 m, starting to level off after reaching its peak. Results show that the recirculation region is formed in the overlap area with its range determined by the relative position of both the forcing and exhausting ducts. The forcing duct outlet determines where the recirculation starts, while the exhausting duct inlet determines where the recirculation ends. In addition, the recirculation region in the overlap area has the worst ventilation where the air velocity is low and the polluted air will stay in this region for a long time.

![Figure 5 - The mean air age in four cases](image)

Among these four cases, the air age of case 5 m is shortest at every cross-section. Between plane 10m and 34 m, the air age in this region increases from 105 s to its local maximum value of 166 s. Right after its peak, it decreases and then continues to go up at cross-section at 52 m. In this particular model, the 5 m case provides the best ventilation efficiency, followed by 2 m, 8 m, and 12 m.

**CONCLUSIONS**

The numerical models are used to analyse the ventilation efficiency under four different cases. The ventilation system is evaluated using two different criteria: dead zone analysis based on the velocity distribution and the distribution of local mean ages of air. Both approaches yield similar results.

For a long dead end using a push-pull ventilation system, the recirculation region can be formed in the overlap area, and make the ventilation conditions worse. In the area close to the working face, only the forcing system can determine where the recirculation region starts. The layout of the exhausting system has big influence on where the recirculation ends.

Another finding is that there is a particular distance between the exhausting duct inlet and the working face that can provide the best ventilation efficiency. In this paper, for four different cases, 2 m, 5 m, 8 m, and 12 m, the ventilation efficiencies are almost the same when considering the velocity. However, the 5 m case provides the best ventilation efficiency by considering the air quality.
REFERENCES


DUST MONITORING AND CONTROL EFFICIENCY MEASUREMENT IN LONGWALL MINING

Brian Plush¹, Ting Ren¹, Ken Cram² and Naj Aziz¹

ABSTRACT: Occupational hygiene has been an integral part of the mining industry for centuries; however its importance has grown with developments in mechanisation. While the focus in the past has quite correctly been on improving the controls on dust exposure, the future lies in identifying the efficiency of installed controls on operating longwalls, evaluating them through robust and quantitative sampling methods to ensure the most effective controls are in place to prevent occupational disease from occurring.

The current statutory testing regime identifies the exposure levels of personnel on an operating face, which gives a snapshot of the dust that these persons will be exposed to over the duration of a mining shift. Although this testing process clearly determines exposure levels, it does not give mine operators any indication of where dust is produced, how much dust is produced nor how efficient the installed controls are at mitigating produced dust.

This paper proposes a new testing methodology to determine installed control efficiency for both respirable and inhalable dust and reports the initial dust measurement results based on this methodology. The main objective of this sampling method is to identify dust loads at independent sources of dust generation on longwall faces and quantify the efficiency of installed controls for the mitigation of produced dust. The use of this new methodology will provide mine operators with a complete dust production signature of their operating longwall and allow the implementation of more efficient controls at independent sources of dust generation.

INTRODUCTION

Production from longwall mining in Australia has increased remarkably over the last several years. This increased productivity has meant that more dust is being produced and controlling respirable and inhalable dust continues to present the greatest ongoing challenge for coal mine operators. A recent report by the Director of mine safety operations branch of Industry and Investment NSW has found that there is an increasing level of inhalable dust being ingested by coal miners in New South Wales, potentially leading to long-term health problems (ILN, 2010). This increased exposure level for underground workers can be directly attributed to the increase in coal production and the continued development of medium and thick seam mines in Australia, which allow the installation of bigger and more productive longwall equipment.

Fugitive dust on longwalls has always been an issue of concern for production, safety and the health of workers in the underground coal mining industry both in Australia and globally. Longwall personnel can be exposed to harmful dust from multiple dust generation sources including, but not limited to; intake entry, belt entry, stageloader /crusher, shearer, shield advance and dust ingress from falling goaf or over pressurisation of the goaf. With the increase in production created from the advancement in longwall equipment, dust loads have also increased and this has resulted in an increase in exposure levels to personnel. Control processes in place for the mitigation of dust vary from mine to mine, with each individual mine having a dust mitigation setup that is only effective for that particular mine operation.

The industry has been using statutory dust measurements in underground coal mines conducted by both SIMTARS and Coal Services that rely on Australian Standards AS 2985 for respirable size dust particles, and AS 3640 for inhalable size dust particles. The majority of dust sampling to date has been with cyclone separation and collection of the sized particles for weighing, generally over the period of a full shift.

Although the above statutory method provides an accurate measurement for the total dust exposure for the period sampled, it does not always accurately reflect the source, quantity and timing of respirable dust generation.

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dust entering the longwall from different sources, which presents difficulties in determining the relative effectiveness of the different control technologies in use.

It is the aim of the safety and health management system at a mine to provide ways of suppressing excessive airborne dust and provide ways of ensuring workers are not exposed to any air stream with an average respirable dust concentration calculated over an eight hour period (as measured in accord with AS 2985) exceeding either 2.5 mg/m$^3$ of air for coal dust or 0.1 mg/m$^3$ of air for free silica (Coal Mines (Underground) Regulation 1999).

In a notice issued by the Chief Inspector of Coal Mines, dated 7th December 2004, in the New South Wales Government Gazette Number 200 (File No: C99/0691), the specified limits for respirable dust were quoted as follows:

“For the purpose of Clause 161 of the Coal Mines (Underground) Regulation 1999 definition of ‘specified limit’), the specified limit for quartz-containing dust is 0.12 mg of respirable quartz and the specified limit for respirable dust, other than quartz-containing dust, is 2.5 mg. These limits are with respect to the mass of respirable dust per cubic metre of air sampled and apply only to the underground parts of underground mines.”

The National Occupational Health and Safety Commission has adopted a limit of 0.1 mg/m$^3$ for quartz (Adopted National Exposure Standards For Atmospheric Contaminants in The Occupational Environment [NOHSC: 1003 (1995)]. This is with respect to an eight hour time weighted average (TWA) and has taken effect on 1st January 2005. There has also been a change to AS2985-2004 (AS2985-2004 Workplace atmospheres - Method for sampling and gravimetric determination of respirable dust), to bring it in line with the relevant ISO Standard and resulting in a higher flow rate during sampling. Studies undertaken through Coal Services Pty Ltd have confirmed that to cater for the change in flow rate the current limit of 0.15 mg is equivalent to a limit of 0.12 mg at the higher flow rate.

The long standing practice in underground coal mines has been to collect samples from crib room to crib room and for a minimum period of five hours. This is to avoid a number of practical difficulties in collecting samples during travel. Research undertaken indicates that crib room to crib room sampling of 0.12 mg, at the higher flow rate and with a travelling time conversion factor applied, corresponds to a limit of 0.1 mg for portal to portal sampling. The end result is that for underground mines the working limit for quartz is effectively unchanged and remains at a level where silicosis has not been observed in the coal mining workforce. The change in limit for respirable dust, other than quartz-containing dust, is to take into account the higher sampling flow rate now required by AS2985-2004.

**SOURCES OF DUST GENERATION**

Regardless of dust loads, which are directly proportional to tonnages produced, longwall dust generation at each independent source, produces relatively the same percentage of dust as a proportion of total face dust in each operating longwall.

Research from NIOSH (Organisciak, *et al.*, 1986) indicates that there are primarily six individual dust generating sources on an average longwall, not only in Australia, but by extension, all operating longwalls globally. Figure 1 shows the location of each of these independent sources of dust generation. Figure 2 shows a chart of total face dust as a percentage generated from independent sources.

- **Shearer cutting and cleaning** – is the largest dust source on a longwall face. A significant portion of dust occurs in the crushing zone around the tip of the pick. In general, the leading drum cuts the full drum height and generates the majority of the dust, while the trailing drum produces less dust due to the lower amount of coal being cut.

- **Roof support/chock movements** – As chocks lowered and advanced, crushed coal and/or rock falls from the top of the chock canopy directly into the face ventilation airflow. Most of this dust becomes airborne, and quickly disperses into the walkway.

- **Stage loader/crusher** – This is an area of high dust make, where the dust can severely contaminate intake ventilation and expose face operators to high dust exposures.
• **Face spalling/AFC dust** – Dust generated due to face spalling ahead of the shearer is a major problem in thick seam longwall faces. Dust can also be lifted up from the AFC by ventilation, particularly when the direction of coal transport is against the direction of the airflow.

• **Intake contaminations** – Dust can be generated at all the conveyor transfer points along the intake airways. The movement of any equipment outbye can also cause significant quantities of dust to be raised into the atmosphere.

• **Goaf falls** – Dust can be generated due to roof caving behind the chocks and sudden goaf falls. A significant proportion of this goaf dust can be pushed onto the face as the leaked airflow returns to the face along the face support line.

![Figure 1 - Sources of dust generation on longwalls](image)

![Figure 2 - Total face dust as a percentage generated from independent sources](image)

**EXISTING DUST MONITORING PRACTICES**

AS2985 and AS3640 clearly define the process used to determine personal exposure levels in coal mines. The same equipment will be used to collect dust load at each individual source of dust generation on a longwall to ensure uniformity of collected data, reliability of data analysis and approved for use in underground coal mines.

Section 6.1 of AS2985 - Workplace atmospheres - method for sampling and gravimetric determination of respirable dust states the essential features of a sampling system consisting of a filter (on which the
sample is collected) and a pump for drawing the air through the filter. The filter shall be secured in a holder that prevents air from leaking around the edge of the filter. The filter shall be preceded by a size-selective sampler.

According to Section 6.1 of AS3640 - Workplace atmospheres - method for sampling and gravimetric determination of inhalable dust the essential features of a sampling system are an inhalable dust sampling device (containing a filter on which the sample is collected) and a pump for drawing the air through the device. The filter shall be secured in the device in such a manner that it prevents air from leaking around the edge of the filter.

Table 1 provides a summary of respirable dust samples results in NSW mines for the 20 year period 1984 – 2004.

Table 1 - Respirable dust results (including re- samples) 1984 - 2004

<table>
<thead>
<tr>
<th>Mining Method</th>
<th>No. Personal Samples</th>
<th>Number &gt; 3mg/m³</th>
<th>% Exceeding Limit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Longwall Faces</td>
<td>16,686</td>
<td>1,131</td>
<td>6.8</td>
</tr>
<tr>
<td>Other Underground</td>
<td>32,583</td>
<td>531</td>
<td>1.6</td>
</tr>
</tbody>
</table>

From these sample results, it is clear that the current controls for mitigating longwall dust exposure levels is highly successful in the removal of respirable dust.

New South Wales government testing of inhalable coal dust conducted by Coal Services Pty Ltd in the state’s longwall mines has found more than a third of the samples taken exceeded the 10 mg per cubic metre limit (Gram, 2006)

A 10 mg/m³ limit on inhalable dust in coal operations was imposed in December 2007 by notice provided under the Coal Mine Health and Safety Act.

In an article dated Tuesday, 9 March 2010 in the International Longwall News (ILW, 2010), The Director of the Mine Safety Operations Branch under the Department of Industry and Investment, issued a safety alert to all mines who have been advised to identify and control risks in relation to excessive failures of inhalable dust exposure levels. According to the article, the results of coal dust testing in the Newcastle region revealed that 44 out of 104 samples taken in longwall operations exceeded 10 mg/m³ – a failure rate of 42.3%. Fifty of the 95 longwall samples in the Hunter region – which is more than half at 52.6% – failed the government limit. None of the 29 longwall samples in the Western region failed while 25.3% of the samples in the Southern District exceeded the limit.

Examining the sampling reports, the following were found to be the likely causes of high coal dust levels:

- Inadequate ventilation;
- Inadequate water or dust control;
- Poor operator positioning;
- Damaged equipment;
- Poor work practices.

It further suggests the following strategies to combat the problem:

- Isolation or capture of dust at source via sealing of transfer points, BSL, crushers;
- Operating water sprays at appropriate locations and as near as possible to the point of breakage with sufficient water volumes, pressure and correct sizing of water jets/droplets;
- Ventilation of the correct quantities and at the right location;
- Advance ventilation ducting/brattice to mine ventilation standard;
- Regular maintenance of dust suppression equipment;
- Operator positioning, job rotation and automation;
- Control of dust levels along travelling roads;
- Respiratory protection by personal protective equipment.
In contrast to the success of the current longwall dust controls in mitigating respirable dust, the above analysed results of inhalable dust exposure levels clearly indicate that the current longwall controls for mitigating inhalable dust are less successful.

USING DUST LOAD MEASUREMENTS TO DETERMINE CONTROL EFFICIENCY

A new testing methodology has been developed between the University of Wollongong, Coal Services, The Department of Investment and Industry and the CFMEU, to determine installed control efficiency for both respirable and inhalable dust. This new methodology retains gravimetric sampling for dust load sampling to ensure uniformity of the collection process, validity of the collected data and quantification of the analysed results. Also, the sampling methodology has been designed to ensure the collected data is deemed quantifiable to satisfy the requirements of a scientific research project and for reference in potential future projects. The objective of this new sampling methodology is to identify dust loads at independent sources of dust generation on longwall faces and quantify the efficiency of installed controls for the mitigation of dust generation. This data will then be used to create a benchmark or signature for the longwall mine in relation to dust loads from different sources of generation. Once this signature is established, quantifiable testing can be undertaken on new or improved controls to ensure maximum efficiency in removing respirable and inhalable dusts.

NEW TESTING METHODOLOGY

The dust collection process on a longwall must be arranged so that there is a collection of respirable and inhalable dust at each independent source of dust generation.

The first stage in this methodology is to determine the placement of monitors on each of the independent sources of dust generation. In each location, separate monitors and heads will be used to sample both respirable and inhalable dust loads. Figure 3 below details monitor and head placement along a longwall face. Next the amount of dust produced at each individual source of dust generation is measured. This will require the mine to turn off the controls at these individual locations during sampling period, to allow produced dust to be measured accurately at each of these sources.

This will not be an issue for the controls on outbye conveyors, travel roads, BSL discharge, crusher and shield sprays; however, turning off all controls on shearers will produce resistance from mine operators. It will be necessary to leave the drum sprays on as in most applications; these are used more for frictional ignition suppression than dust mitigation. Additional sprays such as crescent sprays and shearer clearers could be turned off for the period of the testing; assuming gas levels are below ignition points.
Controls will be turned back on and sampling heads changed to remeasure dust loads with controls operating. The difference between these two tests will determine the efficiency of the installed controls. The sampling strategy can be tailored to each individual mine to have on or minimum impact on production interruption.

RESULTS OF THE NEW TESTING METHODOLOGY

A NSW Hunter Valley underground coal mine was the first mine to have the new testing methodology applied in October 2010. The intention of this research is to test every longwall mine in NSW to obtain a best practice dust control setup and prove the new testing methodology can be used as a valuable alternative tool to the current statutory testing.

The results of the testing found that the current installed controls reduce the amount of respirable dust by an average of 40% and the amount of inhalable dust by an average of 34%.

Other findings indicated that outbye roadway dust entering the longwall through the last open cut-through needs to be addressed along with the dust control on the maingate drum. Whilst the tailgate operator saw a reduction of 40% when the controls were turned on for respirable dust, the maingate operator only saw a 27% reduction in dust exposure levels.

Although this result may look solid on the surface, detailed analysis of each individual source of dust generation gives a true picture of how well installed controls mitigate the dust at each individual source of dust generation and highlights where improvements can be made to further enhance control efficiencies.

The results for the last open cut-through indicate that there was a dust load of 0.32 mg/m$^3$ of respirable dust and 1.18 mg/m$^3$ of inhalable dust during the first set of tests with no controls operating for 255 minutes. The second set of tests indicated, when the raw data was adjusted to 255 minutes, that there was a dust load of 0.23 mg/m$^3$ for respirable dust and 1.97 mg/m$^3$ for inhalable dust. This shows a reduction in respirable dust levels of 29% and an increase in inhalable dust levels of 67%.

The results in the belt road show a 37% decrease in respirable dust levels from 0.40 mg/m$^3$ with no controls on to 0.25 mg/m$^3$ with controls on. Inhalable test results indicate a 31% reduction in dust levels from 2.33 mg/m$^3$ with controls off to 1.61 mg/m$^3$ with controls on. This indicates that the installed controls which include sprays on the BSL discharge are effectively mitigating both the respirable and inhalable fractions from the airway.

At the maingate chock, or chock No 1, respirable levels were reduced by 38% from 0.71 mg/m$^3$ with controls off to 0.44 mg/m$^3$ with controls on. Inhalable levels were reduced by 89% from 96.29 mg/m$^3$ with controls off to 10.56 mg/m$^3$ with controls on.

Chock No 5 saw a respirable dust level of 0.88 mg/m$^3$ with controls off and 0.80 mg/m$^3$ with controls on giving a 9% reduction in dust levels. The corresponding inhalable reading has been discarded as the inhalable sample head for the second set of data had little or no dust on the filter as the pump was either not working or had been turned off and not turned back on when the heads where changed.

Chock No 5 respirable levels have remained high as this is the point on the face where crusher dust is flushed over due to the maingate corner sprays and maingate wing forcing the ventilation further along the face.

Chock No 25 saw a 56% reduction in respirable dust from 2.84 mg/m$^3$ with controls off to 1.25 mg/m$^3$ with controls on. Inhalable dust was marginally decreased by 1% from 19.37 mg/m$^3$ with controls off to 19.09 mg/m$^3$ with controls on.

At chock No 45, respirable levels were reduced by 45% from 3.37 mg/m$^3$ with control off to 1.86 mg/m$^3$ with controls on. Inhalable dust was reduced by 73% from 36.6 mg/m$^3$ with controls off to 9.98 mg/m$^3$ with controls on. This number seems exceptionally high when compared to the samples collected on chock No 65 which showed an increase in inhalable dust of 36% from 28.59 mg/m$^3$ with controls off to 38.77 mg/m$^3$ with controls on.
The corresponding respirable results at chock No 65 indicate a 47% reduction in dust loads from 3.63 mg/m³ with controls off to 1.93 mg/m³ with controls on.

Chock No 85 saw a reduction in respirable dust of 63% from 5.68 mg/m³ with controls off to 2.12 mg/m³ with controls on. Inhalable dust was reduced 36% from 22.28 mg/m³ with controls off to 14.34 mg/m³ with controls on.

Chock No 105 saw a reduction in respirable dust of 50% from 4.07 mg/m³ with controls off to 2.05 mg/m³ with controls on. The inhalable dust load again had a failure during the second set of tests, so no result is available for analysis.

Finally, the tailgate tests have shown to be the most interesting with an actual increase in both respirable and inhalable dust loads. The respirable fraction increased 44% from 2.21 mg/m³ with controls off to 3.19 mg/m³ with the controls on. Similarly, the inhalable fractions increased 25% from 49.43 mg/m³ with controls off to 61.88 mg/m³ with controls on.

The tailgate operators’ position saw a 40% reduction in respirable dust from 1.65 mg/m³ to 0.99 mg/m³ whilst the maingate operator position saw a 27% reduction in respirable dust from 1.33 mg/m³ to 0.99 mg/m³.

The findings of this testing process clearly define an operating signature and benchmark performance of the currently installed controls to remove both respirable and inhalable dusts on the longwall face.

Figures 4 and 5 clearly show the effectiveness of the installed controls at each of the sampling points for respirable and inhalable dust respectively.

**CONCLUSIONS**

It is very difficult and in many instances expensive to measure control efficiencies, with many mines relying on subjective opinion as to the effectiveness of the installed controls. Little or no scientific research has been undertaken to quantify how effective installed controls are in relation to removing the produced dust on operating longwalls.

The current longwall controls for mitigating dust highlight a serious dichotomy in the results obtained during statutory testing for respirable and inhalable dust exposure levels.

Respirable dust exposure levels are well controlled, with less than 6.5% of all samples taken being above the regulatory exposure limit, indicating that current installed dust mitigation controls are working.

In contrast to the success of longwall dust controls in mitigating respirable dust exposure levels, are the results of inhalable dust exposure levels testing, which shows that in excess of 30% of samples taken exceeded the statutory exposure levels.
This dichotomy of results indicates that a serious problem exists where the smaller respirable particles, usually less than 10 μm in size are removed from a contaminated airway, whereas the larger inhalable particles, usually greater than 10 μm, are not removed.

The current statutory testing regime for respirable and inhalable dust samples identifies the exposure levels of personnel on an operating face which gives a snapshot of the dust that these persons will be exposed to over the duration of a mining shift. Although this testing process clearly determines compliance to the Coal Mines Health and Safety Regulation 2006, it does not give mine operators any indication of where dust is produced, how much dust is produced nor how efficient installed controls are at mitigating produced dust.

The use of this new testing methodology will provide mine operators with a complete dust production signature of their operating longwall and allow the implementation of more efficient controls at independent sources of dust generation that will have a quantifiable efficiency number on dust knockdown.

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Coal Mines (Underground) Regulation 1999.
International Longwall News, 1 April 2010 SB10-05 Airborne Dust - Inhalable Dust Control - Chief Inspector Safety Bulletins, NSW.
DEVELOPMENT OF A Water-Mist Based Venturi System for Dust Control from Maingate Chocks and BSL

Ting Ren¹, Graeme Cooper² and Srinivasa Yarlagadda³

ABSTRACT: Advances in modern longwall (LW) technology have resulted in high production faces with more powerful chocks and shearsers that can advance at faster rates. As longwall chocks (supports) advance, crushed roof coal and/or rock can fall from the top of the chock canopy into the face ventilation airflow. Dust survey showed that chock movement is a significant source of dust exposure for shearer operators, accounting for about 47% of total LW face dust make during the cutting cycle. 3D CFD models have been developed to understand the behaviour of longwall dust particles from various sources including maingate (MG) chocks and stage loader/crusher. Modelling results demonstrated that much of the respirable dust particles generated from MG chock movements and the beam stage loader (BSL) will disperse onto the longwall face ventilation, contributing significantly to dust exposure levels. Dust control systems using ultra fine water-mist technology have demonstrated promising results in encapsulating respirable dust particles. A prototype water-mist based venturi system has been developed to firstly produce ultra-fine water droplets (5-15 µm) for suppressing the respirable dust particles from the MG chocks/BSL; and secondly induce a water-mist airflow with sufficient momentum to divert dust clouds off the walkway area along the face. The new system will be trialled on MG chocks in medium to high seam longwalls on which dust contamination appears to be more problematic.

INTRODUCTION

Respirable dust particles have long been known to be a serious health hazard to workers in coal mining. Prolonged exposure to excessive levels of airborne respirable coal dust can lead to coal workers’ pneumoconiosis (CWP), progressive massive fibrosis (PMF), and chronic obstructive pulmonary disease (COPD). These diseases are irreversible and can be debilitating, progressive, and potentially fatal in their most extreme cases.

Dust particles can be generated from several sources on the longwall, primarily including shear cutting, chock movements, stage loader/crusher and intake contaminations. A recent dust survey by Gillies Wu Mining Technology using real-time Personal Dust Monitoring (PDM) units showed that the advancement of MG chocks leads to significant dust falling into the face airstream, accounting for about 47% of total LW face dust make at the Shearer operator position during the cutting cycle (Gillies and Wu, 2008); The survey also indicated that if the BSL scrubber is not working optimally, the bulk of the dust particles will escape over the BSL and end up on the face increasing the overall dust levels.

Effective dust control is important for the occupational health and safety of production crews and eventually production outputs. A variety of water spray dust control systems have been trialled, and some have been applied in the field with mixed results (Goodman, 2000; Pollock and Organiscak, 2007). Dust control systems based on water mist technologies have also been tested and more recently trialled in underground coal mines Australia and overseas with promising results. Dust suppression systems using water mist technology have recently been tested at conveyor transfer points and demonstrated promising results. Water droplets from the water mist system are of a comparable size (1-10 µm) to respirable dust, and thus can be used more effectively to knock-down dust particles before they become airborne.

ACARP is supporting a project to develop and test a new type of venturi system based on ultra-fine water mist technology to reduce respirable dust contamination on medium and thick seam longwall faces, particularly those dust particles from the advancement of MG chocks and the intake ventilation passing the BSL. This new venturi system will be developed as a stand-alone unit that can be easily attached to the chock canopy with minimum interaction with other longwall equipment. This paper reports the development of this new dust control system.

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DUST FROM LONGWALL MG CHOCKS - BASELINE DUST SURVEY

Gillies and Wu (2007; 2008) conducted extensive real-time respirable dust monitoring and baseline surveys on Australian longwalls. The results from a LW face baseline survey located in the Bowen Basin, Queensland, are shown in Figure 1. Two real-time PDM were placed at separate locations (#134 shadowing the MG shearer operator and #139 at the MG Chock 8 position) on the face to measure the dust exposure. During the cutting sequence from MG to TG, 1-5 MG Chocks were advanced immediately after the shearer passed. This action leads to much dust falling from the advancing chocks; dust levels registered by both PDM units were increased significantly, accounting for about 47.8% of total longwall dust make at the Shearer MG operator’s position during the cutting cycle, as indicated in Figure 1. This measurement was in agreement with another longwall dust survey conducted by Gillies and Wu (2007). Similar observations have also been reported at other mines in Australia.

![Figure 1 - Dust survey on a longwall showing MG Chocks (1-5) advance dust versus total longwall dust make at Shearer MG and TG Operator positions (Gillies and Wu, 2008)](image_url)

The dust baseline survey results demonstrated the importance of reducing respirable dust generated from the advancement of MG chocks thereby significantly mitigating total dust make to the longwall face. Dust monitoring also showed that the BSL can be another major dust contributor, even for longwalls equipped with a BSL dust scrubber. Dust surveys indicated that the scrubber can cleanse only a portion of the air travelling to the face, allowing much of the dust particles to escape over the BSL and to end up on the longwall face, increasing the threshold dust levels in the ventilation air.

DUST CONTROL FROM CHOCK MOVEMENT

Longwall supports are typically advanced within two or three shields of the trailing shearer drum. As longwall chocks (supports) are lowered and advanced, crushed roof coal and/or rock falls from the top of the chock canopy directly into the face ventilation airflow. Most of this dust becomes airborne, and quickly disperses into the walkway. As a result, chock movement can be a significant source of dust exposure for shearer operators when supports are advanced behind the shearer during MG to TG cuts. To control dust from chock movement, a number of methods have been developed (Colinet, et al., 2010). Two such systems are:

- **Canopy-mounted spray systems** - A canopy spray system that activates water sprays into the roof material on top of the supports for a short period of time before and during chock movement to wet the material on top of the canopy to lower dust levels during shield advance, as shown in Figure 2.a. Experience in the US and Australia has shown that this type of system is hard to maintain and is not effective in distributing moisture to the material above the canopy.
• **Shield sprays under the canopy** - These sprays were automatically activated by the position of the shearer to create a moving water curtain in an attempt to contain the dust cloud near the headgate and tailgate drum areas, as shown in Figure 2.b. Proper on/off sequencing of these sprays is critical to supplement the directional spray system. These sprays need to be properly aligned toward the face to enhance the envelope of clean air created by the shearer’s directional spray system.

![Spray system over canopy](image1)

![Spray system under canopy](image2)

**Figure 2** - Water sprays located above and on the underside of the canopy (Colinet, *et al.*, 2010)

**DUST CONTROL USING WATER MIST**

Historically water sprinklers/hoses have been used for dust control on longwalls to suppress the dust particles before they become airborne (Goodman, 2000; Colinet, *et al.*, 2010). The fundamental principles for dust suppression is to allow the water drops to collide with dust particles in the air, forming heavy agglomerates of dust and water, resulting in a "settling out" of the airborne dust. However, conventional hydraulic water sprays are not effective on respirable dust. With typical diameters of 200-600 µm sprays, the droplets are much larger than the dust particles they are attempting to suppress. Water drops that are too large will not collide with the finest, most hazardous dust particles smaller than 10 µm. Airborne water droplets and dust particle attraction is most likely to occur when the droplets and dust particles are of similar size.

As shown in Figure 3, consider a water droplet is about to impact on a dust particle, or aerodynamically equivalent, a dust particle about to impact on a water droplet, if the droplet diameter is much greater than the dust particle, the dust particle would simply follow the airstream lines around the droplet, and little or no contact would occur; whilst if the water droplet is of a similar size to that of the dust particle, contact would occur as the dust particle tries to follow the streamlines. Thus the probability of impact can be increased by

- Increasing the number of smaller sized spray droplets per unit volume of water utilised;
- Optimising the energy transfer of spray droplets with the dust-laden air.

![Water spray and mist technology for dust control](image3)

**Figure 3** - Water spray and mist technology for dust control (Joshi, 2009)
Dust suppression systems using ultra fine water mist technology have recently been tested in underground coal mines and demonstrated promising results. In Australia, a simple dust suppression system using ultrasonic atomisers has been installed at the MG transfer point at Broadmeadow and excellent dust control result has been observed (Burges, 2009). Also at Broadmeadow, a simple water mist system is being tested for dust suppression from longwall chocks and promising results demonstrated. Similar initiatives are being considered in other Australian coal mines. In South Africa, a fogger system using water mist technology has been investigated at Thandeka and Twistdraai as water curtains on the intake airway and transfer points (Schoor, 2010). Field results proved that the system was highly effective by reducing dust concentrations by 96% during the test periods. Also in South Africa, an air mover system known as Terrajet® has been under development by Terramin and trialled in the field with promising results (Schoor, 2010).

It was therefore proposed to develop a water mist based venturi system that could be attached to the canopy of MG chocks to suppress respirable dust particles from chock movements and intake air streams passing the BSL, whilst also acting like a directional spray system to enhance the diversion of dust clouds away from the walkway areas along the longwall.

**CFD MODELLING OF DUST DISPERSION FROM MG CHOCKS AND BSL**

**CFD models**

The behaviour of respirable dust on a longwall face is a complex process because of the nature of longwall operations. The generation, dispersion and transport of airborne dust are mainly governed by the spatial velocity and the movement pattern of the ventilation air. To understand the dust behaviour and thereafter assist in the design and evaluation the effectiveness of dust control techniques, ACARP has been supporting research projects based upon CFD modelling to improve the understanding of dust flow patterns around the longwall shearer, and the study of a range of dust control options/concepts for reducing operators dust exposure levels (Ren and Balusu, 2010). In this study, three dimensional CFD models were built to represent longwall faces in medium and thick seams. These models consist of a section of the full scale coal face and the maingate, and embody the major longwall components such as chocks, shearer, spill plate, BSL/crusher and conveyor. In addition, an array of water mist injection points were incorporated into the model along the chocks and around the AFC-BSL transfer point where the venturi units are likely to be installed based upon field observations. Figure 4 shows the layout of the longwall CFD model.

Base model simulations were carried out with a variety of intake airflow rates, ranging from 50 m$^3$/s to 100 m$^3$/s without the intervention of any dust controls. The base-case CFD models were calibrated and validated against field airflow velocity data obtained from field ventilation data and used for further parametric studies of the water mist venturi systems.

**Dust flow modelling**

A major challenge in this study is the modelling of respirable dust particle dispersion in the turbulent flows on the longwall face. Longwall airflow is highly turbulent due to the large Quantity of air supply and the existence and movement of mining equipment, and such turbulent flows will impact on the dispersion of respirable dust particles along the face. For this study, the standard $k-\varepsilon$ Model was chosen to model the turbulent airflow in the longwall face. In addition, the uncoupled approach was adopted to model the dust particle dispersion patterns, in which all the dust particles were treated as ‘respirable’ and as such the discrete phase (respirable dust particles) will not impact the continuous phase flow (airflow) pattern. Essentially, all the simulations were carried out as a single phase steady-state model.

Base-model simulations were conducted to investigate the behaviour of respirable dust dispersion from various sources on the longwall. In these simulations, a group of dust particles were ‘released’ as coal-hv (material) with particle sizes between 1–10 µm. It was assumed the particle size distribution from these releasing points follows the Rosin-rammler distribution function. The dispersion of particles due to turbulence in the continuous phase flow phase (air) was tracked using the stochastic tracking model, which includes the effect of instantaneous turbulent velocity fluctuations on the particle trajectories through the use of stochastic methods. Figure 5 shows the tracking of dust particles released from outbye (belt road), AFC-BSL transfer point, and MG chocks. CFD modelling results show that much of the respirable dust particles generated from chock movements near the MG and the BSL.
will end up on the longwall following the ventilation, contributing significantly to dust exposure on longwalls, if not controlled by effective dust mitigation methods.

![Diagram of the longwall CFD model](image1)

(a) Geometry of the longwall CFD model

(b) Computational grid of the longwall CFD model with ‘venturi units’ on longwall chocks

(c) Computational grid of the longwall CFD model with ‘venturi units’ around AFC-BSC transfer point

Figure 4 - Layout of the longwall CFD model for dust modelling

(a) Dust particles from outbye (Belt road)

(b) Dust particles from AFC-BSL transfer point

(c) Dust particles from MG chocks 1-2

(d) Dust particles from MG chocks 4-5

Figure 5 - CFD modelling showing dust dispersion from belt road, BSL and MG chocks

Operational experiences have demonstrated that it is unlikely to achieve total dust capture for any longwall dust control systems due to the large cross sectional area and very high airflow on the face. Therefore, the water mist based venturi system, in addition to capturing a proportion of the dust particles, must be positioned to divert dust particles away from operators’ walkway area.
Modelling of water mist venturi

To optimise the position of the water mist venturi units on the MG chocks, the base CFD models were used to conduct parametric studies on a number of combinations of venturi positions along the MG chocks (1-6) and on the AFC/BSL plate, including:

1. With venturi attached the chock canopy
   - Venturi at canopy level – towards the coal face;
   - Venturi at canopy level but tilted along the face (10~45˚);
   - Venturi tilted down and along the face.

2. With venturi stationed on the AFC/BSL plate
   - Venturi spray towards the coal face;
   - Venturi spray with angles along the face.

In all cases, the water mist injection was modelled as `air spray’ as much of the spray would be of compressed air with a small portion of fogged water droplets. This also avoids the complexity of modelling multiphase flow which would require much computing power and time. Figure 6 shows the impact of venturi units oriented at different locations on the diversion of dust particles from both MG chocks and the AFC-BSL transfer point. Modelling results show that a more effective control of dust particles from MG chocks and BSL can be achieved by:

   - Slightly tilting the venturi units towards the floor and the coal face with optimum angles between 15-20˚ down and 45˚ along the face;
   - Installing two venturi units on each chock;
   - Installing two venturi units on the AFC/BSL plate;
   - Installing a batch of venturi units in the first 6 MG chocks to achieve the overall dust flow streamlining effect.

Results from the CFD modelling will be used to assist field trials of the newly developed water mist based venturi systems as described below.

**PROTOTYPE WATER MIST VENTURI UNITS**

Using the latest development of ultrasonic nozzle technology, a new prototype of water mist venturi system has been developed to perform the following two functions:

   - To produce uniformly-distributed ultra-fine water droplets (5 - 15 µm) for encapsulating and trapping a high proportion of the respirable dust particles from the MG chocks/BSL before they become airborne and reach the walkway area;
   - To induce a controlled volume of water-mist airflow with sufficient momentum for diverting and suppressing respirable dust clouds from MG chocks/BSL off the walkway area along the face.

To achieve the above design requirements, extensive laboratory tests were carried out with different venturi chamber diameters (42 ~ 70 mm), different venturi lengths, methods for mounting the ultrasonic nozzles, as well as air pressures (ranging from 8 - 2 bar) and water pressure (ranging from 6 - 1 bar). The ultrasonic nozzle holder assembly with different nozzles was used to test the best combination and various distances between the nozzle and the venturi to optimise the water mist production and air induction effect. Tests results indicate that the 70 mm (diameter) x 143 mm (length) venturi is capable of producing an optimum spray coverage and spray distance of approximately 10-12 m. Figure 7 shows the various parts of the venturi unit and laboratory testing process.

This new venturi system essentially consists of a water mist generating chamber incorporating mounting holes via which water and compressed air can be introduced to the ultrasonic atomisers that produce very fine droplets. Water is ejected through a number of orifices into the nozzle air outlet channel, where the high velocity air stream produces a first liquid breakup through shear action. The air stream, carrying
the droplets, collides with a resonator placed in front of the nozzle outlet channel that generates a field of high frequency sound waves. Water delivered to the resonator is shattered into fine droplets which are then carried downstream by air by-passing the resonator.

(a) Venturi at level but tilted at 20˚ along the face

(b) Venturi at level but tilted at 45˚ along the face

(c) Venturi tilted at 20˚ down and 30˚ along the face

(d) Venturi tilted at 20˚ down and 45˚ along the face

Figure 6 - Impact of venturi units at different locations on dust particles

The novelty of this new system is its capability to draw sufficient air into the chamber to carry the atomised droplets downstream with sufficient momentum for maximum dust particle attraction and controlled diversion away from the walkway area. In order to utilise the existing water and compressed air supply on the longwall face, the system is built as a stand-alone module with a magnetic base which can be easily attached to the chocks’ canopy and adjusted with the right spray angle to achieve the droplet size and velocity needed for dust suppression and diversion, as indicated by CFD modelling. Figure 8 shows the complete prototype water mist venturi systems that are ready for field trials.

Laboratory tests showed that having a ratio of 2:1 air to water seems to produce the best atomisation for the UMV nozzle., i.e. if liquid pressure is 3 bar, ideal air pressure will be 6 bar. Water consumption at 3 bar liquid pressure and 6 bar air pressure is 2.15 L/min or 0.0358 L/s. Air consumption is 35.2 Ncm/hour. The mist produced is of such a fine size that it remains airborne as a “dry fog”. As the mist evaporates, it has an evaporative cooling effect which, over a period of time, can have the effect of reducing temperatures on the longwall face.
Figure 7 - Development of the prototype water mist venturi system

Figure 8 - The completed prototype water mist venturi units
FIELD TRIALS

On completion of the design of the water mist venturi systems, it was planned to carry out field trials at longwall faces in the Bowen Basin, Queensland and the Hunter Valley, New South Wales. It was anticipated that the initial trials would involve at least two longwalls, likely Moranbah North, Broadmeadow Mine or South Blakefield. Based upon the initial trial results, the system will be modified, if necessary, and then demonstrated at other mines.

The venturi units will be trialled on the 1-6 MG chocks to knock down and divert the highly concentrated dust clouds from MG chock movements and the BSL. For medium height seam with relatively tight clearance below the chock canopy, the venturi units can be mounted on the top of the longwall Control and Communication Panel and on the AFC-BSL plate, as shown in Figure 9, therefore reducing dust contaminations from 1-3 MG chocks and the belt road. The device will also be trialled near the TG chocks to assist in dispersing methane that builds up around the longwall return corner and to reduce the total dust make from the longwall.

Figure 9 - Proposed locations for installing the venturi units on medium seam longwalls

In December 2010, the first field trials were conducted at Moranbah North and Broadmeadow Mine however the results have not yet been made available for reporting at this stage. Figure 10 shows the venturi units attached to the canopy of a longwall chock before being dispatched to underground longwall face.

Figure 10 - Venturi units attached to the canopy of the longwall chock for field trials
CONCLUSIONS

Chock movement close to the MG has been identified as a significant source of dust exposure for longwall operators when chocks are advanced upwind of the shearer during MG to TG cutting. Extensive CFD modelling studies showed that much of the dust particles will follow the ventilation into the longwall, contributing significantly to longwall workers’ dust exposure, if not controlled by effective dust mitigation methods. A new water mist based venturi system has been developed capable of producing ultra-fine water droplets for suppressing dust particles from the MG chocks/BSL whilst inducing a controlled volume of water-mist airflow for diverting the dust clouds off the walkway area. Further field trials of the new venturi system have been planned and will be conducted in Australian longwalls. Applications of this system will greatly reduce dust contamination from longwall chock movements near the maingate, in particular for medium to thick seam longwalls on which dust control appears to be more problematic.

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REFERENCES

PROACTIVE STRATEGIES FOR PREVENTION AND CONTROL OF FIRES IN BORD AND PILLAR MINES WORKING IN THICK COAL SEAMS

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ABSTRACT: Coal fires associated with spontaneous combustion pose significant risks for underground mines. The hazard to the safety of the people working in mines, the loss of valuable coal reserves, the addition to fugitive Green House Gas (GHG) emissions and the huge costs involved in controlling the aftermath situations put these into one of the most critical parameters which have to be dealt with. Some of the research attempts taken to prevent and control coal mine fires and spontaneous combustion in thick seams worked with bord and pillar mining methods is presented.

A mine located in India which has a 22 year history of working a 10m thick coal seam with the blasting gallery (BG) technology for extraction of the pillars was selected for the study. The project investigated the geological, geotechnical, operational and ventilation parameters causing the frequent fire incidents. Several laboratories, desktop and modelling studies were made to give a better understanding of the contributing factors and develop strategies for the control of mine fires.

In the study computational fluid dynamics (CFD) modelling techniques were used to simulate and assess the effects of various mining methods, layouts, designs, and different operational and ventilation parameters on the flow of goaf gases in BG panels. A wide range of parametric studies were conducted to develop proactive strategies to control and prevent ingress of oxygen into the goaf area preventing spontaneous combustion and mine fires.

INTRODUCTION

Coal mine fire incidents increased significantly in recent years, as underground coal mining progressed towards extraction of thick seams, multiple seams and spontaneous combustion prone seams. Fires initiated by spontaneous combustion pose a major safety risk with the potential for explosion, but also can cause significant loss of coal reserves and contribute to considerable greenhouse gas emissions.

In Queensland about four high potential heating incidents occurred from 2004 to 2009 (Report by Qld mining inspectorate, 2010). In NSW mines about 16 self heating incidents are reported from 2005-2009 (NSW, Mine safety performance report, June 2010). Data from the United States reveal that more than 20 underground coal mines fires are due to spontaneous combustion occurred during the period from 1990 to 2006 (Trevits, et al., 2009). In china every year about 360 fire incidents are reported due to spontaneous cumbustion. In India it is estimated that 75 % of coal mine fires result from prolonged exposure of coal to atmospheric oxygen (Mohalik, et al., 2009).The analysis of the various spontaneous combustion incidents in different countries, indicate that several factors like coal left in the goaf area, slower rate of extraction, presence of geological disturbances, method of mining, ventilation practices and coal characteristics are the main cause of mine fires and heatings (Balusu, et al., 2005).

Working thick seams with bord and pillar technologies is a regular practice in Indian Coal mines. A French blasting gallery (BG) technology; (a thick seam mining method) was introduced in 1988 in several mines of The Singareni Collieries Company Limited (SCCL); a government owned coal mining company in India. In one of the mines where the BG method is deployed there were repeated incidents of heating and fires with spontaneous combustion. About 60% of the BG panels are closed prematurely after extraction of 55-60% of coal (SCCL, internal report, 2007). The study explained in this paper aimed at developing control strategies for prevention and control of fires due to spontaneous combustion at this coal mine. The work taken up though includes field characterisation, laboratory investigations, field monitoring and design of proactive fire control strategies, here more emphasis is given on the gas flow modelling studies. The study also investigated the impact of different mining designs, methods and

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ventilation systems on spontaneous combustion events and developed appropriate proactive control strategies.

OVERVIEW OF THE BLASTING GALLERY (BG) MINING METHOD AND FIRE ISSUES

The Blasting gallery (BG) technology was first introduced at Godavari Khani-10 Incline (GDK10 Colliery) of the Singareni Collieries Company Limited and subsequently followed at other mines of SCCL. The method was introduced in SCCL mines because of the advantage of a high percentage of extraction and its suitability for use in thick seams.

The working of the BG method involves development of rectangular stooks in the bottom section of the coal seam. The stooks are extracted by drilling and blasting of long holes up to the full thickness (10m) of the coal seam. Remotely operated electro hydraulic tyre mounted load haul dumpers of 3 m³ capacity load the blasted coal and discharge onto a chain conveyor with inbuilt lump breakers in the outbye level or the rise gallery. During development, the galleries are supported by roof bolts. While extracting the pillars the galleries are additionally supported by rolled steel joists placed over a pair of 40 T open circuit (OC) hydraulic props.

About 22 BG panels were worked at the mine in three areas separated by faults. During the initial years the BG panels are worked without fire issues in shallow depth areas. When the workings were shifted to deeper regions incidents of heatings and fires due to spontaneous combustion significantly increased. Some of the panels were worked only for 5 months before encountering spontaneous combustion problem. To summarise, the mine experienced fires in 60% of the BG panels leading to pre-closure after extraction of 55-85 % of the area of the panels.

Attempts are made to control the fires in BG panels by following some traditional inertisation practices like flushing with nitrogen and CO₂. In some cases foam technologies and fire sealants are also used. On a review of the past fire instances it is understood that though inertisation attempts were made in both sealed off panels and the working panels but the success rate varied widely. Some panels were sealed off early even following inertisation techniques, whereas some panels worked with limited success. A critical assessment of past fire incidents and inertisation attempts revealed that:

- The spontaneous combustion issues in BG panels are increasing with increase in depth of mining, associated with the changes in panel design such as larger pillars and stooks;
- Ventilation factors such as sluggish air flow in the goaf areas and leakages between the panels and high pressure differentials have contributed to development of spontaneous combustion;
- Geotechnical factors such as strong roof and associated caving difficulties, thick seam extraction, crushing of stooks in the panel and crushing of barrier pillars between the panels contributed to the development of spontaneous combustion;
- Coal left in the floor of the working section and loose coal left in the goaf due to larger stooks have contributed for spontaneous combustion; and
- Traditional inertisation practices such as inert gas injection at the top level in the panel at very low flow rates are found to be ineffective in controlling spontaneous combustion in BG panels.

DEVELOPMENT OF COMPUTATIONAL FLUID DYNAMICS (CFD) MODELS AND PARAMETRIC STUDIES

Simulating the ventilation and goaf gas behaviour with different parametric conditions that exist in the field with the help of computational fluid dynamics (CFD) techniques is already being practised for Australian mine conditions. Similar CFD studies for air flow patterns in BG panels are conducted for a better design of inertisation strategies. The main objectives of the CFD modelling are:

- To identify air flow patterns in the goaf areas of BG panels with different ventilation systems;
- To investigate the inert gas flow patterns in the goaf areas with different inert gases, injection point(s) and flow rate(s);
- To carry out extensive parametric studies on variations in ventilation flow rates, panel design parameters, operational parameters, and
To identify and develop the most effective strategies for control of spontaneous combustion and mine fires.

Modelling is carried out using “FLUENT”, a CFD code that solves the finite volume Navier Stokes equations. Initially the base model is set up to simulate gas flow in the BG panel working and goaf atmosphere. The boundary conditions for the simulations are assumed based on field observation data, geo-mechanical studies and historic data. Figure 1 shows the CFD model geometry for the BG panel with dimensions of the model. The main parameters for the base model are as follows:

- 3D BG panel with a dimension of 216 m (panel length) X 220 m (panel width) X 30m (Goaf height above roof);
- There are 9 rooms for the extraction operations;
- The panel is extracted up to an area of about 9000 m$^2$;
- The height and width of the roadway are 2.8 m and 4.2 m respectively.

A key part of the CFD model is the incorporation of BG panel goaf permeability distributions and goaf gas emission via a user defined function (UDF) linked to the CFD solver. The intake and return of the BG panel are defined as velocity inlet or outlet boundary conditions with other seals and solid pillars as walls in the model. Several other factors like diluting the blasting fumes, dust control and control of face temperature and humidity are also considered before finalising the base case CFD model for conducting different parametric studies. In the base CFD models several simulations are studied with options like, single and multipoint inert gas injection, variations on inertisation by changing the inert gas flow rates, effect of type of inert gas used and effect of sealing the bottom on inertisation.

Studies revealed that effective inertisation is achieved with three optimum inertisation injection points which are located inbye near to the start line along the dip and the bottom level. The inert gas flow rates of about 20 L/s from each point is found to be the optimum quantity for effective inertisation. Better inertisation effects are observed when CO$_2$ is used as inert gas compared to N$_2$ and boiler gas. The CFD model with the optimum inertisation pattern used for parametric studies is furnished in Figure 2.
Inert gas - Carbon dioxide = 20 l/s at each point

Figure 2 - Inertisation results with CO2 at 3 locations

In the BG working system the bottom levels are extracted first and are sealed off periodically from the lower level to higher level in ascending order before the completion of the panel. Several CFD simulations are carried out for finding the pattern of inertisation when the bottom most levels are sealed off. The results indicated that effective and improved goaf inertisation is achieved by sealing off the two bottom most levels of the BG panel.

Several parametric studies are conducted to study the effects of ventilation flow rates, face orientations and, methane emission rates on the inertisation pattern. With respect to the quantity of gas flow in BG panels the CFD results indicate that there is no marked difference in the inertisation pattern at the bottom levels of the BG panel when the ventilation flow is either halved or doubled. But it is observed that, the lower the air intake rate, the better is the performance of inertisation. The studies conducted on the effect of panel orientation indicated that the buoyancy effect is not significant as the chosen inclination of 1.28° is small. Only minor differences are observed when the BG panel is dipping inbye where the inertisation at the start line and bottom levels is found to be marginally more effective. The study simulations on the amount of methane gas released in the BG panels indicated that the variation in gas emission rates has no significant effect on inertisation.

DEVELOPMENT OF PROACTIVE FIRE CONTROL AND PREVENTION STRATEGIES

From the results of CFD modelling and extensive parametric studies the following important parameters are considered for designing the optimum strategies for control of spontaneous combustion and fires in BG panels:

- design of ventilation system for working BG panels;
- establishing a comprehensive gas monitoring system;
- conducting induced blasting for proper caving of the BG panel goaf ; and
- inertisation and sealing practices while working BG panels.

From the results of the initial field trials and the extensive CFD modelling with parametric studies it is considered that choosing the appropriate ventilation system will be one of the most important factors for control of mine fires. CFD models are used for simulating the effect of different ventilation systems on the goaf gas flows in the BG panels. Figure 3 shows the oxygen concentration patterns at the working seam level for BG panels with descentional and ascentional systems respectively. Modelling results indicated that ascentional ventilation allows air flow to migrate through much of the goaf area and therefore provides good flushing and cooling effects to the goaf; on the other hand, descentional ventilation has less goaf airflow flush and therefore offers advantages for goaf inertisation. It is considered better to follow the descentional ventilation in the BG panels for control of spontaneous combustion.
In the earlier BG panels, CO concentration in the return air is used as an indicator for detecting the development of spontaneous combustion. In most of the previous fire incidents the CO levels in the return air have increased more rapidly from almost nil values to about 300 ppm within a very short span indicating that the heating is in an advanced stage leading to premature sealing off the BG panels. The system of collecting the samples and the analysis methods are found to be unsuccessful in detecting the very low values of the spontaneous combustion indicator gases during the initial stages of heating.

Permanent steel pipes are installed in the working BG panels and sealed off BG panels for the dual purpose of injecting the inert gas to the required locations in the goaf and also to draw the goaf gas samples into the bags for further analysis. The sampling system is improved by way of drawing the samples into gas bags using a suction pump arrangement. The samples are analysed with gas chromatographs for the concentration of the goaf gases viz., CO, CO$_2$, CH$_4$, H$_2$, O$_2$ and N$_2$. The low level concentrations of CO and hydrogen (about 10 ppm levels) obtained from gas chromatograph analysis results are used as indicators for determining the early signs of heating in the goaf of BG panels.

In the BG system of mining the caving pattern of the roof strata into the goaf after the extraction of thick coal seam played an important role in contributing to the occurrence of fires. Good caving of the roof strata into the goaf area created a barrier for the ingress of oxygen into the goaf. In the new field strategy induced blasting practice is carried out more effectively for every 5 m of stook retreat with depth of the blast hole up to 10 m.

From the CFD simulation results the behaviour of the goaf gas flow is identified. The parametric studies with CFD models are conducted with several options like varying the ventilation, gas emission rates, inertisation points and quantities and change in operating conditions. The simulations helped to a better understanding of goaf gas flow pattern and helped choose the various options available for developing of most effective fire control strategies in BG panels. The results indicated that introduction of good inertisation practices is essential for preventing the spontaneous combustion and fires in BG panels. The inertisation pattern chosen for the BG panel is shown in Figure 4.

Based on the simulation results the following optimum strategies are implemented at the field site to control spontaneous cumbustion incidents and mine fires in BG panels:

- BG panel is ventilated with descentional system with a quantity of 2200 m$^3$/min to 2800 m$^3$/min for adequate comfort levels at workings and for effective heat dissipation.
- Periodic and frequent induced blasting is carried out in BG panel galleries for uniform caving of goaf and good packing of goaf area.
- Permanent steel pipes are installed for inert gas injection and goaf gas monitoring in BG panel.
Inertisation attempts are initiated after 2-3 months of panel retreat with a flushing rate of 3-4 T/day for a period of 3-4 months at two to three points located most inbye in the bottom most levels.

- CO$_2$ is used as inert gas for the working panel and nitrogen in the neighbouring sealed off panels.
- Bottom most levels are sealed immediately after completion of coal extraction at outbye location and inert gas injection is increased to 8-10 T/day with more inertisation points.

![Figure 4 - Inertisation strategy adapted by injecting CO$_2$ at three locations after sealing off two bottom levels](image)

**RESULTS**

The induced blasting has resulted in regular caving and compaction of the goaf and controlled ingress of air. Injected inert gas into goaf is retained well and created a good inert atmosphere. Figure 5 depicts the sustained levels of CO$_2$ in the goaf area which is obviously due to better compaction of goaf by way of efforts made in the form of junction blasting in lower levels and induced blasting in all working levels.

![Figure 5 - Trend of goaf gases showing the effect of induced blasting](image)

Observation of variations of spontaneous combustion indicator gases shown in Figure 6 found that there is no increase in either CO or H$_2$ with the normal laboratory monitoring. But when the samples are analysed with a chromatograph the presence of considerable concentrations of CO and H$_2$ are observed. The improved goaf gas sample collection and analysis helped in detecting very low levels of coal oxidation gases at an early stage of heating due to spontaneous combustion.
During the implementation of field trials the bottom two levels were sealed off at three months and four months respectively after completion of coal extraction from these levels in the BG panel. The results of goaf gas monitoring at one of the bottom level after sealing is furnished in Figure 7. Observation of the goaf gas trends at these points indicated that CO$_2$ is found to be increasing by up to 50% and the oxygen levels are stabilised below 10% immediately after sealing off the levels. It is also seen from CO and hydrogen levels that coal oxidation is under control.

The results of the various strategies followed for control of fires and heatings due to spontaneous combustion are analysed in different stages during the total working period of the BG panel. During the initial three months of the panel the goaf atmosphere is in the formation stage and high levels of oxygen are observed in the goaf. Inertisation started at the lower level after two months of panel start. The CO$_2$ levels are found building up in the lower levels but not reaching the upper levels till three months period.

During the period of three to six months inertisation was started from two more points. The monitoring results during this period show that at the lower most level CO$_2$ levels reached up to 45-50% after 5 months and stabilised. It is also seen that though after 5 months the oxidation products of coal i.e., CO and hydrogen showed some increased trend for a few days at the lower most level but are immediately brought under control due to effective inertisation. In contrast to the results at the lower levels, the CO$_2$ build up is slow in upper levels and gradually reached up to 40% by the end of 6 months. There is no indication of increased oxidation of coal as CO and hydrogen levels at all the monitoring points are found to be below 10 ppm.

During the last stages of six to eight months of the panel the results show that the goaf inertisation is effective with the CO$_2$ levels stabilising at 40% and oxygen levels at 10-12% throughout the entire goaf area of the BG panel. The CO and hydrogen levels are found to be below 5ppm indicating no incidence of spontaneous combustion and fire in the BG panel. A snap shot of the range of the gas levels at the lower most level from the start till the end of the BG panel is furnished in Figure 8.
CONCLUSIONS

Increasing the frequency of induced blasting, initiation of goaf breakage at the start line and additional blasting at the lower level junctions are found to be effective for regular and good caving of goaf which resulted in better compaction of goaf. Sealing off the bottom levels quickly has contributed to better retention of injected inert gas in the goaf area. Both these factors have contributed to the preventing the occurrence of fires throughout the working of BG panel.

Introduction of advanced gas monitoring system helped in assessing the behaviour of O₂, CO₂, CO and H₂ in the goaf quickly. On detection of changed behaviour of goaf gases spot actions like increasing the inert gas injection and modifying the ventilation quantities are initiated for controlling the ingress of oxygen into the goaf.

Inertisation is found to be most effective strategy which has contributed to the control of spontaneous combustion in BG panels. The CFD models and parametric studies helped in identifying correct locations and the optimum quantity of inert gas required for prevention of fires in BG panel.

The various strategies implemented during the field trials are found to be effective and successful as the BG panel was worked safely and sealed off without any incidents of spontaneous combustion after 8 months from the start. It is to be noted that in earlier times most of the panels were closed without complete extraction of coal in five to seven months period.

Field trials demonstrated that risks of spontaneous combustion and mine fires are reduced considerably in the BG panels which could lead to continuation of the BG technology in the future for working thick coal seams. The study improved the safety status of the BG workings considerably which directly contributed to the better extraction of coal from the panel. Overall benefit to the coal industry is observed by way of improvement in safety, and increase in production and productivity.

REFERENCES


DESIGN OF WATER HOLDING BULKHEADS FOR COAL MINES

Verne S Mutton\textsuperscript{1} and Alex M Remennikov\textsuperscript{2}

ABSTRACT: Water control has been and remains a fundamentally important aspect of underground coal mine design and operation worldwide. Mining in the vicinity of large bodies of water, below a worked out coal seam or under confined aquifer or abandoned water logged workings is always fraught with the possibility of the danger of inundation. Inrush control is now part the Fatal Hazard Protocols which is a risk based process in all operations. In modern Australian underground coal mines, in which panel layouts have been extended and production rates are approaching 10 Mtpa, there is much focus on the control of water inflows. Flooding of mine workings can cause deterioration of roadways as was evidenced at Broadmeadow Mine in 2008 soon after the water receded due to a sudden reduction or pore water pressure, mobilisation of joints/cleats and swelling of clay layers in coal measures.

Ventilation seals are primarily used in underground coal mines to isolate abandoned or worked out areas. However these seals are often required to impound large volumes of water to control the hazard mostly at the inbye end of longwall operations and in natural valleys. A systematic approach is required for the design of bulkhead seals including consideration of the longevity of building materials, quality control during construction and methods to monitor performance of the retention system. In recent years it has become accepted practice to use numerical methods to provide engineering ratings for engineering structures including mine seals. In this paper, structural response under hydraulic loads was evaluated using high-fidelity physics-based (HFPB) finite element models of ventilation seals.

INTRODUCTION

Mine operations are reliant on water retention bulkheads to provide an active barrier betwen impounded water and active mine workings. Catastrophic failure of a bulkhead could put workers and the continuance of mine operation at risk. Vutukuri and Singh (1995) demonstrated that accidental inundation is one of the major hazards when mining near old water logged workings or workings near hydrogeological anomalies e.g. karsts. Major sources of water inflow are seepage from poorly sealed shafts and boreholes, water bearing strata that is intersected by development drivage/ subsidence from caving and abandoned mines. There have been three water inundations in Australian underground mines producing fatalities since commencement of mining; Creswick Gold Mine 1882, Emu Mine 1989 and the Gretley Colliery in 1996, in which old flooded workings were intersected. In Queensland longwall mines, nuisance water is more often the result of flow from mining equipment (sprays and hydraulic fluid) and subsidence cracking intersecting aquifers generally being more prevalent as workings become shallower. Heavy wet season rain can flood opencuts that provide highwall access for underground workings and connect with aquifers fractured from subsidence.

Bulkheads must have long-term structural integrity while significantly reducing the risk of inundation for miners. However in contrast construction of plugs may be required in urgent and emergency situations where there is little time for rigorous analysis and investigation. In most cases pressure requirements for explosion rated seals will exceed pressure requirements of a water holding bulkhead. Often the impounding of water behind 138 kPa (20 psi) explosion rated (Type C) seals is unintentional. There is a belief that 20 psi seals can withstand lower, long-term hydraulic pressure. This frequently not the case and will be discussed later in the paper.

Dams used for storing water for particle settling before pumping to the surface are normally less than the height of the roadway which means that through ventilation is possible. Sometimes they have access through the structure for desilting.

The Queensland coal mining industry has now had nine years of experience in the routine sealing of parts of mines pursuant to legislation (Queensland, 2001). It is appropriate that our industry continually

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reviews the effectiveness with which it seals parts of the mine and that we continually seek improvement in our sealing processes. In the proposed new Queensland Mines Inspectorate standard and process “A guide for Inspectors and mine management to efficiently expedite the sealing process” there is no mention of the impoundment of water by seals.

Published applications to control water in coal mines date back to the 1930s when empirical design principles evolved. In the 1980s and 90s engineers started to use finite element (F.E) numerical analysis for developing plug and bulkhead designs more effectively as hardware development caught up with sophisticated software programs. The results from using modern FE software that can predict progressive time-related damage in dam structures and surrounding roadway materials will be presented. Most software being used by design engineers provides information on stresses and strain in the material models and some strain softening data which is suitable for static load analysis in bulkhead design but limited for considering the effects of transient (dynamic) explosion loads.

Design guidelines for the construction of bulkheads in coal mines have only recently been introduced in the United States in the form of publication IC 9506 (Harteis, et al, 2005). Previously the USBM had published IC9020 “Design of Bulkheads for Controlling Water in Underground Mines” that presented three methods for designing bulkheads to impound water underground. A British publication by the Health and Safety Executive published in July 2005 had been prompted by the inundation that occurred at Longannet Mine (Scotland) in 2002 which resulted in the loss of the mine. The subsequent inquiry concluded that some failure occurred in the vicinity of two high pressure plug seals, implying a possible failure of the surrounding strata. This guide provides information for the design and construction of water-tight plugs in working mines and relates only to parallel plug cement, concrete or grout type plugs. This incident prompted British Mine owners to carefully consider the inrush potential at their mines. Since 1982, water holding dams in British coal mines have been constructed to an industry code of practice. This code of practice has made the assumption that provided a dam is of adequate length, the failure mechanism would be at the dam/rock interface. However this assumption is not necessary for all dam designs at lower water heads.

A review of existing bulkhead designs indicates that a variety of material and construction techniques have been used in the past. They range from low strength monolithic grout plugs, to high strength steel reinforced concrete slabs and include a composite design with masonry wall formwork containing a core of sized limestone aggregate and polyurethane foam. Plug designs are basic and require single and universally accepted design parameters, whereas composite designs require the expertise of a certified structural engineer and in particular for Queensland a locally registered engineer. For permanent high head plugs, concrete have been used because of the resistance concrete provides to chemical attack and its low solubility.

**SOURCES OF WATER AND LIKELY IMPACT**

A variety of geological and geographic conditions can be a source of water and in extreme cases inundation causing environmental damage along with the potential for severe loss of life and property. On the surface, lakes, ponds, streams and rivers are potential sources of water to underground workings. In the Newcastle region mines have been driven under lakes (Wyee Mine) and the ocean (Burwood Colliery) where legislation restricts depth of cover and panel width to depth ratios. In 2003 MSHA (MSHA, 2003) developed guidelines for evaluation of sites with potential for a breakthrough into an underground operation.

Vutukuri and Singh (1995) after a review of past inundations classified inundations as (i) event controlled (ii) accidental and (iii) spontaneous inundation. An example of event controlled inundation would be where, during longwalling, periodic roof falls and strata relaxation intersect a confined aquifer. This is characterized by a sudden increase of water flow over and above background flows followed by exponential flow decline. The driving force for a breakthrough is the pressure gradient between the impoundment and the underground workings where the stress on the underlying strata is increased and seepage may cause erosion or piping. A spontaneous inrush can occur when a solution cavity is intersected such as those found in Karst (limestone) formations associated with coal measures.

Where underground mines have been driven off the highwall of an opencut, a 100 year rain event could cause severe flooding. With the help of surface contour maps, flow paths and water storage ability should be given careful consideration in particular potential breaches due to storage dam failure.
Traditional mining countries have flooded old abandoned workings in adjacent and overlying/underlying coal seams, a source of stored water. An example of this is the impoundment of stored water with bulkheads at Racoon No. 3 Mine in the United States which was separated from the down dip and actively worked Meigs No. 31 Mine. There was a bulkhead failure with the likely mechanism erosion (or piping) along the concrete interface at the base of the bulkhead. Gretley inundation caused the death of four workers in 2004 when an old adjacent shaft was accidently breached, an incident caused by misinterpretation of old plans. For this reason during risk assessment or preparation of hazard management plans, the capacity of the underground drainage system and likely flow paths should always be evaluated.

Most longwall mines in Queensland retreat to the rise and it is necessary to provide bulkheads for impounding and controlling water inflows from longwall equipment and from aquifers that have been breached by upward propagation of subsidence cracking. In situ joint patterns, mining induced fractures and separation of bedding planes contribute flow paths. The maximum reported inflow in Central Queensland coal mines was 400 L/s recorded at Southern Colliery in December 1995. Generally it has been reported (Klenowski, 2000) that larger inflows in longwalls were experienced as workings became shallower. It was found at cover depths less than 120 m, additional protection might include goaf based borehole submersible pumps for hazard elimination.

Considerable experience has been gained (Gale, 2006) by comparing field measurements with computer modelling of rock fracture, caving and stress redistribution to predict strata hydraulic conductivity.

Figure 1 (Gale, 2006) illustrates caved panels with a width: depth ratio of above 1.0 have a high probability of connection and inflow and those with width: depth ratios of less than 0.4 have not had any connection. Mines such as Crinum and Kestrel have thick clay beds which have acted as an aquiclude, separating active workings from aquifers.

CONSIDERATION AND TREATMENT OF STRATA SURROUNDING A BULKHEAD

Bulkhead effectiveness is impacted by the properties and condition of the surrounding strata. Bulkheads can fail if the material that they are anchored into or keyed into is not strong enough to resist the applied pressure and pressure from water seeping around the structure. When failure has occurred, leakage in bulkheads has generally been through the surrounding strata or along the strata/bulkhead interface. At Meigs No. 31 Mine in July 1993 it was thought a piping or erosion failure occurred along the bulkhead/weak floor contact causing inundation of workings (Harteis and Dolinar, 1993). Keying extends into the stress relief zone around the roadway removing yielded strata. The hydraulic pressure head created by impoundment must be contained by the surrounding strata. Consideration should be given of the possibility that water could find a leakage path some distance from the seals and this has been experienced along the coal seam-floor contact. Most roadways driven in coal suffer some form of overbreak that is often influenced by joint sets, coal cleat, cover depth and drivage direction in conjunction with the magnitude and orientation of the principal stress. For this reason a plan should be prepared showing details of immediate roof, ribs and floor including all strata that could be affected by
water, structural geology (faults) in the general locality of proposed seal sites. Water progressively reduces the strength of geological features such as joints and faults. Durability testing may reveal the presence of materials that could degrade e.g. clay bands or partings that become a conduit for water. Bulkheads are required to be sited in competent strata and all materials prone to erosion should be removed from the site. It is important to consider the intended history of the bulkhead locations whereby future mining events could redistribute the stresses within the strata surrounding the bulkhead.

Factors such as the complex interaction of the rock mass and redistribution of stresses due to an imposed water load, the unknown influence of adjacent geological features has led to using large analytical design safety factors for bulkhead design e.g. safety factor of four is commonly used in Queensland mines. If a 30 m water head bulkhead is designed with a safety factor of four then this implies that the 120 m head would require the strata in the roof and floor to cater for a pressure of 1.2 MPa. This considerable load could conceivably fail thinly laminated coal measures by buckling. It has been suggested that a series of open holes placed centrally within the roadway could relieve such pressure and for the purposes of pressure equalization within the strata large and frequent changes in water holding levels are not encouraged. Stability calculations for adjacent pillars are often considered when there is a requirement to impound large heads of water.

In order to investigate bulkhead design the following site properties require investigation.

- Geological cross-section at or near the site showing the UCS and thickness of coal measure plies, rock quality designation and consideration of coal cleat and joint sets. Depending on the bulkhead design, rock properties at the seal boundary contacts, such as shear strength and for shotcrete slab designs the strata Young's Modulus, influences the resistance of the bulkhead to horizontal load from water impoundment. As a thinner bulkhead flexes under load it pushes outwards into the strata.

- The proneness of coal measure plies to weathering/swelling and softening in the case of clay bands i.e. measure slake durability. Consider whether erosion and piping can occur. An example of this is the Cow Pad band present in the Whybrow Seam (Hunter Valley Coal Measures) where the compressive strength is highly dependent on moisture content.

- The effect of possible future stress changes on the enclosing strata and the likely convergence in roadways. The depth of yield zones surrounding the roadway will influence the bulkheads leakage path resistance.

Bulkheads can fail if the material that they are anchored into or keyed into is not strong enough to resist the applied pressure from water seeping around the bulkhead. The permeability (MSHA, 2003) of the rock mass centres on the nature of the discontinuities present, that is, the joints, cleats, fractures, shears, faults, and bedding planes. Intact rock permeability is generally much less than that of the actual rock mass. Note that joint filling material deposited when water has been flowing through the strata, will inhibit grout penetration and distribution.

Improving the rock mass in cut-throughs involves design of additional supplemental support such as rib/roof bolts, shotcreted linings, grouted cable tendons, Link N Locks, Rocprops or Burrell Cans providing long-term roadway stability. Typically there has been greater focus on reinforcing gateroads in longwall mines and often gateroad direction is chosen to minimize the impact of the difference between the major principal stress $\sigma_1$ and minor principal stress $\sigma_3$ (often close to vertical). Sealing the ribs with a cement based shotcrete (typically 50 mm thickness, fibre reinforced to increase energy absorption) protects the skin of the roadway as grouting is not effective in the immediate skin. Shotcrete will also provide some corrosion protection for installed steel mesh and bolts. The effectiveness of passive support will be influenced by how rapidly it can react to and effectively resist convergence i.e. its initial stiffness and ability to deform. It is pointless if the roadway span has failed before the support takes significant load. Consider how impounded water will affect the long-term stability of these supports. These measures resist convergence reducing the depth of yield zones, reducing potential leakage paths and effective roadway span. Seal designs should also consider the effects of floor heave where bed separation occurs in thinly laminated strata and tendon reinforcement can be used.

Preferred practise is to construct bulkheads within a roadway which will not be affected by changes in vertical stress; however this is not always possible when sealing longwall gateroads where chain pillars experience increased vertical load from abutments causing further breakage and dilation of surrounding strata.
The primary objective of grouting activity is to decrease the flow of water and gases through the excavation damage zone once the bulkheads are in place. Pre-injection is going to be more effective and far less costly because the joints and fractures are being sealed before being dilated by water pressure which also increased the potential for grout washout. It is important to consider that a bulkhead under increasing water pressure will transfer load to the strata. Soft coal strata acting as a foundation for the bulkhead could suffer further damage and dilation unless it has been reinforced.

Grout injection can be split into two categories or phases, curtain grouting and structural or contact grouting. Curtain grouting involves drilling a pattern of radial holes in the vertical plane, out from the roadway (typically of three metre length) forming an evenly spaced series of rings each which is grouted from the bottom up, filling voids in the strata surrounding the bulkhead with overlapping grout zones. Contact grouting involves sealing gaps at the bulkhead/strata interface. Often porous injection hoses are attached to the strata and sealed into the bulkhead during construction in case further injection is necessary. Where holes tend to collapse, downstage (Klenowski, 2000) grouting has been used where the hole is drilled in a short distance grouted and allowed to set for 24 h. Then the holes are extended an additional depth and the process repeated. Grouting methodology and techniques are more thoroughly covered in ACARP Project C5016 where cement based grouting phases are described including maximum recommended injection pressures and lagoon water pressure testing of strata.

Sealing of bulkhead sites can be undertaken with cement based grouts and two-component organic resins such as PUR which used to treat finely fissured rock masses. PUR acts like a glue resisting any further strata movement whereas injected cements are more of a gap filler used in bulk injection. In more permeable strata a useful strategy is to inject a curtain around the immediate roadway with PUR then drill through and inject behind this liner with lower cost cement based grouts. As PUR sets rapidly the pressure in the grouted holes does not have to be maintained by stopcocks/valves in the holes as is required with slow setting cement based grouts. Table 1 compares both systems.

<table>
<thead>
<tr>
<th></th>
<th>Polyurethane</th>
<th>Stratabinder Slowset</th>
</tr>
</thead>
<tbody>
<tr>
<td>Solution</td>
<td>Suspension in water</td>
<td>W:P normally 0.5-1:1</td>
</tr>
<tr>
<td>Expands/foams</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Long life &gt; 50 years</td>
<td></td>
<td>Durability lower at W:P of 1.0</td>
</tr>
<tr>
<td>Resistant to chemical attack</td>
<td></td>
<td>Lower resistance to CO₂, acid water</td>
</tr>
<tr>
<td>Self inject</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Non shrink</td>
<td>Some shrinkage due to W:P =1.0</td>
<td></td>
</tr>
<tr>
<td>High bond strength</td>
<td>Moderate bond strength</td>
<td></td>
</tr>
<tr>
<td>Resistant to ground movement</td>
<td>More of a gap filler</td>
<td></td>
</tr>
<tr>
<td>Penetrate finer voids-limited by viscosity</td>
<td>Particle size limits penetration</td>
<td></td>
</tr>
<tr>
<td>Instant strength gain</td>
<td>Slower strength development</td>
<td></td>
</tr>
<tr>
<td>Used for stopping high pressure water inflows</td>
<td>Not suitable for injection of strata where there is water migration</td>
<td></td>
</tr>
</tbody>
</table>

Table 1 - Comparison of PUR with Stratabinder Slowset cement based grout

Figure 2 - Injection guidelines (Source: 5th Annual Tunnelling Workshop)
DESIGN OF PLUGS AND BULKHEADS

Structural design considerations

There are several existing simplified methods that can be used to design ventilation seals and bulkheads. However, only high-fidelity physics-based computer simulations are able to predict the results from physical testing of mine-based seals in a most realistic way. Explosion testing is still extensively used to test existing designs and NIOSH’s relatively new hydraulic test facility provides a cost-effective method to develop stress-strain response data using a water load as an alternative to full-scale explosion testing (Sapco, et al., 2008). In order to provide a “fit for purpose” seal/bulkhead design the conditions that the seal will be subject to for its intended life must be defined.

Imposed stress changes on ventilation seals and bulkheads

Imposed stress changes that may affect the structural integrity of the seal (and surrounding strata) and hence its ability to safely resist an explosion and a hydraulic load have been discussed by Mutton and Remennikov (2010) and are summarised as being caused by:

- Abutment loads;
- Crushing of chain pillars;
- Breaching of aquifers;
- Strength of existing supports, and
- Movement along joints and faults caused by stress changes.

TYPES OF BULKHEADS AND PLUGS

Many types of water holding structures have been constructed in mines; however, they fall mainly into two types—bulkheads or plates and plugs—which could collectively be described as barriers. Slabs or plate bulkheads have length or thickness (along the drive axis) less than their height and strength is limited by flexure for thinner structure and shear resistance along the drive wall. Virtually all Australian bulkhead designs in coal mines are in this category and rarely designed for water heads over 30 metres. Plugs, whose lengths are greater than the roadway dimensions are limited by shear resistance at the strata contact. Some interesting designs in hard rock have involved laying down mullock from development firing and grouting with either a mortar (mortar intrusion technique used in South Africa) or a high yield grout such as FB200 (Mutton, et al., 2010) to form a plug. The designs of water retaining structures used in underground mines in Australia coal mines are summarised as:

- Shotcrete bulkheads or slabs (notched or keyed).
- Plug type structures using a high yield grout such as FB200 or Tekblend.
- Polyurethane and aggregate core bulkheads using dry stack concrete block containment walls.
- Plugs for surface portal sealing using flyash/cement blends (Note: Flyash/cement mix designs have been used to build emergency bulkheads to seal Paradise No. 9 Mine, Kentucky in the United States.)
- Gypsum based plaster (water resistant) bulkheads.

However globally the primary bulkhead designs for underground coal mines (Garrett and Campbell, 1958) are tapered plugs, parallel plugs and notched slabs summarised in Figure 3.

Plate bulkheads

These lower pressure bulkheads are typically constructed of shotcrete or concrete placed within formwork, however, they can be sprayed against backing formwork using the dry shotcrete method (cement based shotcretes and Gypsum based plaster). Typically arching type behaviour sees the bulkhead resist hydraulic loads by the strength of the concrete and resistance to buckling by the reinforcing steel anchored into the strata. Increasing loads will see the bulkhead flex and push outward into the strata. Often this type of structure is keyed into the strata improving shear resistance and reducing flexure.
Plugs

These are constructed where high pressures may develop and can be constructed with plain (unreinforced) concrete, grouted aggregate or flyash/cement mixes. Often the wall is tapered downstream to enhance internal arching and strength. Often plugs have been constructed for emergency situations to prevent inundation from for instance an unmanageable inflow of groundwater where potentially a hydraulic head may develop to the surface.

**HIGH-FIDELITY PHYSICS-BASED MODELLING OF SEALS**

LS-DYNA, a general purpose transient dynamic finite element program (LS-DYNA, 2008) was used to develop the finite element models in this study. LS-DYNA is used to solve multi-physics problems including solid mechanics, heat transfer, and fluid dynamics either as separate phenomena or as coupled physics, e.g., thermal stress or fluid structure interaction. LS-DYNA is an industry accepted dynamic first-principle based code for analysis of structures under extreme loads generated by blast and impact events with the ability to compute large deformations due to flexure, shear, and material failure.

**Model description**

As an example, a shotcrete seal which is 3.4 m high and 300 mm thick was analysed. Due to the symmetry of the seal, the boundary conditions, and the loading about the central vertical plane, the model includes only one half of the seal allowing for a model width of 2.7 m. The model includes roof and floor skeleton bolts (650 MPa steel) of 21.7 mm diameter that are placed at 600 mm centres around the periphery. The 200 mm deep rib keys are modelled for 300 mm thick seals. The rib keys are modelled with a single row of 1200 mm long bolts with 600 mm tails protruding and 600 mm full encapsulation. To simulate the seal-rock interfaces, floor, ribs and roof are explicitly modelled as large solid bodies surrounding the seal. The overall thickness of the floor and the roof in the model is 2.5 m. The Meshblock seals have 1.8 m of coal in the roof and 0.6 m of coal in the floor. The remaining depth is filled with the rock materials. Figure 4 shows the components of the seal model used in this study.

In the finite element model, solid elements with a single integration point were used to model the shotcrete seal and the surrounding coal and rock materials. Overall model dimensions and the sizes of finite elements were determined from a mesh convergence study. The mesh convergence study included a number of runs of the model with variable model dimensions and increasing levels of mesh refinement. In the final model, the concrete seal was modelled with 50 mm cube solid elements, and the surrounding rock was modelled with 250 mm cube solid elements.

Beam elements were used for the skeleton bolts in the ribs, roof and floor. Each beam element shared two of the solid element nodes to model the strain compatibility between the steel and the concrete. As a result, slip between the steel reinforcement and the concrete was included explicitly in the model. Slip occurs as a function of the failure of the concrete attached to the reinforcing bars. Reinforcing bars were extended 600 mm into the ribs, roof and floor to provide sufficient anchorage length. The bond between
the steel bars and the rock was modelled using constrained conditions provided by LS-DYNA for connecting meshes of dissimilar densities.

Figure 5 shows the finite element model of the rib keys. The rib key is modelled by extending the concrete seal model into the body of the coal ribs. Interaction between the key and ribs is simulated using surface to surface contact surfaces. The full model of the seal consists of 127 050 nodes, 336 beams, and 114 000 solid elements.

![Figure 4 - Model of roadway and strata enclosing seal](image1.png)

![Figure 5 - Modelling the keys for the ribs and the skeleton bolts](image2.png)

**Material models**

The concrete model employed for modelling the shotcrete seal was model 159 in LS-DYNA implemented in keyword format as MAT_CSCM_CONCRETE for Continuous Surface Cap Model. The model formulation includes a smooth and continuous intersection between the failure surface and hardening cap. The model includes isotropic constitutive equations, yield and hardening surfaces and damage formulations to simulate softening and stiffness reduction. A rate effects formulation increases strength with strain rate. The model has been thoroughly tested by several US Governmental agencies (Murray and Lewis, 1995; Murray, 2007) for predicting damage in concrete under severe impact and blast loads, which has demonstrated its reliability and accuracy. Default input values for model parameters were used in this study. Default material parameters are generated by the model based on the specification of the unconfined compression strength. In this study, the unconfined compression strength of 50 MPa was used based on the test data from testing of Hanson shotcrete in Queensland.

Roof, floor and ribs were modelled using Material Type 173 based on Mohr-Coulomb criterion in LS-DYNA. The material has a Mohr Coulomb yield surface, given by \( \tau_{\text{max}} = C + \sigma_n \tan(\theta) \), where \( \tau_{\text{max}} \) is maximum shear stress on any plane, \( \sigma_n \) is normal stress on that plane, \( C \) is cohesion, \( \theta \) is friction angle. The tensile strength is given by \( \sigma_{\text{max}} = C/\tan(\theta) \). After the material reaches its tensile strength, further
tensile straining leads to volumetric voiding. Material 173 is intended to represent soils, rock and other granular materials.

The appropriate material modelling parameters for roof, floor and ribs are summarised in Table 1 for the boundary roadway conditions investigated in this study. It should be noted that coal mine strata are variable in geomechanical properties with adjustments required when considering bulk properties as compared to laboratory test results of intact cored specimens. Coal shows (directional) compressive strength variations due to variable cleat, moisture and gas content changes, stone partings, varying materials shown in laminae found in a vertical seam section and changing ash content. Table 2 material properties represent values that have been used when modelling mine strata for ground support and chain pillar design.

Table 2 - Material properties for models of roof, floor and ribs

<table>
<thead>
<tr>
<th>Boundary Roadway Condition</th>
<th>Material</th>
<th>Young’s Modulus (MPa)</th>
<th>Poisson’s Ratio</th>
<th>Friction Angle (deg)</th>
<th>Cohesion (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Roof</td>
<td>Coal</td>
<td>3,000</td>
<td>0.4</td>
<td>30</td>
<td>1.0</td>
</tr>
<tr>
<td></td>
<td>Stone</td>
<td>5,000</td>
<td>0.2</td>
<td>35</td>
<td>5.0</td>
</tr>
<tr>
<td>Floor</td>
<td>Coal</td>
<td>3,000</td>
<td>0.4</td>
<td>30</td>
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<tr>
<td></td>
<td>Stone</td>
<td>5,000</td>
<td>0.2</td>
<td>35</td>
<td>5.0</td>
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<tr>
<td>Ribs</td>
<td>Coal</td>
<td>3,000</td>
<td>0.4</td>
<td>30</td>
<td>1.0</td>
</tr>
</tbody>
</table>

Predictions of response of the 500-mm Meshblock seal to water head pressure

Final longwall seals are required as structural bulkheads, acting as water tight dams capable of withstanding the maximum hydrostatic head that may develop as a result of flooding sealed areas. Based on the finite element model shown in Figures 4 and 5, and the loading and material properties described above, non-linear transient dynamic analyses were carried out for the example Meshblock seal design. Crack patterns for the seal are visualised using the contour plots representing damage levels from zero to one calculated by the concrete model. A contour value of zero indicates no damage, so concrete strength and stiffness are those originally specified as input values. A contour value of one indicates maximum damage and severe cracking, in which the concrete strength and stiffness are reduced to zero.

The 500-mm Meshblock seal was analysed to evaluate its resistance to hydrostatic pressure due to a 10-m, 15-m, and 20-m water head. Hydrostatic pressure was applied as a pressure load on one face of the seal model varying from 65 kPa at the roof level to 98 kPa at the floor level for a 10 m water head, from 113 kPa to 147 kPa for a 15 m water head, and from 163 kPa to 196 kPa for a 20 m water head.

The pressure load was applied as gradually increasing from zero to the maximum value within a sufficient period of time to simulate a quasi-static response of the seal. Analysis results are presented in Figures 6 to 9.

Figure 6 - Concrete damage contours in 500 mm Meshblock seal under 10-m water head pressures
Figure 6 demonstrates that the 500-mm seal will not experience cracking or concrete damage under the 10-m water head. Minor concrete cracking will develop only around the bolts in the ribs, floor and roof. The seal will experience maximum horizontal deformation of 0.6 mm.

Figure 7 shows that the 500-mm seal will experience damage caused by minor to medium sized cracks under the 15-m water head. Minor and medium size cracks will develop mostly around the bolts in the ribs, floor and roof as depicted in Figure 7. Maximum horizontal deformation will be about 2.6 mm.

Figure 8 demonstrates that the 500-mm seal will experience damage caused by medium sized cracks located along the mid-height of the seal to the depth of about 300 mm under the 20-m water head. Medium size cracking will also develop near the bolts in the ribs, floor and roof as depicted in Figure 8.

Contours of horizontal deformations for the Meshblock seal under the 20-m waterhead are shown in Figure 9. It can be seen that the maximum displacement in the seal under the 20-m water head pressure is about 3.7 mm.
CONCLUSIONS

The practice of constructing bulkheads in underground mines to impound water is becoming increasingly important. Bulkhead failures can cause catastrophic flooding putting the workers at risk with loss of infrastructure and assets. Surface flooding due to large rain events can have an impact on underground operations if there is limited surface storage. It is important to have a systematic approach to the control of water inflows underground with adequate time given to planning of the dewatering system, consideration of the risks and the provision of contingency plans. Detailed construction plans are required for each bulkhead with all phases of the construction audited and material samples strength tested for compliance. Sometimes in order to prevent inundation it is necessary to construct emergency bulkheads without the normal time to prepare the sites and check geological conditions.

Strata properties at the sites must be determined and the effect that long term impoundment of water has on the containing strata must be understood. Because of the layered nature of coal measures including the variability in geomechanical properties, presence of joints and adjacent geological features that may affect bulkhead stability, it is necessary to design and implement an injection program with suitable materials to strengthen the surrounding rock mass and block potential leakage paths. Having built the bulkhead(s) it is necessary to monitor the sites including checking the hydraulic heads that the seals are being subject to and to ensure that an effective goaf dewatering system is maintained with sufficient reserves for storage.

One such seal, Meshblock, introduced into Australian mines in 1994 is constructed from cement based shotcretes. Meshblock has been subjected to explosion test programs with outcomes previously summarised in an engineering model. Further design information is required to determine how these seals react under hydraulic loads and what safety factors over failure should be used.

A high-fidelity physics based finite element model for the explosion rated Meshblock ventilation seals was developed. The model is suitable for computing both static and dynamic responses of ventilation seals in coal mines subject to water head pressure, explosion loading, convergence loading and other possible loadings. The seal model includes the concrete material model that incorporates many important features of concrete behaviour, such as tensile fracture energy, shear dilation, effects of confinement, and invariant failure surfaces. Damage metrics is used to gauge the evolution of the concrete’s behaviour from elastic to elasto-plastic, and to softening or fracture.

Numerical modelling and simulation of the seals can be undertaken in stages to determine their resistance to the combined effects of water head pressure, explosion loads, and roof to floor convergence acting at different time instances. In this way, complex scenarios of extreme loading events...
(e.g. explosion followed by flooding) can be investigated at the design stage. Also, detailed investigation of the interface stresses between the seal and the surrounding strata can provide important information for the grouting program for seal construction.

ACKNOWLEDGMENT

The authors wish to thank Minova Australia for permission to present this paper and gratefully acknowledge the assistance and advice provided by various mine operators, and consultants during the evaluation of seal sites, provision of suitable materials and design and construction of bulkheads.

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INFLUENCE OF TEMPERATURE ON THE GAS CONTENT OF COAL AND SORPTION MODELLING

Lei Zhang, Naj Aziz, Ting Ren and Zhongwei Wang

ABSTRACT: An experimental study was undertaken to examine the sorption and desorption characteristics of coal at temperatures of 35 °C, 45 °C and 55 °C. The study focused on the effect of changes in temperature and coal particle sizes on gas sorption and desorption characteristics. The coal size used ranged from fragmented coals, 16 mm, 8 mm, 2.4 mm, powdered coal of 150 µm and 54 mm core samples. The samples were tested in pressure vessels, known as bombs, and charged with CO₂ gas at different pressure levels up to a maximum of 4000 kPa. It was found that temperature was a significant influence the sorption and desorption behaviour. The degree of hysteresis phenomenon was found to be influenced by the coal surface area as well as temperature. The Langmuir equation was used to model CO₂ sorption and desorption. Both Langmuir volume and Langmuir pressure were assessed in relation to the changes at different temperatures and was found to tally well with the experimental results.

INTRODUCTION

Gas sorption in coal has been studied over the years by many researchers (Busch, et al., 2003; Harpalani, et al., 2006; Day, et al., 2008; Florentin, et al., 2009), and in recent years, the coal sorption characteristics at high temperatures, beyond the normal temperature of 20 ~ 29 °C, have been identified as an attractive option which can shed light on the behaviour of gas sorption in and desorption from coal in situ, especially when mining is taking place at increasing depth.

Lama and Bodziony (1996) reported that the term “sorption” consists of two parts: adsorption and absorption. Adsorption refers to the accumulation of gas on the surfaces of pores and cracks and absorption means the penetration of gas into the internal structure of coal. Sorption is an exothermic process and is opposite to desorption, which is endothermic. Studies reported by Moffat and Weale, 1955; Yang and Saunders, 1985; Stevenson et al., 1991, cited by Lama and Bodziony (1996), indicated that the heat of sorption is less than two or three times the heat of desorption/ vaporisation.

Siemons and Busch (2007) measured CO₂ sorption isotherms on both dry and moist coals of various ranks from coal basins from around the world and these measurements were made in temperature of 45 °C. Day et al. (2008) carried out experiments on supercritical gas sorption of carbon dioxide on moist coals at temperatures of 21 °C and 55 °C and pressures up to 20 MPa. The differences of gas content due to different temperatures were not compared and the samples for the experiment were prepared by crushing and screening fresh air-dried lumps of coal to a particle size range of 0.5 ~ 1.0 mm.

Florentin et al. (2009) carried out sorption tests of coal with different particle sizes at 24 °C and in different gases, and concluded that adsorption of CO₂ was typically the highest and that of CH₄ was lowest and that the gas content of coal was dependent strongly on gas type, sorption time and particle size. But the effect of coal gas content for different temperatures was not discussed in the paper.

Harpalani et al. (2006) carried out sorption tests on USA coals at temperatures of 23.5 °C and 45 °C in situ temperatures. The paper discussed the results of the sorption study of Methane and CO₂ in a single gas environment. Furthermore, the experimental data were modelled using different models including Langmuir and they found that, for certain gas, all the models performed satisfactorily with reasonable accuracy.
EXPERIMENTAL

Apparatus

The indirect gravimetric method was used to calculate the volume of gas adsorbed and desorbed from coal. The gas sorption apparatus used in this study and shown in Figure 1 was described previously by Lama and Bartosiewicz (1982); Aziz and Ming-Li (1999) and by Aziz and Florentin (2009). In this apparatus, each vessel, known as “bomb”, has its own pressure transducer so that the sorption process and changes in bomb pressure can be readily determined. The equipment has been further modified to accommodate increases in temperature up to 100°C. The addition of a heat isolation jacket outside the water bath as well as the insulation cover can help the bomb to maintain the desired experiment temperature with an accuracy of 0.1 °C.

![Figure 1 - Schematic diagram of the gas sorption apparatus](image)

Coal sample preparation

The coal samples used in this sorption study were collected from longwall panel 520, Area 5, West Cliff Colliery. Details of the coal samples are shown in Table 1 and Table 2 (Saghafi and Roberts, 2008). The coal core samples were prepared according to the International standard of rock core sample preparation (ISRM, 1981). In addition to coal cores, larger coal lumps were freshly dug out of the development headings. They were wrapped in plastic sheets and taken to the laboratory where they were immersed in water tanks to minimise oxidation and adverse environmental effects. Samples of the collected coals were then crushed and sieved to obtain the appropriate particle sizes for the test. In addition to 54 mm core samples, coal fragments of 16 mm, 8 mm, 2.4 mm and coal powder, 150 µm were used in this study. All samples were oven dried at 105 °C for 18 hrs and then maintained at 25 °C for acclimatisation prior to testing.

<table>
<thead>
<tr>
<th>Sample Code</th>
<th>Depth (m)</th>
<th>Moisture (%)</th>
<th>Volatile Matter (%)</th>
<th>Fixed Carbon (%)</th>
<th>Ash Yield (%)</th>
<th>Volatile Matter (%daf)</th>
<th>Coal Density</th>
</tr>
</thead>
<tbody>
<tr>
<td>520</td>
<td>450</td>
<td>1.3</td>
<td>21.7</td>
<td>71.4</td>
<td>5.6</td>
<td>23.3</td>
<td>1.43</td>
</tr>
</tbody>
</table>

Table 1 - Coal density and proximate analysis

<table>
<thead>
<tr>
<th>Sample Code</th>
<th>Vitrinite Reflectance (%)</th>
<th>Maceral (%)</th>
<th>Maceral (% mineral free)</th>
</tr>
</thead>
<tbody>
<tr>
<td>520</td>
<td>1.28</td>
<td>Vitrinite</td>
<td>41.6</td>
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<td>Liptinite</td>
<td>0.1</td>
</tr>
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<td></td>
<td></td>
<td>Inertinite</td>
<td>55.3</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Mineral</td>
<td>3.0</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Vitrinite</td>
<td>42.9</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Liptinite</td>
<td>0.1</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Inertinite</td>
<td>57.0</td>
</tr>
</tbody>
</table>

Table 2 - Coal petrography
Prior to testing the moisture content of the samples was determined in accordance with the Australian standard (AS, 2002) for the determination of the moisture content of coal. All samples were then enclosed in pressure bombs and subjected systematically to CO₂ gas pressurisation at various temperatures of 35 °C, 45 °C and 55 °C respectively. The level of gas pressurisation of the samples were carried out initially in 500 kPa steps until reaching 1000 kPa and later it increased in 1000 kPa steps until reaching a maximum of 4000 kPa.

RESULTS AND ANALYSIS

Sorption test of coal samples at 35°C, 45°C and 55°C

Figure 2 shows the variation in the CO₂ pressure decrease with time from the initially charged pressure level of 900 kPa at different temperatures of 35°C, 45°C and 55°C respectively. The variation in the rate of pressure drop during the first 6000 s is different for different coal particle sizes. The rate of pressure drop appears to be affected by the particle size particularly during the first 2500 s of the sorption process. The sharpest drop in gas pressure is in 150 µm samples and the slowest being in 54 mm core sample. However the consistency of the pressure does not hold well over longer periods of time and after 3000 s of sorption time. The 150 µm particle size easily achieves pressure equilibrium in a relatively shorter time period in comparison with other larger coal particle sizes, and irrespective of the temperature level. It is concluded that coal particle size has an important bearing on CO₂ gas sorption in coal, and that the environment temperature appears to influence the speed of the sorption process.

Figure 2 - 900 kPa pressure drop at 35 °C, 45 °C and 55 °C

Figure 3 shows that the adsorption isotherm at three temperatures for different particle sizes. For each particle size coal samples, at every pressure step from 0 to 4 MPa the adsorbed volume of CO₂ is decreased with the increasing temperature step. The sorbed volume achieves the highest level at 35 °C and the lowest at 55 °C.

Figure 4 shows the CO₂ adsorption isotherm for different particle sizes at different temperature levels. It can be deduced from this study that:

- The amount of sorbed volume of gas in coal decreases with increasing temperature.
- With the exception of 54 mm diameter core sample, there is no common trend of variation between sorbed volume of CO₂ gas and particle size, and
- Increased temperature reduces or minimises the influence of particle size on gas sorption.

Figure 3 - CO$_2$ adsorption isotherm in terms of different temperature

Figure 4 - CO$_2$ adsorption isotherm in terms of different coal particle size
It was found that high temperature helped to reduce sorption duration. This is because high temperature stands for the high gas molecule energy, the CO$_2$ molecule with the higher energy can readily reach the adsorption position. There is no clear relationship between the CO$_2$ adsorption volume and coal sample particle size. This finding is agreement with previous findings Florentin, et al. (2009).

**Desorption test of coal samples at 35 °C, 45 °C and 55 °C**

Figure 5 represents the desorption isotherms of five particle size coal sample at 35 °C, 45 °C and 55 °C, the desorption hysteresis of CO$_2$ sorption on coals at each temperature is clear for every particle size tested in this experiment that is the desorption isotherms lie above the sorption isotherms. It is found that the desorption isotherm is inconsistent with regard to coal particle size. However it is clear that at each temperature level the desorption hysteresis of the 150 µm size in Bomb 1G is relatively smaller than that of the other larger particle sizes because of the smaller amount of coal matrix. Also, higher temperatures caused a reduction in the degree of desorption hysteresis.

![CO$_2$ Adsorption and Desorption Isotherm at 35°C](image1)

![CO$_2$ Adsorption and Desorption Isotherm at 45°C](image2)

![CO$_2$ Adsorption and Desorption Isotherm at 55°C](image3)

Figure 5 - CO$_2$ desorption isotherm in terms of different coal particle size

According to (Harpalani, et al., 2006), the desorption hysteresis on coal or any adsorbent may occur due to two different reasons, which are the changes in the adsorbent properties/structures and /or the capillary condensation in the adsorbent micropores. Busch et al. (2003) observed significantly positive deviations for CO$_2$ desorption curves, attributed to a metastable sorbent-sorbate system, which prevents the release of gas to the extent corresponding to the thermodynamically equilibrium value with decrease in pressure during desorption. This was also confirmed by Ozdemir et al. (2004), where the positive deviation of CO$_2$ desorption was attributed to the swelling of the coal matrix. Shrinkage/swelling of coal matrix is believed to be associated with the desorption/adsorption process.

**Langmuir modelling of sorption data at 35 °C, 45 °C and 55 °C**

The Langmuir model, based on the concept of dynamic equilibrium between the rates of adsorption of gas on the solid and desorption from the solid surface is used to model the CO$_2$ sorption data. In order to
obtain the values of the Langmuir parameters, the experimental pressure (P) versus sorbed volume (P/V) is plotted and the best fit straight-line found as shown in Figure 6.

![Langmuir Modeling of CO₂ Adsorption Data at 35°C](image1)

![Langmuir Modeling of CO₂ Adsorption Data at 45°C](image2)

![Langmuir Modeling of CO₂ Adsorption Data at 55°C](image3)

**Figure 6 - Langmuir modelling of CO₂ adsorption data at 35 °C, 45 °C and 55 °C**

As shown in the Langmuir equation (Equation 1), the inverse of the slope of the Langmuir plot provided the Langmuir volume (V_L). The product of the Langmuir volume within the Y - intercept gave the Langmuir pressure P. V_L is the maximum monolayer capacity and when the sorbed volume is half of the Langmuir volume, the pressure value is referred to as the Langmuir pressure P_L. Both V_L and P_L are important parameters for economic assessment of Coal Bed Methane (CBM) resources. While V_L is the maximum sorption capacity of the coal, which is the value of gas content at very high pressure, P_L represents the pressure to which the coalbed reservoir has to be depleted to obtain a 50 % recovery (Harpalani, Prusty et al., 2006).

\[
\frac{P}{V} = \frac{1}{V_L} \times P + \frac{P_L}{V_L}
\]

(1)

The Langmuir parameters at 35 °C are described, in Figure 7, the Langmuir volume of adsorption is larger than that of desorption for each particle size. Also the Langmuir pressure of adsorption is larger than desorption, it means that in sorption, higher gas pressure is needed to achieve the half of Langmuir volume compared with desorption. Tests at temperatures of 45 °C and 55 °C show the same results as at 35 °C. In Figure 7, there is no clear relationship between V_L and P_L parameters and the coal particle sizes and this is in agreement with the experimental result.

From Figure 8, it is clear that:

- There is a gradual reduction of Langmuir volume with respect to increasing temperature for both sorption and desorption;
- The Langmuir volume is inconsistent with coal particle sizes with respect to the temperatures for both sorption and desorption, and
- The Langmuir volume gap is larger at 35 °C in comparison with other high temperatures such as 45 °C and 55 °C.

**Figure 7 - Langmuir parameters of adsorption and desorption at 35 °C**

Figure 8 shows the comparison of Langmuir volume of adsorption and desorption for each of temperature at 35 °C, 45 °C and 55 °C.

**Figure 8 - Langmuir volume of adsorption and desorption at 35 °C, 45 °C and 55 °C**

**CONCLUSIONS**

Higher temperatures reduce the CO$_2$ sorption capacity of coal for different particle sizes, and for each temperature at 35 °C, 45 °C and 55 °C, there is no clear relationship between the CO$_2$ sorption capacity and coal particle size.

The CO$_2$ sorption rate of all the five samples is consistent with their particle sizes particularly at the early stage of sorption, that is the finest sample achieves the quickest sorption speed and hence reduces the sorption duration for all the temperature conditions, this clearly indicates the large surface area of coal attract the CO$_2$ to adsorb first.

During the sorption process, it takes two days for coal to be saturated with CO$_2$ when the coal particle size is larger than 1 mm, however, it takes one day for the micron size. For the desorption process, it is found that it only takes one day to get equilibrium for all the particle sizes tested in this study.

The adsorbed volume of CO$_2$ is decreased with each increasing temperature step, according to the coal adsorption isotherm for a certain particle size at different temperatures, at every pressure point from 0 to 4 MPa. Also it is found that increased temperature reduces or minimises the influence of particle size on gas sorption.
At each temperature the desorption hysteresis of the 150 µm size is apparently smaller than the other larger particle sizes. The degree of desorption hysteresis is reduced by the higher temperatures, Langmuir sorption models of CO₂ at different temperatures of 35 °C, 45 °C and 55 °C is in agreement with the experiment results.

REFERENCES

THE EXPERIMENTAL STUDY OF THE IMPACT ON SUPERCRITICAL CO2 FROM CH4 COMPOSITION IN COAL

Dongmei Wu, Yuanping Cheng

ABSTRACT: Carbon dioxide in a supercritical form can be stored in deep unmineable coal seams in great quantities than in the free state. However, other components of gases in coal seams can affect the storage conditions of the supercritical CO2. Using the critical opalescence of carbon dioxide as a critical point criterion, with the experimental temperature 10~40 °C, and pressure 10 MPa, the critical points of ten groups of CH4-CO2 binary system and CH4-CO2+coal sample ternary system were measured with a high pressure visualization device with a sapphire window. The effects of different contents of CH4 on the supercritical CO2 were studied. The results show that, the critical temperature and critical pressure of the CH4-CO2 binary system varies with CH4 content. When the CH4 component concentration is less than 10%, the critical temperature range of the CO2-CH4 binary system is between the critical temperatures of the two components, it is lower than that of pure CO2 and higher than that of pure CH4. The critical pressure increases with the increase of CH4. After adding coal, CO2-CH4 binary mixture critical opalescence phenomenon still occurs, but the critical temperature is slightly higher than that of the corresponding CH4-CO2 binary system, while the critical pressure is lower than that of the binary system.

INTRODUCTION

Supercritical CO2 means the state over the critical temperature (Tc = 31.02 °C) and pressure (Pc = 7.38 MPa). The self-diffusion coefficient and viscosity of supercritical CO2 are close to that of the gas, it has a similar flow behavior as the gas, similar density to liquid, and more kinetic energy than liquid (Zhang, 2000). Supercritical CO2 can be captured and stored in coal seams because coal has a strong adsorption for it (IPCC, 2005; IPCC, 2007; Reeves, 2004). There are two positive reasons to inject supercritical carbon dioxide into coal seams: 1) it is one of the effective ways to reduce greenhouse gas emissions by storing supercritical carbon dioxide in the deep unmineable coal seams (Yu, et al., 2008; Ye, et al., 2007; Reeves, 2004). 2) During CO2 injection and storage, the efficiency of coalbed methane (ECBM) can be effectively improved (Van Bergen, et al., 2006; Yamaguchi, et al., 2006; Mazzotti, et al., 2009; Mazumder and Wolf, 2008, Gunter, et al., 2004; Day, et al., 2008). But it can also have a negative effect. There are a large number of unmineable coal seams in China, they can capture and store a large quantity of CO2 from industrial production. Once the coal seams are mined in the future, serious coal and gas (CO2) outbursts will be caused. The main gases in coal seams are CH4 and CO2, and the vast majority is CH4, CH4 and coal have great impact on the critical temperature and pressure of CO2. Therefore the study of supercritical parameters of CO2 in coal (critical temperature and critical pressure) and the impacts of other gases on the supercritical CO2 is significant.

In general, the critical parameters of supercritical CO2, CO2 binary and multi-component systems can be measured by the fixed volume high visibility method (Belandria, et al., 2010), variable volume high visibility method (Johannes, et al., 1990; Nieuwoudt, et al., 2002; Rand, et al., 2001); supercritical fluid chromatography (SFC) (Zhao, et al., 1996), and high-performance liquid chromatography (HPLC) (Lee, et al., 1995; Cui, et al., 1991, 1995). There are a few critical parameters of the CO2-CH4 two-phase system, and they were mentioned in Bezanehtak et al., 2002. For example, the CO2-CH4 system P-V-T-X relations around the phase boundaries were researched (Arari, et al., 1971). The pressure range covered was from 2.026 to 15.198 MPa and experimental temperatures were 288 K, 273 K, and 253 K. The values of the critical pressure and temperature for the CO2-CH4 system agreed well with those of Donnelly and Katz (1954). The phase equilibrium composition and critical properties of the CO2-CH4 system were measured at 298.1 K and 301.0 K by an improved high-pressure vapor-liquid equilibrium device (Bian, et al., 1993). Thiery et al. (1996) described that modified SRK EOS are successful for predicting near-critical conditions of nonpolar mixtures of CO2-CH4. The predicted results agreed with the experimental data (Al-Sahhat et al., 1983). Xu et al., (1992) studied the vapor-liquid equilibrium of the CO2-CH4 binary system at 298.36 K using a static high-pressure equilibrium device. The parameters

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10 – 11 February 2011 277
of critical region were measured and the interactive calculation of the experimental data were done with a modified Peng-Robinson equation. The data and model can be used for gas separation process simulation calculations and engineering design. The above research was from the phase equilibrium properties of the CO₂-CH₄ binary system without involving the phase equilibrium state of the presence of coal.

Using the critical opalescence of carbon dioxide as a critical point criterion, the critical points of ten groups of CH₄-CO₂ binary system and CH₄-CO₂-coal sample ternary system were measured in the lab with a high pressure visualization device with a sapphire window. Based on the test data, the effects of different contents of CH₄ and coal samples on the supercritical CO₂ were studied. At the same time, this study provided the basis for testing supercritical CO₂ in the Haishiwan mining area of Yaojie Coal Mine.

The coal samples used for these measurements were collected from coal seam No.2 in the Yaojie Coal Mine, in Gansu province, China. The original carbon dioxide concentration in this mine ranges from 30% to 98%, which experienced violent outbursts of carbon dioxide in 1978 (Zhang, 1992). The currently measured temperature of coal seams in the mine is 31-38 °C, and gas pressure exceeds 7.5 MPa.

**EXPERIMENTAL SECTION**

**Experimental reagent**

The required high purity gas CO₂ (99.999%) and mixed gas 98.16% CO₂+1.84% CH₄; 96.50% CO₂+3.50% CH₄; 95.75% CO₂+4.25% CH₄; 90%CO₂+10%CH₄ were purchased from a Nanjing special gas plant.

**Coal sample preparation**

The crushed samples with moderate levels of metamorphism, were screened and divided into two groups, one of particle size 0.2 to 0.25 mm and the other with 1 to 3 mm particles, dried two hours under 105 °C vacuum. Parameters of the coal sample were determined according to China standards: GB/T212, DL/T 1030-2006 and GB/T 217-1996. Coal density was measured using the automated apparatus Micropore Structure Analysis Apparatus-Pore Sizer 9510, made in the USA (Table 1).

The samples, examined by petrographic methods, were cut parallel to the bedding plane. Petrographic analyses were performed with the use of a Jenapol polarising microscope, equipped with a 50×immersion lens and a 10×microscope eye-piece. Maceral analysis (see Table 1) indicated that the coal samples dominated by vitrinite and inertinite, where the amount of inertinite in coal sample was more than 47%. The moderately ranked coal was brittle and fissured. the vitrinite of the coal samples was mostly devoid of pores because its empty cells were filled with gelinite.

**Table 1 - Properties and petrographic analysis of coal samples (Wu, et al., 2010)**

<table>
<thead>
<tr>
<th>Properties Parameters</th>
<th>Petrographic analysis</th>
</tr>
</thead>
<tbody>
<tr>
<td>Depth(top) (m)</td>
<td>Vitrinite</td>
</tr>
<tr>
<td>1305</td>
<td>48.68%</td>
</tr>
<tr>
<td>Mad (%)</td>
<td>Inertinite</td>
</tr>
<tr>
<td>1.01</td>
<td>47.35%</td>
</tr>
<tr>
<td>Aad (%)</td>
<td>Liptinite</td>
</tr>
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<td>3.35</td>
<td>2.85%</td>
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<td>Vdaf (%)</td>
<td>Mineral substance</td>
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<tr>
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</tr>
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<td>Fcd (%)</td>
<td>Reflectivity R</td>
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<tr>
<td>69.43</td>
<td>0.9310</td>
</tr>
<tr>
<td>Porosity (%)</td>
<td></td>
</tr>
<tr>
<td>8.6%</td>
<td></td>
</tr>
</tbody>
</table>

**Experiment device**

Based on literatures (Ye, et al., 2007), we design a high pressure visualization device with a sapphire window to measure critical values of CH₄-CO₂ binary system and CH₄-CO₂-coal sample ternary system (see Figure 1). The experimental device mainly consists of a visual autoclave, pressure systems, constant-temperature system, vacuum system, data acquisition system, and fast imaging system. The experimental device schematic diagram is shown in Figure 2. A frontal view and cross section of the view cell are shown in Figures 3 and 4.
The visual autoclave is the core of the experimental device made in stainless steel. Its body cavity is an oval-shaped vessel, and the volume is 100 ml. A magneton equipped with the electromagnetic stirrer is set in the bottom of it.

Around the autoclave body is fitted a thermocouple, and in it are two high-precision four-wire PT-100 temperature control sensors which can accurately monitor the inside temperature. The experimental temperature range is 0 to 150 °C; digital temperature display instrument value: 0.01 °C; and control temperature sensor accuracy is ± 0.1 °C. There are pressure sensors in the air intake manifold line of the visual autoclave to accurately measure the inside pressure. The maximum pressure that can be measured is 30 MPa, the maximum pressure of the sensor is 40 MPa, and the measurement accuracy is ±0.02 MPa.

The visual autoclave window is made of analysis class sapphire material (h = 2.5 mm and d = 2 mm of sapphire cylinder). Through the sapphire window the sample state and transformation process under high pressure can be directly observed, and the gas-liquid interface disappearance and critical opalescence phenomenon can be clearly observed.
The experimental process

Gas test

To inject the tested gas into the autoclave body through the high-pressure pump (The air from the system was purged by pressurizing the system to 1MPa with the tested gas and then depressurizing the equilibrium cell. Before injection, the body of the autoclave was washed with the tested gas three times and then discharged to exclude air and other gases, and ensure the purity of the gas); to start magnetic stirring, adjust the temperature and pressure of the test equipment, observe the supercritical state of the phase process through the visual window. The temperature was then lowered slowly through the cooler, lower the pressure slowly by gas release, and carefully observe the state in the autoclave. Until the recurrence phase transition within the autoclave, the phenomenon of critical opalescence occurs, record the temperature and pressure at this time, which is the measured critical temperature and critical pressure. To ensure the accuracy of the results, each test is repeated five times measurements received and their average values calculated.

Testing with coal sample

Put 30 g dry coal sample with particle size about 1-3 mm into the autoclave reactor and keep the coal sample surface from exceeding one-third of the visual window to avoid the adsorption expansion of coal gas hindering the video window. After vacuum degassing of the coal sample for six to eight hours at 80 °C, the tested gas was injected. Critical parameters testing is the same with the pure gas testing.

RESULTS AND DISCUSSION

Using a high pressure visualization device with a sapphire window to measure critical values of CH4-CO2 binary system and CH4-CO2+coal sample ternary system, based on the critical opalescence of carbon dioxide as a critical point criterion, the critical points of the CH4-CO2 binary system and CH4-CO2+coal sample were measured. The effects of different contents of CH4 on the supercritical CO2 were studied. There is no literature reports regarding the critical point data of CH4-CO2+coal samples. In this test, the experimental temperature is 10~40 °C, pressure 10 MPa below. The critical temperature and pressure variation with the composition contents were measured and the results are shown in Table 2.

<table>
<thead>
<tr>
<th>Number</th>
<th>Components</th>
<th>Tc/K</th>
<th>Pc/MPa</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>CO2</td>
<td>31.05</td>
<td>7.35</td>
</tr>
<tr>
<td>2</td>
<td>98.16% CO2+1.84%CH4</td>
<td>29.58</td>
<td>7.46</td>
</tr>
<tr>
<td>3</td>
<td>96.50% CO2+3.50% CH4</td>
<td>28.74</td>
<td>7.49</td>
</tr>
<tr>
<td>4</td>
<td>95.75% CO2+4.25%CH4</td>
<td>27.27</td>
<td>7.53</td>
</tr>
<tr>
<td>5</td>
<td>90% CO2+10%CH4</td>
<td>15.21</td>
<td>8.13</td>
</tr>
<tr>
<td>1'</td>
<td>CO2+coal</td>
<td>31.07</td>
<td>7.33</td>
</tr>
<tr>
<td>2'</td>
<td>98.16% CO2+1.84%CH4+coal</td>
<td>30.00</td>
<td>7.44</td>
</tr>
<tr>
<td>3'</td>
<td>96.50% CO2+3.50% CH4+coal</td>
<td>28.82</td>
<td>7.48</td>
</tr>
<tr>
<td>4'</td>
<td>95.75% CO2+4.25%CH4+coal</td>
<td>27.50</td>
<td>7.50</td>
</tr>
<tr>
<td>5'</td>
<td>90% CO2+10%CH4+coal</td>
<td>16.12</td>
<td>7.66</td>
</tr>
</tbody>
</table>

The results in Table 2 and Figure 5 show how the critical temperature and critical pressure of the CH4-CO2+coal system vary with the CH4 contents. When the CH4 concentration is less than 10%, the critical temperature of the system is between the critical temperatures of CO2 and CH4, which are both lower than the CO2 critical temperature and higher than the CH4 critical temperature, the critical pressure increases with the increasing CH4 contents (it agrees with the reported phase diagram, Swanenberg (1979), see Figure 6). The critical temperature of the CH4-CO2+coal system is slightly higher than that of the CH4-CO2 binary system, while the critical pressure is lower than that of the CH4-CO2 binary system, especially when the concentration of CH4 component is equal to 10%, the critical pressure of the CH4-CO2+coal system is 7.66 MPa, significantly lower than that of the CH4-CO2 binary system (8.13 MPa).

It is difficult to measure the critical parameters, and the opalescence phenomenon of the critical points is difficult to capture, the colors of critical points of different components are also different, the experimental results show that (see Figure 6), the color of opalescence phenomenon of the CO₂-CH₄...
The binary system critical point is yellow-brown, the color becomes dark with the increase of methane content, and the color of pure CO₂ critical opalescence is light yellow. The color of opalescence phenomenon of the CO₂–CH₄ system critical point is lighter than that of the corresponding CO₂–CH₄ binary. With CO₂ phase transition, the gas-liquid separation interfacial phenomena will occur near critical points, as shown in Figure 6D. When the pressure has an instant decrease to the critical points, the carbon dioxide ice inside the autoclave will block the window, as shown in Figure 6E, it is necessary to be careful to control the experimental pressure and temperature and thus the critical parameters may be measured.

![Figure 5 - The supercritical parameters of the CO₂–CH₄ system](image)

Figure 5 - The supercritical parameters of the CO₂–CH₄ system

![Figure 6 - Pictures of critical points of different components of the CO₂–CH₄ binary system and CO₂–CH₄+coal system](image)

Figure 6 - Pictures of critical points of different components of the CO₂–CH₄ binary system and CO₂–CH₄+coal system

In order to verify the accuracy of this test method, the critical values of pure CO₂ were measured, the measured critical temperature is 31.05 °C and pressure 7.35 MPa, which agrees with the theoretical values (Zhang, 2000). At the same time, comparing the critical pressure and critical temperature of the different component CO₂–CH₄ binary system and different component CH₄–CO₂ system phase diagrams (Swanenberg, 1979) (see Figure 7) with the test results of Arai (1971), there is good agreement. The locus of critical points for all bulk compositions is the so-called critical curve of the first order (a), connecting CCO₂ and COCH₄, it shows that, in general, liquid-gas equilibria shift towards lower temperatures at increasing XCH₄ at the low–temperature side the two-phase liquid gas field intersects
the univariant triple point (solid-liquid-gas equilibrium) curve, between L and V. The triple point curve extends from the triple point of pure CO\(_2\) (T\(_{co2}\)) to the quadruple point of the system CO\(_2\)-CH\(_4\). The curves labeled (c) are the boiling curves of the unary systems CO\(_2\) (Weast, 1975) and CH\(_4\) (Zagoruchenko, et al., 1969).

![Diagram showing phase behavior with labels and curves](image)

Figure 7 - Composite phase diagram of the system CO\(_2\)-CH\(_4\). The dotted 5, 6, 7 is the experimental data are those of Arail et al. (1971) and dotted 1, 2, 3, 4 are data from this test.

CONCLUSIONS

The critical parameters of the CH\(_4\)-CO\(_2\)+coal system were studied with a high pressure visualization device with a sapphire window. The experimental temperature is 10~40 °C (equal to the actual temperature of coal seams), and pressures are all below 10 MPa. The critical temperature of the CO\(_2\)-CH\(_4\)+coal system is between the critical temperatures of CO\(_2\) and CH\(_4\), which are both lower than the CO\(_2\) critical temperature and higher than the CH\(_4\) critical temperature.

To compare the results and critical parameters of the CO\(_2\)-CH\(_4\) system, the opalescence phenomenon is found, and the colour is lighter than that of the CO\(_2\)-CH\(_4\) system. The critical temperature is higher than that of the CO\(_2\)-CH\(_4\) system and the critical pressure is lower than that of the CO\(_2\)-CH\(_4\) system. When CH\(_4\) composition concentration is less than 10% and there are coal samples, the critical values of the CO\(_2\)-CH\(_4\) system change on different levels, indicating that the coal has a certain influence on the critical parameters of the CO\(_2\)-CH\(_4\) system. The critical values with coal samples become more complicated. This may be because the Yaojie Haishiwan coal is outburst coal, it has great adsorption capacity, the critical pressure becomes greater, and the possible reason may be the change of gas composition.

The critical parameters of CO\(_2\)-CH\(_4\)+coal system in this study can provide a reference to CO\(_2\) storage in coal seams, and they are close to the related parameters in actual coal seams of Haishiwan Mine of Yaojie Coal and Electricity Company (the actual measured temperature is 31-38 °C, coal seam gas pressure exceeds 7.5 MPa, the highest CO\(_2\) content in coal seams is up to 98%). According to that, there may be supercritical CO\(_2\) in the coal seams of Yaojie Haishiwan Mine.

ACKNOWLEDGEMENTS

The authors would like to acknowledge the support of the Key Project of the Natural Science Foundation of China (Nos.70533050 and 50774084), the project was financially supported by “the Fundamental Research Funds for the Central Universities” of China 2010QNB02.
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GAS CONTENT AND EMISSIONS FROM COAL MINING

Abouna Saghafi

ABSTRACT: Gas content can be considered the most important parameter for assessing emissions from coal seams during and post mining. Traditionally, the purpose of gas content determination was to assess gas outburst potentials and to quantify the magnitude of gas emissions into underground headings and at the coal face. Therefore, it’s a traditional definition; measurement and determination are based on these objectives. However, the calculation of emissions from mining for the purpose of establishing a greenhouse gas inventory would require new definitions for gas content and new measurement methods. Moreover, errors inherent in measuring gas content need to be quantified so that the uncertainty of emissions inventory can be evaluated. Therefore, various definitions of gas content in relation to the purpose of its use are suggested. Anew method of measurement for low gas content coals is discussed. Moreover, various parameters influencing the value of measured gas content are discussed and errors of estimation using the direct method are evaluated.

INTRODUCTION

Mining leads to large disturbances of coal seam reservoirs as fractures develop both in coal and rock strata. Gas trapped in the coal seam and enclosing strata escapes to the atmosphere via fractures and exposed coal surfaces. The intensity of emissions depends on the flow properties and the volume of gas present in the coal seam and strata at the time of mining. The method of mining affects the extent and density of induced fractures which could increase the permeability by several orders of magnitude. This increase in permeability can in turn accelerate the discharge of ground water from mining which leads to a further increase in the relative permeability to gas. The rate of gas liberation during mining, therefore, depends primarily on the virgin gas content of coal and the extent of induced fracture permeability. However, the magnitude of post mining emissions depend mainly on residual gas content, matrix permeability and gas diffusivity through coal and non-coal strata.

It can be shown that if coal mining proceeds at a relatively constant rate during the life of mining, annual emissions from mining can be evaluated by using virgin gas content and production, besides the lithology of strata and geometry of mining (Saghafi, 2010). Gas content is therefore the most important parameter for evaluation of fugitive emissions from mining. Moreover, in view of large outputs of coal in Australia, even small errors in measuring the gas content could lead to large errors in calculated estimates of emissions. In addition to the accuracy of gas content determination, the limit of measurability is also an issue for low shallow coals such as in open cut and ‘non gassy’ underground coal mines. Traditionally the requirement for gas content testing had been limited to the ‘gassy’ mines where the safety is the major driving force aimed at evaluation of outburst potentials and high gas emissions. Therefore, low gas content determination has not been a focus of research. At this time the lowest measurability level is about 0.1 to 0.5 m³/t.

Thus the development of more suitable and accurate methods of gas content testing for low gas content coals an important task ahead for coal researchers and the coal mining industry. Another beneficiary of a more accurate method for low gas content coals is the CBM industry which is also active in producing gas from low rank coal regions where coals of low gas contents are present.

GAS CONTENT OF COAL

Gas content is generically defined as the volume of gas contained in a unit mass of coal and is generally expressed in cubic metres, at standard pressure and temperature conditions, per tonne of coal (m³/t, STP). In Australia the standard conditions are a temperature of 20°C and an absolute pressure of 101.325 kPa (Standards Australia, 1999).

Gas in coal is stored mainly in the adsorbed phase but also in the free phase. Though the contribution of the latter to the total gas volume is small, particularly at shallow depths (<500 m), at greater depths the
volume of in the free phase can be large due to higher density at such depths. From a viewpoint of gas storage in coal, gas content should include both the free and adsorbed volumes, however, the current method of determination does not allow for the measurement of free gas in coal. The current method mainly targets the ‘desorbable gas content’ and to some extent the ‘residual gas content’.

Desorbable and residual gas content

As soon as coal is brought to the surface it desorbs its gas. Desorption of gas from coal could continue for days or weeks until there is no ‘measurable’ gas. The total volume of gas released from coal, when the gas pressure outside the coal is at atmospheric pressure, is called the desorbable gas content (Q_d). Gas remaining in coal at this stage is the residual gas content (Q_r). Total gas content (Q_t) is the sum of the desorbable and residual gas contents (Figure 1).

![Figure 1 - Desorbable and residual gas content defined based on natural release of gas from coal at atmospheric pressure](image)

The rate of gas release from coal depends on the gas concentration gradient (between the inside and outside of coal sample) and the diffusivity of coal. The process of gas release from coal can be considered a combination of instantaneous desorption of gas from the internal surface of pores and diffusion to the fractures. The release of gas from coal can be expressed mathematically assuming a diffusion mechanism as shown in Equation 1,

\[
\frac{\partial Q_r(t)}{\partial t} = D \frac{\partial^2 Q_r(t)}{\partial x^2}
\]

(1)

Where Q_r(t) is the gas remaining in coal (temporal gas content) at any time t (s) after the start of the desorption process and D is the gas diffusivity in coal (diffusion coefficient, m²/s). Assuming a ‘pseudo steady’ diffusion mechanism the solution of Equation 1 yields equation 2.

\[
Q_r(t) = Q_0 + Q_d e^{-t/\tau}
\]

(2)

Q_r and Q_d are the residual and desorbable gas contents (Figure 1). Parameter \( \tau \) (tau) is a characteristic time parameter (s) related to the diffusivity of gas in coal, it is sometime called diffusion time constant or desorption time. It is expressed in terms of the diffusivity parameter \( D \) and a diffusion characteristic length \( a \) shown in Equation 3

\[
\tau = \frac{a^2}{D}
\]

(3)

Evaluation of parameter \( \tau \)

Tau (\( \tau \)) is a physical parameter related to the diffusivity and the characteristic length for a given gas and coal. If the diffusion coefficient and characteristic length values are not available, \( \tau \) can be evaluated.
from the gas desorption testing curve. Based on Equation 2, the volume of gas desorbed from coal since the start of the desorption process is given in Equation 4.

\[ Q_d(t) = Q_d(1 - e^{-t/\tau}) \]  

(4)

Where \( Q_d(t) \) is the volume of gas released since the start of desorption (m\(^3\)/t). In Eq (4), if the time \( t \) is replaced by \( r \), the volume of gas released from coal after a period of \( r \) would be: \( Q_d(r) = 0.63Q_e \). In other words \( r \) is the time required for coal to release 63% of its desorbable gas. Hence, \( r \) can be estimated from the gas content testing desorption curve (Figure 2). This method, however, is costly because the slow desorption measurement could take weeks to complete.

![Graph](https://via.placeholder.com/150)

**Figure 2 - Determination of diffusion characteristic time from the desorption curve**

### MEASUREMENT OF GAS CONTENT OF COAL

Gas content of coal is determined by direct measurement of the volume of gas desorbed from coal. In Australia two methods are routinely used, namely the ‘slow desorption’ and ‘fast desorption’ methods (Williams, et al., 1992; Saghafi, et al., 1998). Variants of both methods have been used in other coal mining countries. In France a variant of the fast desorption was used since early 1960’s (Bertard, et al., 1970). In the USA variants of the slow desorption method have been used over the years (Kissell, et al., 1973; Diamond and Levine, 1981; Diamond and Schatzel, 1998). The Australian slow desorption method was a developed form of the USBM method after some enhancement (Australian Standard, 1991). The Australian fast desorption method otherwise, known as the quick crash method, was developed in the early 1990’s (Williams, et al., 1992; Saghafi, et al., 1998) and is currently the method of choice for assessment of gas emissions and outburst risk in underground coal mines (Australian Standard, 1999).

Though both methods consist of similar steps to determine the gas content of coal, the length of the procedure is significantly longer in the slow desorption method. In the fast desorption method coal is crushed after a short period of natural desorption so that all its gas is forced to release rapidly (in the space of minutes to an hour). An advantage of the fast desorption method besides its rapidity is a reduction in the risk of CO\(_2\) dissolution in the measuring water for mixed gas conditions (when both CH\(_4\) and CO\(_2\) are present in seam gas).

The slow desorption and fast desorption methods are both based on measurement or estimation of the volume of gas desorbed from coal in several stages. Both methods start with estimating ‘lost’ gas during drilling and retrieval of the coal sample to the surface. In the slow desorption method coal is allowed to desorb its gas ‘naturally’ until no further desorption is detected. In the fast desorption method, however, after a short time allowed for gas to be released naturally during coal transport to the laboratory and in the lab, the coal is crushed. Note that in the slow desorption method the crushing stage may be also included to determine the ‘residual’ gas content of coal \( (Q_e) \).

The three components of gas contents from three stages of gas content testing in the fast desorption method are commonly represented by \( Q_1 \), \( Q_2 \) and \( Q_3 \) parameters. The ‘measured gas content’, \( Q_m \), is the sum of the 3 components (Australian Standard, 1999) shown in Equation 5,

\[ Q_m = Q_1 + Q_2 + Q_3 \]  

(5)

The \( Q_1 \) or the lost gas stage is identical for the two methods. The \( Q_2 \) gas desorbed during transport and in the lab is also called desorbed gas in the slow desorption method and is the main component of the
gas content in this method. In this method this stage is allowed to continue until no further measurable
gas desorption is observed. In the fast desorption method the \( Q_3 \) step is generally short as coal is
crushed soon after reaching lab depending on the availability of a measuring system and proper
conditions. The last component of gas content \( Q_3 \), which is the gas desorbed from crushed coal in the
fast desorption method, is also usually the largest volume of desorbed gas in this method. For the slow
desorption method this stage is often of no importance as residual gas is expected to be small.

**Errors associated with the standard method of gas content testing**

The measurement of the volume of gas released (in all three stages) is done by using a measuring
cylinder. The released gas is admitted into a water filled inverted cylinder. The displacement of water
provides the measure of the volume. This system has worked well over the years and is used routinely in
Australia. There are, however, some problems with this way of measuring the volume of gas including
calibration of the measuring cylinder. The released gas is admitted into a water filled inverted cylinder. The displacement of water provides the measure of the volume. This system has worked well over the years and is used routinely in Australia. There are, however, some problems with this way of measuring the volume of gas including coal oxidation, gas partial pressure effects and dissolution of desorbed gas, particularly \( \text{CO}_2 \), into the measuring water. Some of these issues have been addressed over the years and some improvements have been suggested and applied (Saghafi and Williams, 1998; Saghafi et al., 1998; Danell et al., 2003).

Measurement of mass (or weight) produces relatively small errors, hence, the accuracy (or error) of measured gas content depends mainly on the accuracy of determination volume. The accuracy of the graduation of the measuring cylinder is, therefore, of primary importance. Accuracy of the graduation depends on the quality and resolution of measuring cylinders. For example glass cylinders are less prone to error than plastic cylinders, which are affected by temperature and other environmental conditions. Burettes have higher resolution but are delicate and their use is limited to the laboratory.

The total error of measurement should be reported in gas content results. It depends on individual errors of \( Q_1 \), \( Q_2 \), and \( Q_3 \). Measured gas content inherits all these errors and the total error of gas content (\( \varepsilon_m \)) could be calculated as follow,

\[
\varepsilon_m = \varepsilon_1 \frac{Q_1}{Q_m} + \varepsilon_2 \frac{Q_2}{Q_m} + \varepsilon_3 \frac{Q_3}{Q_m}
\]

(6)

Where \( \varepsilon_1 \), \( \varepsilon_2 \) and \( \varepsilon_3 \) are the errors produced in measuring \( Q_1 \), \( Q_2 \) and \( Q_3 \) components. For example in one case we have

\( Q_1 = 0.5 \text{ m}^3/\text{t}, \ varepsilon_1 = 20\% \),
\( Q_2 = 3.0 \text{ m}^3/\text{t}, \ varepsilon_2 = 15\% \),
\( Q_3 = 2.5 \text{ m}^3/\text{t}, \ varepsilon_3 = 5\% \).

The measured gas content is then presented as \( Q_m = 6.0 \pm 0.67 \text{ m}^3/\text{t} \).

**GREENHOUSE GAS ESTIMATION AND GAS CONTENT**

The gas content of coals from shallow seams in open cuts and ‘non-gassy’ underground mines can be very low. For these coals, the conventional method of measurement of volume by using water displacement may not deliver anything meaningful and errors of measurement can be larger than the gas content itself. Currently by using the standard method, the lower limit of gas content can be shown to be about 0.1 to 0.5 m\(^3\)/t depending on the sample size and the measuring cylinder used. Another source of error for low gas content testing is the correct determination of nitrogen (N\(_2\)) in the seam gas. As N\(_2\) is not a greenhouse gas it is important that its volume be corrected in seam gas. The current method of determination of N\(_2\) content of seam gas is to calculate the ‘excess N\(_2\)’ volume from the gas composition of the samples collected from desorbed gas. In calculating the N\(_2\) content of seam gas it is assumed that the O\(_2\) deficiency in the desorbed gas is due to coal oxidation. Accordingly, a reliable method of gas content testing for low gas content coal is required, as accurate gas composition and gas content measurements for these coals would have an important impact on large coal mining operations. For low gas content coals (\( Q_m < 0.5 \text{ m}^3/\text{t} \)) the water displacement method of measuring the volume is inadequate. Often for these coals there would be no measureable \( Q_1 \) and sometime no measurable \( Q_2 \).

**Suggestions on measuring low gas content coals**

For the measurement of gas content of these coals it is suggested that the best practice would be to seal the fresh sample in a gas tight canister in the field, and then dispatch it to the lab for crushing. Ideally
coal should be sealed in a canister which can be directly mounted on the crusher so that there would be no need to open the coal canister before crushing. The total desorbed gas can then be indirectly evaluated using a method based on measuring of the gas composition rather than measuring gas volume. If coal oxidation is an issue the sample should be flashed by helium gas (He) before sealing the coal canister assuming that there is no $Q_1$. If $Q_1$ is to be estimated then He flushing should take place after measurement for the $Q_1$ estimate.

Once in the lab coal is crushed in the sealed canister (the CSIRO lab is equipped to crush sealed coal without opening the canister on arrival from the field). Crushed coal in the container is then kept for a period of time to allow for equilibrium and desorption of gas. A gas sample is then taken from the headspace to measure gas composition. Knowing the volume of the canister and the gas composition, gas content can be evaluated. Note that the increase in gas pressure would be minimal due to the low gas content of the coal. However, a gas pressure sensor may be connected to the container to measure the gas pressure. This method was used at the CSIRO in a number of ACARP projects on the determination of residual gas content of coal. The method was applied after the completion of the three stages of gas content testing ($Q_1$, $Q_2$ and $Q_3$). This new component of gas content was called $Q_3'$ ($Q_3$ prime).

For routine measurement of low gas content coals a design similar to the set up conceptually illustrated in Figure 3 is suggested. After completion of $Q_1$ and $Q_2$ stage of testing coal is crushed using the lab standard crusher. Note that the crushing canister should be initially flushed with He gas to reduce air in the system and also to reduce the seam gas partial pressure. After completion of the crushing and allowing time for temperature equilibrium, the canister is opened to a closed circuit with an in-line vacuum pump. More helium can be allowed into the system to further reduce the partial pressure of desorbed seam gas. Gas samples are collected from the system after sufficient periods of time for gas composition measurement. Knowing the volume void space and the seam gas composition the gas content can be determined.

![Figure 3 - Schematic diagram of proposed system for low gas content testing](image)

The lower limit of gas content which can be determined using this method can be evaluated from the knowledge of void volume in the system (crusher and piping) and the lowest or optimal lower limit of gas chromatography in use. For instance, if the void volume is about 500 cm$^3$ (typical void in the CSIRO quick crush canister for a 100 g coal sample), and a GC which can accurately measure a concentration of 100 ppm of methane is used, then the gas content of about 0.001 m$^3$/t can be theoretically determined. This method can, therefore, theoretically measure, gas content values of 100 times smaller than the standard method.

**Gas content determination and excess nitrogen**

Nitrogen is frequently reported in the desorbed gas composition. While nitrogen is also a by-product of the coalification process, its high values in reported measurements is of concern. For the purpose of greenhouse gas emissions inventory it would be required to quantify the N$_2$ component of the seam gas as accurately as possible. One cause of high N$_2$ reported for ‘non gassy’ coals could be O$_2$ absorption and oxidation. Coal and carbonaceous materials oxidise in the presence of air and slow oxidation occurs at ambient temperatures. Laboratory and field measurements of gas emitted from coal in the presence of air show that the overall low temperature oxidation (<75°C) of coal and carbonaceous materials produce mainly CO$_2$ and to a lesser extent CO (Carras et al., 1994; Saghafi et al., 1995; Roberts et al., 2003). CO$_2$ is the main result of the reaction and uses about 90% of the total consumed O$_2$ (see for
example Wang et al, 2003). If seam gas did not contain CO$_2$ then it was possible to estimate N$_2$ content of coal seam gas from the knowledge of the O$_2$ and CO$_2$ composition. However, seam gas in Australia is often rich in CO$_2$ and new methods are required to identify and quantify the true N$_2$ content of coal seam gas.

CONCLUSIONS

The current method of determination of gas content for the low gas content of non gassy coals is prone to error. As the gas content is the most important parameter for estimating mine emissions, new methods are required to increase the accuracy and measurability of the gas in coal. The high nitrogen content of seam gas reported in gas content testing using the standard method, particularly for low gas content coals, is also an important issue which requires attention and development of new methods to quantify the true N$_2$ content of coal seam gas.

A new method of measurement for low gas content coals is suggested. The new method is based on lowering seam gas partial pressure in the coal canister and the measurement of gas composition. The new method should largely increase the limit of measurability and accuracy of the gas content testing. Analysis of the method indicates that theoretically gas content of two orders of magnitude smaller can be measured by using the new method instead of current methods.

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GAS CONTENT MEASUREMENT AND ITS RELEVANCE TO OUTBURSTING

Ian Gray

ABSTRACT: The use of core and alternatives to its use for gas content measurement of coal is examined. The measurement of gas content from cuttings is presented in two forms, one involving the recovery of cuttings with air drilling and the second involving the collection of all released gas from the hole during overbalanced drilling. The latter approach is suitable for not only coal but all gas bearing formations including siltstones and sandstones, provided they do not have major open pore space such as vugs. The paper also deals with the importance to outbursting of gas content, broken coal particle size and diffusion coefficient.

CORE DESORPTION

Core desorption is the standard process for determining the gas content of cores. This process is described by McCulloch and Diamond (1976), and more recently Standards Australia (1999). The process generally involves using wire line coring to cut a core so that the core may be retrieved quickly. Once the core is retrieved to surface the core is placed in a canister and the released gas is monitored with respect to time. This should be undertaken at reservoir temperature. An example of gas release versus time is shown in Figure 1.

Figure 1 - Example of desorbed gas measurement (Q2). Note the time here refers to time of day

Once the core has been further desorbed the canister is opened and the core is logged and weighed with density determination. Weighed sub sections of the core are then crushed to enable the remaining gas to diffuse out more quickly than from the core.

This process is relatively straightforward but the determination of the gas lost before the core is placed in the canister is not. The usual process adopted is to assume a time when the core begins to release gas and to plot the gas release with respect to the square root of time. An example of such a plot is shown in Figure 2.

Figure 2 shows a very good straight line plot. Equation (1) for Fickian diffusion from a homogeneous cylinder is as published by Crank (1975).

\[
\frac{M_t}{M_\infty} = 1 - \sum_{n=1}^{\infty} 4 \cdot \text{JORD}_n \cdot e^{-D \cdot \frac{\text{JOR}}{a} \cdot t}
\]  

(1)

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Where: $\frac{M_t}{M_\infty}$ is the ratio of desorbed gas over the total gas that may be released;

$\text{JQR}_i$ are the roots of a Bessel function of the first kind for the equation;

$D$ is the diffusion coefficient (length$^2$/time);

$t$ is time;

$a$ is the radius of the cylinder.

Figure 2 - Example of lost gas determination plot (Q1)

For small values of $Dt/a^2$ equation the general equation may be approximated to that of Equation 2 below, also taken from Crank (1975):

$$
\frac{M_t}{M_\infty} = 4 \frac{(Dt)^{1/3}}{\sqrt{\pi} a^2} - \frac{Dt}{a^2} - \frac{1}{3\sqrt{\pi} a^2} \left( \frac{Dt}{a^2} \right)^{3/2} + ...
$$

The straight line approximation with the square root of time comes from the first term of the above equation and shows a 10% error at a value of $Dt/a^2 = 0.05$. For values of $Dt/a^2$ greater than 0.05 the value from the first term approximation of equation 2 diverges rapidly from the theoretically correct solution.

Care must be exercised in the use of the straight line approximation for gas loss. A prime source of error is the incorrect determination of the time when initial gas loss occurs. Standards Australia (1999) arbitrarily sets this at the mean time between when the core starts being pulled and reaches the surface. In some cases determining the onset of gas release in the hole needs to be looked at more carefully. Another source of error occurs if the core is retrieved too slowly and substantial gas loss occurs. In this case the value of $Dt/a^2$ may mean that the linear approximation is quite simply incorrect. This can be checked quite readily from a calculation of the slope of the lost gas plot and the total gas content.

It must be remembered that the coal core is not a uniform cylinder. It is inhomogeneous and fractured and contains various macerals and ash. The more highly fractured components of the core and those with higher diffusion coefficient will release their gas more quickly than the less fractured ones with a slower diffusion coefficient. A basic method of checking the validity of the initial gas loss estimation is to examine the ratio of the lost gas (Q1) to total measured gas content (Q2+Q3). If this value is too high then a question will remain over the total gas content value.

It is possible to derive an estimate of the diffusion coefficient from the slope of the lost gas plot. This has particular relevance in assessing gas an energy release from coal particles in an outburst.

**CUTTINGS DESORPTION FROM AIR DRILLING**

It is possible to determine gas content by collecting cuttings using air drilling. This is possible because the high speed at which the coal is delivered to the hole collar minimizes the gas loss. The process for doing this in the underground mine context is shown in Figure 3. Here drilling is taking place using air flushing with the cuttings being collected by a cyclonic separator and bag filter arrangement. The cuttings collected are transferred to a canister and the gas release with time is measured, much as in the case of core desorption, except that the process happens more quickly (approximately 2 hrs) because...
the coal is in small pieces. As desorption slows the canister can be opened, the sample weighed and a sub sample taken for crushing to obtain a value of the residual gas content. Thus the measured gas release includes both the values from normal desorption (Q2) and from crushing (Q3).

Figure 3 - Underground drilling setup to collect cuttings with air flush drilling

There is a need to then determine the lost gas volume (Q1). This is in some ways easier to do than for the case of core desorption because the time at which gas loss starts is known with precision (within 30 s) as the time at which drilling takes place. The key to determining the lost gas is to measure the particle size distribution of the remaining cuttings. Using this size distribution, and the gas release versus time information, combined with the residual gas content it is possible to use a model of diffusion to determine the lost gas using a best fit history match. Equation (3) from Crank (1975) has been found to be model the situation quite adequately. It describes Fickian diffusion from spherical particles.

\[
\frac{M_t}{M_w} = 1 - \frac{6}{\pi^2} \sum_{n=1}^{\infty} \frac{1}{n^2} e^{-\frac{Dn^2\pi^2}{4a^2t}}
\]

Here the symbols are the same as previously noted except that \(a\) refers to the sphere radius.

The total gas content is thus determined from the estimate of lost gas and the measured gas released, providing a very accurate estimate of the gas content of coal and a value of the diffusion coefficient of the coal particles. Figure 4 shows an example of a real gas content determination from this process. The example is from work by the author in the D6 seam at Lenina mine in the Karaganda Basin, Kazakhstan. This was a dry coal seam which made the operation easier.

Figure 4 - An example of gas content measurement from air drilled cuttings desorption

The results taken from the case described in Figure 4:

- the diffusion coefficient is calculated at \(1.54 \times 10^{-12} \text{ m}^2/\text{s}\);
- the total gas content is calculated at \(18.4 \text{ m}^3/\text{t}\);
the lost gas estimate is 3.19 m$^3$/t; and
the residual gas measured is identical to that predicted - 4.6 m$^3$/t.

**GAS CONTENT MEASUREMENT WITHOUT CORING (GCWC) IN HOLES DRILLED WITH MUD**

There are many cases where air drilling is impractical, more particularly from a well control and hole stability viewpoint. While in the past there have been many efforts made to determine the gas content of the material drilled from cuttings these have been subject to substantial inaccuracy due to an inability to determine the volume of gas lost from the cuttings in their passage up the well bore. This inaccuracy comes from the same source as that for core, namely an uncertainty as to when gas release commences and the rate at which it takes place.

The proposed solution which is being tested is to drill overbalanced so that no gas is released from the formation into the well bore. This is combined with the use of a rotary seal between the drill pipe and the casing to ensure that no gas escapes without measurement at surface. Thus all the material coming from the well including mud, cuttings and gas may be directed to a separation system. The separation system splits the gas from the mud and cuttings and permits the gas volume to be measured. The mud and cuttings are then further separated on a shaker and the cuttings may be desorbed further. Indeed the gas released while the cuttings are on the shaker may be determined by covering the shaker with a shroud ventilated at a known rate, and by measuring the rate of gas release in the air stream from the shroud.

The gas remaining in the cuttings may be determined by sampling these and measuring the gas release from them. This is followed by further grinding to measure the residual gas content of the cuttings.

The total gas volume measured must be related back to where the gas came from in the hole. This requires the position of the bit to be logged along with the mud flow rate, including periods of no flow while drill pipe is added. Care must be taken to ensure that air does not become entrained in the drill string when pipe is added to it. The use of a gas analyser to measure the composition of the gas delivered from the separator helps in detecting air that has become entrained in the system, either on the suction side of the mud pump or during the connection of drill pipe. It is also necessary to relate the volume of gas release back to the volume of the material which has been drilled. This can be approximated from the theoretical volume of the hole but is better checked by using a caliper log. The total estimation of gas content is calculated by incorporating all of this information into a model.

Figure 5 is a schematic diagram of the system used to obtain gas content information from open hole drilling from surface using mud. This method has the potential to be more accurate than core desorption because there is no reliance on lost gas determination – all gas released is captured.
It is possible to arrange an underground analogue of the system shown in Figure 6. In this case though the drilling fluid must be maintained at an artificially high pressure so that fluid flow into the borehole does not occur. Such a system is shown in Figure 6. This system was developed as part of an ACARP project (Gray, 1998). Variants of it have been used successfully on surface for controlled pressure drilling but the system has never been used underground. If it were to be used the waste could be diverted to a separator system similar to that of Figure 5 to measure the gas released from the chips that are sampled.

![Borehole pressurization system](image)

**Figure 6 - Borehole pressurization system**

**IMPLICATIONS FOR OUTBURSTING**

Outbursts are sudden expulsions of coal and gas from the working face. They may injure or cause fatalities through the action of mechanical force or through asphyxiation. In worst circumstances the sudden gas release may fuel and ignite which leads to a dust explosion.

The key to knowing danger presented by an outburst is substantially related to the energy release associated with it. Gray (1980, I and 2006) identified the energy sources as being strain energy of the failing material, the expanding gas contained in pore (cleat) space within the coal and from expanding gas which diffuses from the particles. The rate of gas diffusion from particles is related to their size, their gas content and diffusion coefficient. In the 2006 work the bulk of the energy release is identified as being related to expanding gas. Importantly the risk of outbursting is not simply related to gas content as is simplistically assumed by current Australian practice.

The system shown in Figure 3 has most of the components needed to determine outburst risk. The particle size distribution from drilling is determined by sieving the sample following desorption. It may conservatively (on the fine side) be assumed to resemble that produced in and outburst. In addition the diffusion coefficient and gas content can be calculated from the particle sizes, desorption rate and mass of the sample which is collected from the cyclonic separator. The volume of the cuttings held in the cyclone can be easily compared with the nominal volume of the borehole section drilled as another indicator of outburst proneness. If the air flow rate is known and the gas proportion in the return airstream is measured the total gas release from the hole may be measured to reveal abnormalities.

**CONCLUSIONS**

This paper reviews the process by which gas content of coals is determined by coring noting the main deficiency; namely the determination of the quantity of gas lost during core retrieval. Core gas content measurements can be made more reliable by calculating whether gas loss assumptions are handled properly. It presents a system for finding the gas content of coals by measuring the release of gas from cuttings obtained during air drilling. This has proven to be quite accurate because of the speed with which the cuttings are retrieved, and also the mathematical rigour used to calculate the gas lost from the cuttings between being cut and placed in a canister. This technique is derived from old technology from Europe and Japan (Gray, 1980, II) disregarded in Australian coal mining practice. Used with the correct analytical techniques, updated instrumentation and modern computer power it can be used to provide all the information to determine outburst risk. The exception being that it does not permit the calculation of strain energy.

Drilling with air does however simulate mining into the coal seam the results of which can be observed through measurement of the coal and gas volumes produced. Used with the correct analytical process
these measurements can be used to produce energy release estimates on outbursting that incorporate such factors as toughness (through particle size), gas content and diffusion coefficient which are likely to be far more reliable than the blanket process of working off a gas content measurement derived from core.

The paper also presents a system by which the gas content of all strata, including coals may be determined. This is achieved by drilling using over balanced conditions and collecting all gas released during drilling of a hole. This system is intended to permit the measurement of all strata that might break up to form the goaf to be determined. This has important implications for gas release into the goaf and for greenhouse gas emissions.

REFERENCES

STRESSES IN SEDIMENTARY STRATA, INCLUDING COALS, AND THE EFFECTS OF FLUID WITHDRAWAL ON EFFECTIVE STRESS AND PERMEABILITY

Ian Gray

ABSTRACT: This paper describes the methods which can be used to determine the in-situ stresses in sedimentary strata including coals. It also examines the changes in effective stresses brought about by fluid withdrawal such as that caused by gas and water drainage. These changes in effective stress are brought about directly by changing fluid pressure and, in the case of coals, by the effects of shrinkage as the coal releases gas and if it dries out.

As the permeability of coal is very significantly affected by its effective stress the drainage of coals is dependent on the state of effective stress. Coal permeability may either increase or decrease during drainage. Permeability frequently shows some initial decline as fluid pressure decreases before shrinkage effects cause an increase. In some cases though, the overall trend is for a continuing decline in permeability. Under these circumstances no amount of stimulation to induce drainage will work and some other means must be sought to de-stress the coal. This has typically been by working an adjacent seam which is more amenable to being mined.

INITIAL STRESSES IN COALS

Determining the initial stress in the coal is important because it represents the starting level of stress before varying fluid pressures and coal shrinkage changes the stress levels. The importance of knowing initial stress is so that the initial value of the coal’s mechanical behaviour may be assessed at the correct stress level. Mechanical behaviour specifically refers to the non-linear stress strain behaviour of coal and to the stresses at which coal may fail. It is also important to know the initial stresses so that it is possible to determine whether coal is de-stressed.

The measurement of stress in coals is, however, not simple because coal is generally fractured and weak. This means that hydrofracturing in coals usually only reveals the stress level normal to the cleat direction closest to being perpendicular to the normal principal stress. For the same reasons the process of overcoring cannot be used in coals as they break up. It is however possible to measure stress in competent strata surrounding the coal seam and make an estimate of the stress in the coal itself. The most accurate procedure for this is to use overcoring. Hydrofracture is a second option while some rather less accurate measurements may be made by examining borehole breakout trends or hole ovality. Overcoring requires that the rock should remain in an elastic state through the process. The extraction of information on the major principal stress from hydrofracture test data also requires that the rock is elastic though the minimum stress may be determined in rock with more plastic behaviour. Borehole breakout requires that the rock should have passed any elastic situation and to have actually failed.

Overcoring is a process where a pilot hole is drilled at the end of a larger hole and a tool is inserted to measure either strain on the wall of the pilot hole or the diameter of that hole. The equipment includes some means to orient the tool and then the core is cut over the top of the tool while the deformations of the pilot hole are measured (Gray, 2000). The tool is retrieved in the core and the information downloaded from it and the rock sample taken for measurement of Young’s modulus and Poisson’s ratio. A pictorial view of the overcoring process is shown in Figure 1 while examples of the diameter traces during the overcore are shown in Figure 2. Figure 3 shows the best fit of a theoretical pilot hole deformation to the measured deformations.

The stresses measured in sedimentary rocks will vary significantly through the strata. This variation is substantially due to the varying stiffness of the rock. Having obtained stress measurements in some of the strata surrounding the coal seam it is necessary to consider how these are distributed through

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10 – 11 February 2011

297
sequence and to obtain an estimate of the stresses which may exist in the coal. To do this, it is necessary to develop a suitable model.

Figure 1 - Pictorial view of overcoring operation

Figure 2 - Example of diameter change with time during overcoring

Figure 3 - Example of best theoretical deformation fit to real diameter change points

Sedimentary strata is generally laid in marine or lacustrine environments and built up as a series of layers. Where the strata remains fairly horizontal and not too contorted or faulted, the vertical stress ($\sigma_v$) is essentially due to self-weight and can be calculated on the basis of 0.025 MPa/m depth or an effective stress of 0.015 MPa/m depth. Assuming the rock is laterally constrained, such that there is no allowable strain in the horizontal plane, the horizontal effective stress due to self-weight ($\sigma'_{h/sw}$) can be calculated using the following Equation (1):

$$\Delta \sigma_{h/sw} = \Delta \sigma_v \left( \frac{1}{1-v} \right)$$

(1)
In practice, the horizontal stresses are very seldom equivalent to this. Part of the reason is that this equation represents a simplified elastic model which does not account for creep processes. More generally, there are other components which result from horizontal effective tectonic stress \( \sigma_{htec} \), generated by tectonic movements. This may be due to tectonic plate loading, but is more frequently due to local structural conditions such as anticlines and synclines.

The major and minor principal effective tectonic stresses, \( \sigma'_{htec/1} \) and \( \sigma'_{htec/2} \), are calculated using Equations 2 and 3:

\[
\sigma'_{htec/1} = \sigma_1 - \sigma_{h/sw}
\]

\[
\sigma'_{htec/2} = \sigma_2 - \sigma_{h/sw}
\]

It is desirable, regionally, to consider the strain caused by tectonic movements, rather than focusing on stress fields. Stresses vary with the modulus of the rock; the stiffer the rock, the more stress it carries for a given strain. Using the values of tectonic stress calculated from Equations 2 and 3, the components of tectonic strain can be calculated as follows in Equations 4 and 5.

\[
\varepsilon_{tec} = \frac{\sigma'_{htec/1} - \sigma'_{htec/2}}{E}
\]

\[
\varepsilon_{tec/2} = \frac{\sigma'_{htec/2} - \sigma'_{htec/1}}{E}
\]

To examine the average tectonic strain for a group of stress measurements it is necessary to rotate the principal strains into direct N-S and E-W strain and shear strain components to find the mean of these. The principal tectonic strains and their direction may be calculated from these three mean strains. If tectonic strains are relatively uniform between adjacent stress measurements they may be used to calculate stresses in rock of varying stiffnesses and Poisson’s ratios in locations stress measurements have not been made. This process is the reverse of that used to derive the tectonic strain.

In these cases, the horizontal stress due to overburden is calculated according to Equation 1. The effective stresses due to tectonic strain may be calculated using Equations 6 and 7.

\[
\sigma'_{h/tec/1} = \frac{-E}{1 - v^2} \left( \varepsilon_{tec/1} + v \varepsilon_{tec/2} \right)
\]

\[
\sigma'_{h/tec/2} = \frac{-E}{1 - v^2} \left( \varepsilon_{tec/2} + v \varepsilon_{tec/1} \right)
\]

The principal effective stresses are calculated by adding the horizontal stress due to self-weight to the above figures.

Figure 4 illustrates a theoretical example of a layered sedimentary strata with varying stiffness and Poisson’s ratios. The rock is subject to gradually varying tectonic strains. The major tectonic strain, increasing with depth, indicates some features of a possible anticline, while the minor tectonic strain shows the reverse trend. The stiffness of the strata varies considerably and so, correspondingly, do the stresses.

Vertical variation in tectonic strain is indicative of unconformities. Faulting may often be detected by lateral variations in tectonic strain. Faults are invariably locations of stress relief, and normally faulted zones generally show very low to negative tectonic strain, though not tensile stress.

The tectonic strain model fairly frequently provides a good basis for understanding the stress distribution within strata. It is a useful analytical technique for understanding the stress distribution. Departures from this model need to be considered as the abnormal and deserving of further study. They may be due to unconformities or extreme plasticity of the rocks.

A good example of even tectonic strain across major variations rock properties is gained from a borehole drilled in the Surat Basin in Queensland. Here two stress measurements were taken adjacent to each other in quite different material and the tectonic strain was calculated and found to be almost identical. This is presented below in Table 1. A similar case is presented from a proposed mine in the Bowen Basin in Table 2. Here the major tectonic strains are very similar while the minor tectonic strains show more variation.
It should be noted that tectonic strains can be quite variable as they are influenced by local geological features.

![Figure 4 - Theoretical stress distribution through a typical sedimentary sequence showing monotonically varying tectonic strain and the induced stresses in rocks of varying stiffness](image)

**Table 1 - Stresses and tectonic strains in Juandah coal measures in the Surat Basin Queensland**

<table>
<thead>
<tr>
<th>Depth</th>
<th>Young's Modulus (MPa)</th>
<th>Poisson's Ratio</th>
<th>Principal Stresses</th>
<th>Tectonic Strains</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Major</td>
<td>Minor</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Major</td>
<td>Minor</td>
</tr>
<tr>
<td>507.49</td>
<td>1315</td>
<td>0.09</td>
<td>1.13</td>
<td>0.79</td>
</tr>
<tr>
<td>3.14E-04</td>
<td>3.66E-05</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>508.89</td>
<td>31646</td>
<td>0.17</td>
<td>11.72</td>
<td>5.01</td>
</tr>
<tr>
<td>3.02E-04</td>
<td>5.32E-05</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

It should be noted that tectonic strains can be quite variable as they are influenced by local geological features.

With the benefit of stress measurements in rocks it is possible to calculate tectonic strains and to apply these values to other strata which have not had stress measured using the known mechanical properties of the rock. This includes the coal seams. There are however some problems with this in that the determination of the coal's mechanical properties is not easy and that the coal has undergone some chemical changes following deposition in the coalification process. It does however provide a first estimate of the stresses and their direction. This may be clarified further by hydrofracture testing to confirm minimum stress levels within the coal.

The processes to arrive at the initial stress calculations in coal are useful. Lacking such information, however, it is possible to estimate the initial stress and focus on the stress changes.
CHANGES IN STRESS BROUGHT ABOUT BY FLUID REMOVAL

Effective stress is the total stress minus the product of fraction (close to unity in a cleated coal) and the fluid pressure as shown in Equation 8.

\[ \sigma' = \sigma_r - \beta * p \] (8)

The tendency for coal lumps to change dimension is associated with gaining or losing gas from coal. This can be measured mechanically. The general trend is for the coal to swell with absorption and shrink with loss of gas. The shape of the strain curve induced by gas pressure may sometimes, but not always, be considered to be similar to the shape of the sorption isotherm curve. An example of such a strain curve is shown in Figure 5. Here the x-axis shows the sorption pressure while the y-axis displays the strain. This example shows a change of 1600 microstrain over a sorption pressure change of 3 MPa.

Table 1 - Stresses and tectonic strains at a proposed mine site in the Northern Bowen Basin

<table>
<thead>
<tr>
<th>Sigra IST (Test Ref)</th>
<th>101.021</th>
<th>103.021</th>
</tr>
</thead>
<tbody>
<tr>
<td>Depth, m</td>
<td>241.10 m</td>
<td>247.30 m</td>
</tr>
<tr>
<td>Location to Seam</td>
<td>Leichhardt Seam Roof</td>
<td>Leichhardt Seam Roof</td>
</tr>
<tr>
<td>Material Description</td>
<td>Fine Grain Sandstone with Dark Grey Siltstone Banding</td>
<td>Fine Grain Sandstone with Dark Grey Siltstone Banding</td>
</tr>
<tr>
<td>Young’s Modulus, MPa</td>
<td>11765</td>
<td>34762</td>
</tr>
<tr>
<td>Poisson’s Ratio</td>
<td>0.12</td>
<td>0.23</td>
</tr>
<tr>
<td>Unconfined Compressive Strength</td>
<td>28.6</td>
<td>100.3</td>
</tr>
<tr>
<td>Angle of Maximum Principal Effective Stress (degrees E of Magnetic North)</td>
<td>33.12(^0)</td>
<td>30.82(^0)</td>
</tr>
<tr>
<td>Maximum Tectonic Strain</td>
<td>7.07E-04</td>
<td>6.22E-04</td>
</tr>
<tr>
<td>Minimum Tectonic Strain</td>
<td>3.57E-04</td>
<td>7.47E-05</td>
</tr>
<tr>
<td>Max Principal Effective Stress, MPa</td>
<td>9.46</td>
<td>24.58</td>
</tr>
<tr>
<td>Min Principal Effective Stress, MPa</td>
<td>5.77</td>
<td>9.10</td>
</tr>
<tr>
<td>Ratio of Maximum Effective Stress over UCS</td>
<td>0.33</td>
<td>0.25</td>
</tr>
</tbody>
</table>

Figure 5 - Example of coal’s dimensional change with gas pressure

The stresses in coal during fluid production are a function of the initial total stress in the coal and the changing fluid pressure and changes in stress brought about by shrinkage of the coal as it gives up gas and dries out. Shrinkage by removal of moisture is reported by Pan et al. (2008). However, whether
actual removal of moisture to achieve drying and related shrinkage occurs in a commercial gas production situation, is of some doubt, though the effect is widely recognized in coals that are drained from underground.

Stress changes in coals are brought about by the removal of fluids. In laterally extensive environments the horizontal dimension remains the same while the ground level drops. Thus the horizontal strain remains constant whilst a reduction in vertical strain occurs. Using Equation 3 and assuming that β is unity (consistent with bedding plane fractures) then the increase in effective vertical stress can be expected to be exactly the same as the reduction in fluid pressure as per Equation 9.

\[ \Delta \sigma_v' = -\Delta P \] (9)

This change in vertical stress causes a corresponding change in horizontal stress given by Equation 10, which is similar to Equation 1.

\[ \Delta \sigma_h = \Delta \sigma_v' \left( \frac{v}{1-v} \right) \] (10)

The effects of fluid pressure reduction on effective horizontal stress are given in Equation 11. They are a combination of the direct effect of reducing fluid pressure (as in the horizontal form of Equation 9), and the effects brought about by the Poisson’s effect as given in Equation 10.

\[ \Delta \sigma_h' = -\Delta P \left( 1 + \left( \frac{v}{1-v} \right) \right) \] (11)

If, however, shrinkage occurs then the effects need to be taken into account as described in Equations 12 and 13.

The horizontal stress changes brought about by shrinkage are:

\[ \Delta \sigma_{shh} = \frac{E}{1-v^2} \left( \Delta \varepsilon_{sh1} + v \Delta \varepsilon_{sh2} \right) \] (12)

\[ \Delta \sigma_{shv} = \frac{E}{1-v^2} \left( \Delta \varepsilon_{shv1} + v \Delta \varepsilon_{shv2} \right) \] (13)

Therefore the net effective changes in horizontal stress caused by fluid removal are given in Equations 14 and 15:

\[ \Delta \sigma_{h1} = \Delta P \left( 1 + \left( \frac{v}{1-v} \right) \right) - \frac{E}{1-v^2} \left( \Delta \varepsilon_{sh1} + v \Delta \varepsilon_{sh2} \right) \] (14)

\[ \Delta \sigma_{h2} = -\Delta P \left( 1 + \left( \frac{v}{1-v} \right) \right) - \frac{E}{1-v^2} \left( \Delta \varepsilon_{shv1} + v \Delta \varepsilon_{shv2} \right) \] (15)

Without shrinkage effects, a reduction in fluid pressure will bring about an increase in effective horizontal stress (per Equation 11). If shrinkage effects occur, then Equations 14 and 15 describe the combined effect. In some cases horizontal effective stress will increase while in others it will decrease.

If one of the effective horizontal stress levels in the coal drops to a low enough value compared to either the other effective horizontal stress, or to the vertical effective stress then the potential for a small scale localised failure may occur. Otherwise this horizontal stress may drop to zero whereupon further shrinkage will lead to direct opening of the cleat.

The equations presented in this paper are formulated in terms of linear elastic parameters for Young’s modulus (\(E\)) and Poisson’s ratio (\(v\)). It should be appreciated that coal is a inhomogeneous rock which may undergo stresses which take it into a non-linear range of behaviour. It is therefore essential in the analysis of the effects of stress change to take into account this behaviour by proceeding with step wise calculations for each increment in fluid pressure change, using as far as can be determined, the tangent values of \(E\) and \(v\) at the appropriate stress levels.
Figure 6 shows the stiffness versus effective stress behaviour of a coal under hydrostatic loading. This is highly non linear. Coals also frequently show a significant difference between vertical and horizontal stiffness.

A STRESS PATH CALCULATION

Information has been used from the proposed Northern Bowen Basin Mine site in the Bowen Basin of Queensland to calculate the stress path. The initial effective stress in the seam has been calculated from the tectonic strains, using an appropriate secant Young’s modulus and Poisson’s ratio. The stress path is shown in Figure 7 and has then been calculated based upon desorption as soon as the fluid pressure is lowered. The calculation uses the shrinkage curve shown in Figure 5.

Figure 6 - Stiffness versus effective mean stress behaviour of a coal sample under hydrostatic loading

Figure 7 - The stress path due to drainage of a Northern Bowen Basin coal

The two horizontal effective stresses increase with a drop in reservoir pressure from 2050 kPa to 1250 kPa. Below this pressure the effects of shrinkage are significant and the effective stress decreases are marked. As the coal in question has a very low permeability the implications of this are important. It is likely that drainage will be very slow until the reservoir pressure is dropped to 1250 kPa whereupon the process of drainage will speed markedly. Getting the coal to this reservoir pressure may require
closely spaced horizontal holes and the use of stimulation such as hydrofracture to enhance initial drainage.

It should be appreciated that the equations which are used to derive the results presented in Figure 7 are based upon uniform drainage over a wide area. The real position from a mining perspective is likely to be different as uniform drainage is unlikely.

THE EFFECT OF VARYING EFFECTIVE STRESS ON PERMEABILITY

It can be readily shown in tests on coal core that the permeability of coal is directly related to the effective stress which exists. This type of relationship is shown in Figure 8. Equation 16 shows a log permeability-effective stress equation can be used to describe this relationship. The permeability of stiffer coals may be expected to be less affected by changing stresses though stiffer coals show much greater stress change associated with shrinkage behaviour.

\[
\log k = \log k_o - \sigma'/b
\]  \hspace{1cm} (16)

In the most dramatic cases it appears that \( b \) may be as low as 3 MPa thus indicating a change of one order of magnitude in permeability with 3 MPa effective stress change.

This type of relationship is supported by core testing (Somerton, et al., 1975; Gray, 1987). Field evidence exists through history matching, albeit with complications associated with the effects of shrinkage of coal and two phase flow effects. It is also supported by the results of permeability tests conducted at various depths in of a number of seams including the Goonyella Middle Seam.

![Permeability Versus Effective Stress](image)

**Figure 8 - A prediction of permeability change with effective stress**

The significance of changing effective cleat width may be appreciated by the theoretically derived Equation 17 (Snow, 1968) for permeability based on cleat width and spacing. Permeability increases as the cube of effective cleat width and is directly proportional to the inverse mean cleat spacing.

\[
k = \left( \frac{w^3}{12S} \right)
\]  \hspace{1cm} (17)

If permeability is measured then it is possible to arrive at an effective cleat width provided that measurements of cleat spacing have also been made. This then provides a basis for predicting how permeability may change when cleats open due the effective horizontal stress dropping below zero.

A PRACTICAL EXAMPLE

Figure 11 approximates the situation as was thought to exist on the basis of field testing at Leichhardt Colliery in the Central Bowen Basin Queensland. It shows where the permeability began at 0.1 md with a 4.2 MPa seam pressure. Initial water drainage takes place to 3.8 MPa when the sorption pressure is reached and gas is emitted from the coal.
In this pressure range the permeability declines due to an increase in effective stress. This trend would continue (blue line) except that in the real case shrinkage occurred (red line) causing a reduction in effective stress and an increase in permeability to 1 md at 2.7 MPa gas pressure. At this pressure zero effective stress exists across the cleats. As gas pressure declines further the permeability increases dramatically to 500 md at 0.5 MPa pressure. This case is extreme but demonstrates the importance of shrinkage.

CONCLUSIONS

This paper describes the process of determining the effective stress path of a coal by determining the initial stress level through the process of fluid removal, incorporating the effects of shrinkage. Permeability of the coal is strongly affected by the effective stress and therefore if this increases then permeability decreases and vice versa. Determining the likely stress path is a key to knowing whether permeability will increase or decrease with drainage. This is particularly important for those with coals in the low permeability range.

Figure 11 - Changes to permeability due to effective stress and shrinkage (from Gray, 1987)

Stress measurement by overcoring in strata adjacent to the coal seam provides information on the initial stress regime. The initial stress conditions can be transferred to adjacent strata of different elastic properties by using the concept of tectonic strain.

Where the coal has low permeability and the effective horizontal stresses do not decrease, but may indeed increase with dropping fluid pressure, then there is sometimes no potential to drain the seam using conventional drilling methods, even if these are assisted by stimulation such as hydrofracture of the in-seam drainage holes. From a mining perspective the only option is to remove material to drop the stress level. This is sometimes achieved by mining an adjacent seam which can be drained and worked thus releasing the problem seam.

REFERENCES

NOMENCLATURE

$b$ is the stress – perm coefficient, Pa$^{-1}$.
$E$ is Young’s modulus, Pa.
$g$ is gravitational acceleration, m/s$^2$.
$k$ is the permeability, m$^2$.
$k_0$ is the permeability at zero effective stress, m$^2$.
$l$ is the length in the direction of flow, m.
$P$ is the pressure, Pa.
$ΔP$ is the change in fluid pressure, Pa.
$S$ is the mean cleat spacing, m.
$v$ is the apparent flow velocity, m/s.
$z$ is the elevation, m.
$w$ is the effective cleat width, m.
$β$ is the fraction representing the proportion of continuous fracture area (approximately unity).

$Δε_{sh1}$ is the strain change due to shrinkage in direction 1 in the horizontal plane. Note a positive value implies shrinkage.
$Δε_{sh2}$ is the strain change due to shrinkage in direction 2 in the horizontal plane. Note a positive value implies shrinkage.
$μ$ is the absolute viscosity, kg m$^{-1}$s$^{-1}$.
$ν$ is Poisson’s ratio, kg m$^3$.
$ρ$ is the density of the fluid (mass/length$^3$).
$σ'$ is the effective stress normal to the cleats, Pa.
$σ_T$ is the total normal stress across a fracture, Pa.
$Δσ_{h1r0}$ is the change in effective horizontal self weight stress, Pa.
$Δσ_e$ is the change in effective vertical stress, Pa.
$Δσ'_{h1}$ is the effective stress change in direction 1 in the horizontal plane, Pa.
$Δσ'_{h2}$ is the effective stress change in direction 2 in the horizontal plane, Pa.
$σ_{sh1}$ is the stress change due to shrinkage in direction 1 in the horizontal plane, Pa.
$σ_{sh2}$ is the stress change due to shrinkage in direction 2 in the horizontal plane, Pa.
**ABSTRACT:** Coal seam gas drainage is affected by many factors which may be generally divided into two groups, geological factors and operational factors. Studies conducted in the Bulli seam found geological factors had a dominant impact on coal seam gas drainage while operational factors had a secondary impact, affecting the amount of optimum gas drainage performance achieved within the limitations imposed by the prevailing geological conditions. Various operational factors, which are controllable by the mine operator and which have a significant impact on gas drainage effectiveness, are presented and recommendations made to improve and optimise gas drainage performance.

**INTRODUCTION**

The use of in-seam drilling ahead of mining for gas drainage was first introduced in Australia in 1980 to reduce the coal seam gas concentrations to levels sufficient to be managed by the mine ventilation system during both the roadway development and longwall coal extraction processes. Since 1980 underground to in-seam (UIS) drilling has evolved from simple rotary drilling rigs to more advanced units which utilise down-hole motors and are capable of drilling borehole in the order of 1,600 m. The use of UIS drilling has expanded throughout the Australian coal mining industry to become the method of choice for underground gas drainage drilling, particularly in mining regions such as the Illawarra which operate at depths in the order of 450-500 m and have substantial surface access constraints which restrict access for surface based methods.

In gassy mines, such as those operating in the Bulli seam, it is common for substantial UIS drilling to be completed ahead of development, with more than 100 000 m drilled annually. The annual cost of such intensive drilling programs, along with supporting infrastructure, may be in the order of $4-6 million. Mechanisms that control and influence coal seam gas drainage are generally not well understood and therefore most coal mine gas drainage programs achieve less than optimum performance. Recent studies to evaluate the effectiveness of such intensive UIS gas drainage programs found that almost half the boreholes delivered little to no benefit to gas content reduction (Black and Aziz, 2008). Where difficult drainage areas are encountered a common response is to increase the borehole density, often allowing minimal drainage time for the additional boreholes; a high cost to achieve generally poor results.

Studies conducted in the Bulli seam in areas where seam gas was found to be extremely difficult to drain from the coal found geological factors, in particular the degree of saturation (DoS) and permeability, had a dominant impact on gas drainage effectiveness, with gas drainage found to be most difficult from deeply undersaturated, low permeability coal. While prevailing geological conditions tend to cap total gas drainage potential operational factors were found to affect the ability of coal seam gas drainage programs to achieve the potential maximum gas production performance.

The operational factors presented in this paper are generally within the control of mine personnel to change as appropriate to optimise total gas drainage performance within the limits imposed by prevailing geological properties.

**SELECTION OF GAS DRAINAGE METHOD**

Many deep, gassy mines rely on UIS drilling to provide boreholes for the purpose of coal seam gas drainage. The UIS method involves drilling boreholes into unmined coal from formed roadways to drain gas from the coal seam ahead of advancing mine development. With the UIS method both drilling and the time available to drain gas from the drilled area (drainage window) fall on the critical path of the mine production schedule. It is typical for the drainage window associated with the use of the UIS method to

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be less than 12 months. In deeply undersaturated and low permeability coal seams the rate of reservoir pressure reduction can be extremely slow, often taking much longer than 12 months to reach the critical desorption pressure. In such conditions UIS gas drainage may not be capable of delivering the necessary gas content reduction within the time available.

The impact of prevailing geological conditions on the rate of gas emission from the coal seam must be understood and considered when selecting an appropriate gas drainage method, or combination of methods. Figure 1 shows the results of a study to determine the likely drainage time required to reduce the in situ gas content to the Level 1 outburst threshold limit using the UIS gas drainage method. In areas where the required drainage time exceeds the available drainage window alternative surface-based drilling and gas drainage methods must be used. A variety of drilling and gas drainage enhancement techniques are available and each is appropriate for use in a particular set of geological conditions. Surface-based drilling and gas drainage enhancement methods for example enable drilling and gas drainage to be conducted independent of mine operations, preferably many years ahead of mining activity, and therefore avoid becoming a critical path activity on the mine production schedule.

Figure 1 - Total time required (days) to reduce seam gas content to Bulli seam Level 1 outburst threshold limit

DESIGN OF BOREHOLE LAYOUT

The orientation of boreholes relative to (a) the principal horizontal stress, (b) the cleats and joints, and (c) the dip of the coal seam, impact on gas drainage performance and therefore must be considered when designing the layout of the gas drainage drilling program. Typically boreholes oriented parallel to the major horizontal stress and perpendicular to the face cleat produce gas at an increased rate. The influence of both stress and cleat orientation on drainage performance should be investigated to determine which is the more dominant factor in the coal seam to be drained. In a study of factors impacting gas drainage performance in the Bulli seam, borehole orientation relative to the major horizontal stress was found to have a more significant impact on gas drainage than orientation relative to cleat.

Figure 2 - Impact of borehole spacing in achieving effective gas drainage
The spacing between boreholes, i.e. drilling density, must also be considered. As shown in Figure 2, should the spacing between boreholes be too great the gas content of the coal seam midway between the drainage boreholes may not be reduced below the required level within the available drainage period. Alternatively if the spacing between boreholes is too low interference occurs and the intensity and cost of the drainage program becomes unnecessarily high.

**DRILLING AND COMPLETION OF DRAINAGE BOREHOLES**

Standpipes of sufficient length to extend beyond the zone of fracturing in the coal rib are necessary to reduce the risk of excessive gas leakage from the rib surrounding the borehole when the borehole has been shut-in and to prevent excessive air leakage through the rib to dilute the drainage gas when the borehole is producing and suction pressure is applied to the borehole. The standpipe should be effectively grouted into the borehole and it is good practice to conduct a pressure test on each standpipe following installation, prior to resuming drilling of the borehole, to determine if leakage paths exist.

As roadway development advances in adjacent panels each of the UIS boreholes is intersected. These open boreholes represent a source of potentially high air dilution entering the gas drainage system. The exposed borehole may be plugged or returned to operation, typically by using a length of hose with the ends inserted and grouted into the coal rib. The grouting of the plugs or hose ends into the coal rib must be to a high standard to minimise air dilution into the drainage range and reduce the rate of gas emission into the mine airways.

Following the introduction of directional drilling and the development of the DDM-MECCA in-hole survey system in the 1990’s there has been little change in the equipment and methods used for drilling UIS gas drainage boreholes. Although industry funded projects have developed several generations of sensors and logging systems to improve drilling guidance and in-hole data collection the development process has stopped short of commercialisation and therefore technology has not been adopted by drill rig operators to improve UIS gas drainage performance.

UIS drilling is presently undertaken using predominantly a slide-drilling technique which involves minimal rotation of the drill rods while a downhole motor is used to rotate the drill bit and a bent-sub assembly is used to aid directional control. The position of the borehole is typically surveyed at 3 m intervals and assessed relative to planned trajectory. As required the drill string is rotated to change the orientation of the bent-sub and therefore the trajectory which the borehole will be drilled for the subsequent 3 m interval. Although the aim of the driller is to maintain the borehole within a relatively small zone surrounding the planned trajectory of the borehole this is not always achieved with installed boreholes often taking a tortuous path with frequent and sometimes dramatic changes in orientation. Also, in the absence of coal interface detection sensors such as directional gamma, the position of the borehole within the coal seam is determined by intentionally extending the borehole to intersect either the roof or the floor. After intersecting the coal interface the drill rods are retracted a short distance and a branch formed from which the borehole continues to be drilled. An example of a directionally drilled inseam borehole shown in section (Figure 3) illustrates the change in position of the borehole within the seam forming multiple peaks and troughs along the length of the borehole.

![Figure 3 - Profile of an inseam directionally drilled borehole (after Brunner et al., 2008)](image-url)
Although generally considered to be an effective and relatively low cost drilling method to support coal seam gas drainage there are inherent deficiencies associated with the existing UIS drilling equipment that have an adverse effect on coal seam gas drainage, such as:

- **Limited borehole length.** The length of in seam directionally drilled boreholes is limited by the development of frictional drag forces along the length of the holes which must be overcome to advance the drill string (Gray, 2000; Thomson and Qzn, 2009). Factors that impact the drag force include borehole geometry, wall roughness, cuttings accumulation, annular pressure and total weight of the drill string (Thomson and Qzn, 2009). There is a limiting condition, known as ‘lockup’, which occurs when the frictional forces developed along the borehole exceed the axial force exerted by the drill rig eventually resulting in helical buckling of the drill pipe within the borehole (Gray, 1994: Thomson and Qzn, 2009). The rate of change of borehole trajectory and the interval between changes in tool face angle have a dramatic impact on friction developed along a borehole and the thrust and pullout force required to drill and extract rods from the borehole (Gray, 1994).

- **Potential loss of downhole equipment.** Where multiple bends exist along the length of a borehole the force required to pull the drill string from the borehole will be greater than the push force required during drilling. When advancing the drill string into the borehole the drill rods are pushed to the outer side of the bends whereas when recovering the drill string the rods are pulled against the inside of the bends whereby increased friction is developed. In such cases it may not be possible to recover the drill string from the borehole making it necessary to abandon the downhole equipment in-hole (Gray, 1994).

- **Increased friction due to cuttings accumulation.** During slide drilling, cuttings fall to the bottom of the borehole forming a bed. The cuttings, along with any additional broken coal from weak zones encountered during drilling, accumulate along the length of the borehole increasing the contact surface area and hence friction developed between the drill rods and the borehole. In cases where the annular pressure exceeds the formation pressure the differential pressure may push the drill string against the side of the borehole further increasing the friction developed during slide drilling. Excessive friction may lead to a condition known as ‘differential sticking’ possibly resulting in the drill string being temporarily stuck in the borehole or complete loss of the downhole equipment (Thomson and Adam, 2007). Figure 4 illustrates the presence of drill cuttings and high annular pressure acting on the drill pipe that lead to differential sticking during slide drilling (Thomson, 2009).

- **Lack of annular pressure monitoring and control.** Measurement of annular pressure within the borehole provides useful information to determine whether drilling is underbalanced, balanced or overbalanced. Where a high differential exists between annular pressure and formation pressure the borehole and surrounding coal may be damaged, affecting drilling performance and future gas drainage. Indication of increasing annular pressure may indicate fines accumulation allowing the drill rig operator to take corrective measures prior to reaching a borehole blocked and lock-up situation.

In seam directional drilling, under normal conditions, is ‘underbalanced’, whereby formation pressure exceeds the annular pressure of the circulating drilling fluid. The higher pressure of the formation in the underbalanced condition creates a pressure differential resulting in the flow of water and gas from the coal seam into the borehole. In underbalanced drilling, particularly where the pressure differential is high and when negotiating zones of weak coal and geological structures, the risk of the borehole collapsing around the drill string is increased. Rapid build-up of material within the borehole may disrupt fluid circulation causing increased annular pressure potentially causing differential sticking, mechanical jamming and borehole failure (Thomson and Qzn, 2009).

Where annular pressure exceeds formation pressure an ‘overbalanced’ condition is created. In an overbalanced condition the higher pressure of the borehole annulus forces drilling fluids and fines into the cleat and pores of the surrounding coal seam forming a ‘skin’ around the wall of the borehole which adversely impacts future gas drainage (Thomson and Qzn, 2009). Where the rate of fluid loss into the surrounding coal seam is high the velocity of drill fluid circulating in the borehole may be insufficient to effectively clear cuttings from the borehole potentially leading to increased friction and bogging of the drill string (Thomson and Adam, 2007). Figure 5 illustrates the nature of fluid and gas flow in underbalanced and overbalanced conditions associated with a localised failure within the borehole (Thomson, 2009).
A borehole pressurisation system was developed under ACARP project C3072 in the late 1990’s however this system was not tested in an underground mine UIS drilling application and to this day UIS drilling continues without any form of monitoring or control of fluid pressure balance relative to the conditions present in the reservoir.

Control of annular pressure may also offer some resistance to the adverse impact of stress induced borehole instability. Figure 6 illustrates the effect of excessive vertical and horizontal stress experienced by a borehole and the resulting stress induced failure (break-out). Creedy et al. (1997) suggested the strength of the material being drilled is the most important factor controlling borehole stability as lower strength materials are more likely to fail, particularly those affected by water. The magnitude and orientation of the stress field relative to the borehole can also have a significant impact on resulting stability. In areas where the coal seam is weak and there is an increased risk of coal failure and potential restriction or blockage of the borehole a liner should be inserted into the borehole.

The presence of water in the coal seam and within a borehole tends to impede gas desorption. Therefore following drilling each borehole should be flushed to remove residual water and coal fines. When drilling using fan patterns there is an increased risk of connection between boreholes due to the high density close to the collar. In such cases drilling fluid may readily pass between boreholes and it may be necessary to delay flushing until all holes in the pattern have been drilled. Where boreholes are aligned down-dip or deep troughs exist along the length of the borehole, subject to groundwater make, in-hole dewatering systems should be installed and used to maintain conditions conducive to gas drainage.

**DESIGN OF GAS RETICULATION SYSTEM**

To avoid discharging drained gas into the mine ventilation system and potentially creating an unsafe condition within the mine a gas reticulation pipe network may be utilised to remove the drained gas from...
the borehole directly to the surface where the gas may be utilised for power generation, flared or discharged directly to atmosphere. Prior to committing to a particular gas reticulation system the location, volume and concentration of the current and future gas sources must be assessed to ensure the pipe network is of sufficient size. Where pipe diameter is too small internal pressure losses will potentially be very high resulting in total gas reticulation capacity being capped at less than is required.

Vacuum is typically applied to the gas reticulation pipe network via a series of liquid ring pumps installed in a surface gas drainage plant. The purpose of the surface vacuum plant is to maintain the pressure throughout the underground pipe network at below atmospheric to prevent gas leaking from the pipes into the underground mine airways. Applying a suction pressure of 10-15 kPa (gauge) to drainage boreholes is considered acceptable. Where increased suction is applied increased gas drainage is unlikely to be achieved whereas there is a far greater risk of air leakage into the system which not only reduces the quality of the drainage gas but also reduces the effective area of the pipes and therefore the capacity of the gas reticulation system, as shown in Figure 7.

![Figure 7 - Reduction in effective area due to fines accumulation (Black and Self, 2007)](image)

The design of the gas drainage reticulation system should also allow for the installation of measuring stations positioned at all major junctions throughout the network to record gas flow and composition. Water traps should also be installed throughout the pipe network, particularly at the bottom of synclines and other low sections where water is likely to accumulate and create a restriction within the network. In order to reduce the risk of water and coal fines entering the gas reticulation pipe network gas/water separators should be installed and maintained to provide an interface between each gas drainage borehole, or group of boreholes, and the pipe range, particularly during the initial production phase of the boreholes. A separator unit, similar to the design shown in Figure 8, could be located near to the boreholes and receive all gas, water and coal fines, produced from the boreholes. Once inside the main body of the unit separation occurs with the seam gas exiting at the top of the unit to the gas drainage range leaving the water and fines to be pumped and drained from the bottom of unit and removed from the site.

![Figure 8 - Conceptual gas, water, coal fines separation unit](image)

**MONITORING AND MANAGEMENT**

Restrictions and blockages can easily occur within gas drainage boreholes due to an inflow of groundwater from the coal seam or failure of the coal seam surrounding the borehole. Such restrictions
or blockages may dramatically slow or even prevent gas drainage and although water accumulation is more likely to occur early in the life of the borehole failure of the coal may occur at any time.

Dedicated resources should be assigned to regularly monitor and actively manage the performance of UIS gas drainage boreholes and the complete gas reticulation network. If the performance and status of the gas reticulation network is not continuously monitored, measurement of the pressure, flow rate and gas composition should be recorded at least weekly at each drill site, each panel entry, the entry to the surface gas drainage plant and all other major junctions throughout the network. The flow rate and composition of gas from each borehole should also be measured weekly. This information will enable all gas production sources to be quantified as well as identify sources of leakage and changes in system performance. Water production and applied suction pressure should also be recorded to enable detailed analysis and understanding of site specific gas drainage characteristics.

An important aspect of data analysis and reporting must be identification of changes in performance, for example a sudden decrease in gas production rate, excessive air dilution, or excessive water production. Regular monitoring enables potential issues to be identified, investigated, and appropriate corrective action to be taken.

**CONCLUSIONS**

From an assessment of operational factors that impact coal seam gas drainage the following is a summary of actions that should be considered by mine operators to optimise the performance of the total gas drainage program.

- Develop an understanding of the gas drainage characteristics of the coal seam to be mined. In particular, gather data and conduct testing to forecast maximum likely gas production based on prevailing geological properties.
- Determine suitable gas drainage method(s) based on a review of the mine production schedule and available gas drainage window.
- Design the drilling program considering that increased gas production is likely from boreholes oriented parallel to principal horizontal stress and perpendicular to the face cleat and joints. Boreholes should ideally be drilled up-dip to facilitate removal of any inflow of groundwater to the borehole. When drilling down-dip is necessary and where troughs are present along the length of the boreholes that may allow water to accumulate in-hole dewatering systems should be installed and maintained.
- Existing UIS drilling equipment and methods have an adverse impact on the drainage of gas from the coal seam due to lack of directional control that necessitates frequent changes in drilling orientation, branching and roof/floor touches. An absence of pressure control has the potential to create unstable conditions within the borehole and damage the permeability of the coal seam surrounding the borehole. The development of sensors and control systems to support automation of drilling guidance, used in conjunction with pressure sensing and control systems, would have a significant impact on increasing drilling productivity, reducing cost per metre drilled and improving total gas drainage performance.
- Standpipes should be of sufficient length to extend beyond the zone of fracturing in the coal rib and be effectively grouted into the coal seam to prevent leakage. Each standpipe should ideally be pressure tested following installation to determine whether leakage paths exist.
- Following the completion of each borehole, or pattern of boreholes, all holes should be flushed to remove drilling fluid and fines as any residual material within the boreholes has the potential to impede gas drainage.
- The gas reticulation pipe range should include pipes of sufficient diameter to cater for all current and future gas production demands. Monitoring stations should be installed throughout the network to record system performance. Water traps should also be installed throughout the pipe network, particularly at the bottom of synclines and other low sections where water is likely to accumulate and create a restriction within the network. All water traps should ideally be of an automated design to eliminate the reliance on manual release of accumulated water from the system.
Where system performance is not continuously monitored, measurement of the pressure, flow rate and gas composition should be recorded at least weekly at each drill site, each panel entry, the entry to the surface gas drainage plant and all other major junctions throughout the network. The flow rate and composition of gas from each borehole should also be measured weekly to enable total gas production and quality to be recorded by source as well as identify leakage and changes in system performance. Water production and applied suction pressure should also be recorded to support detailed analysis and improved understanding of site specific gas drainage characteristics.

REFERENCES


**REVIEW OF GAS EMISSION PREDICTION AND CONTROL METHODS FOR MULTI-SEAM MINING IN CHINESE COAL MINES**

Zhongwei Wang, Ting Ren and Lei Zhang

**ABSTRACT:** In China, the practice of multi-seam mining with high gas emission rates is common, and many coal mines are suffering from major gas management issues. To deal with these problems, empirical formulas for gas emission prediction and gas control methods suitable for site-specific conditions are developed. Specifically, the statistical method and split-source-method are widely used for gas emission prediction. In addition to ventilation, various gas control methods, including cross-measure boreholes, directional long-holes, dedicated roadways and surface gob wells are used for gas drainage. Techniques for integrated extraction of coal and coal seam gas have been developed in these coal mines, resulting in significant economic and environmental benefits. A detailed review of gas emission prediction and control methods adopted in highly gassy Chinese coal mines is presented.

**INTRODUCTION**

Gas-related disasters have long been recognised as one of the most serious threats to mine safety in underground coal mines in China, especially for those extracting multi-seams where gas migration from adjacent seam and gas-bearing strata may result in unexpected or uncontrolled gas issues. According to a recent survey by the Ministry of Land and Natural Resources (http://www.to-gd.com/webpage/2010-07-12/gd559381.htm), China has a mineable coal reserve of 204 billion tons, most of which is associated with multi-seam conditions.

Due to various specific coalifications, some mining areas, typically including Yangquan, Huainan, Tiefa and Pingdingshan, have been suffering from high gas emission rates. To deal with the gas issues in these gassy mines, an accurate gas emission prediction is essential for mine ventilation design and the deployment of suitable gas control methods. These mining areas have developed empirical formulations under their site conditions, and the gas control methods vary from site to site as a result of various geological conditions, reservoir characteristics and mining methods.

Although gas control techniques have been widely improved worldwide either by ventilation or gas drainage, they may not suit for Chinese high gassy coal mines due to the complex site conditions characterized by low permeability of the coal seams, deep burial or high geostress. Many coal mines in the east of China have developed into a deep mining stage with mining depth exceeding 1000 m. It is also estimated that the mining depth will increase at a rate of 8-12 m annually providing a great challenge to the existing gas control techniques. Table 1 describes the general exploitation conditions of typical high gassy multi-seam mining areas in China. It can be seen from the table that permeability of coal seams is critically low. For the other mining areas, the permeability of coal seams is generally between $10^{-1}$-10^{-4} md, leading to poor performance of pre-drainage technique and the development of post-drainage methods either through cross-measure boreholes or dedicated gas drainage roadways.

**Table 1 - Gas related parameters in typical high gassy mining areas (You, 2008a; Yuan, 2008a; Li and Liu, 2005; Huang and Yang)**

<table>
<thead>
<tr>
<th>Mining area</th>
<th>No. of coal seams /minable coal seams</th>
<th>Overburden (m)</th>
<th>Gas content (m³/t)</th>
<th>Gas pressure (MPa)</th>
<th>Permeability (md)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Yangquan</td>
<td>16/3</td>
<td>150-500</td>
<td>7.13-21.73</td>
<td>0.25-2.3</td>
<td>3.75*10^{-4}</td>
</tr>
<tr>
<td>Huainan</td>
<td>8/15</td>
<td>400-1500</td>
<td>10-38</td>
<td>4-8</td>
<td>2.75*10^{-6}</td>
</tr>
<tr>
<td>Tiefa</td>
<td>13/1</td>
<td>600-1200</td>
<td>8.13-24.8</td>
<td>0.85-3.92</td>
<td>2.75<em>10^{-1} - 8.73</em>10^{-4}</td>
</tr>
<tr>
<td>Pingdingshan</td>
<td>8/1</td>
<td>300-1200</td>
<td>6-15</td>
<td>0.6-2.55</td>
<td>-</td>
</tr>
<tr>
<td>Huaibei</td>
<td>8/4</td>
<td>400-1300</td>
<td>6-22</td>
<td>0.1-4.5</td>
<td>0.02-0.3</td>
</tr>
<tr>
<td>Songzao</td>
<td>12/3</td>
<td>200-500</td>
<td>17.1-29</td>
<td>1.2-4.5</td>
<td>1.43<em>10^{-4} - 7.96</em>10^{-4}</td>
</tr>
</tbody>
</table>

Gas emission prediction and control methods adopted in typical high gassy mining areas in China are discussed.
GAS EMISSION PREDICTION

Gas emission prediction is a basic parameter required for the design of mine ventilation systems and corresponding panel layouts, and the selection of proper and effective gas drainage technique. The accurate prediction of gas emission therefore plays a significant role especially for multi-seam mining where gas emission into a working face may increase rapidly compared with single seam mining. Furthermore, for production mines, gas emission must be checked regularly to ensure that its ventilation system and gas drainage system are sufficient enough to deal with the gas issues due to increasing mining depth and changing geological conditions.

Generally, two main methods are employed in the process of gas emission prediction: a statistical method and a split-source method, specifically, a statistical method is usually used for predicting gas emission of an entire coal mine, whilst a split-source method can be used for both a coal mine and a particular active longwall face.

Statistical method

Statistical methods have been widely adopted in China for mine gas emission prediction. This method involves the collection of gas data from previous experience of a mine, a particular area of a mine or of neighboring mines where similar mining methods are being used in similar geological conditions, thus establishing the relation between relative gas emission and mining depth by regression analysis. Figure 1 shows some typical curves obtained using this method (Yu, 2005).

![Figure 1 - Relationship between relative gas emission and mining depth](image)

(a) 4th mine of Hebi  (b) Xieer mine of Huainan

Based on this relationship, the gas emission for a further mining depth can be predicted empirically by extending the line to a proper depth provided that the relationship has been obtained. The accuracy strongly depends on the precision of previous data from which the relationship is founded and the similarity of geological and mining conditions. It is noted that this method is most applicable for the methane-bearing zone, and the extended prediction for seams at greater depth should be discounted with the decrease of seam dip within 100 to 200 m, otherwise, significant errors may occur (Yu, 1992).

Split-source method

Gas emission can also be predicted by the analysis of emission contributing sources. A typical grouping of gas sources in a coal mine is shown in Figure 2, from which it can be seen that gas liberated into an active longwall face can be divided into two main sources, the mined coal seam and adjacent gas-bearing strata (overlying and underlying) (State Administration of Work Safety, 2006). Thus, most of the split-source methods for longwall gas emission are similar, what makes the difference is the various methods of determining the quantity of gas released from overlying and underlying seams or gas emission degree.

Yu et al. (2000) demonstrated that gas released from rib, excavated coal and goaf were the main contributors to longwall face emission. For the first two parts which can be attributed to emission from the mined seam, the following equation was proposed:

\[ q_m = C \times X \]  (1)
Where, $q_m$ is the relative gas emission from mined coal seam, in $m^3/t$; $X$ is the gas content of mined coal seam, in $m^3/t$; C is the degassing coefficient of mined coal seam, and is generally between 0.4-0.8 for thin and medium thickness coal seams.

$$q_g = C_g \times v \times \frac{mL}{24 \times 60} \sum_{i=1}^{n} \frac{m_i}{m} X_i \eta_i$$  \hspace{1cm} (2)

Where, $C_g$ is the coefficient determined by ventilation system ($\leq 1$);
- $v$ is advance rate of face, m/d;
- $av^2$ is the correction factor taking into account the lag of roof and floor break and the displacement relative to time and space;
- $m$ is the mining height, in m;
- $L$ is face length, in m;
- $\delta$ is coal density, in t/m$^3$;
- $m_i$, $X_i$ and $\eta_i$ are thickness, gas content and emission degree of adjacent gas-bearing strata, respectively.

Figure 3 gives a typical emission degree curve developed in the Yangquan and Huainan mining areas.

By taking into account the residual gas content, the following formula was proposed (Yu, 2005; State Administration of Work Safety, 2006; Cheng, et al., 2006):

$$q = q_m + q_a = k \frac{M}{m} (X_0 - X_c) + \sum_{i=1}^{n} \frac{M_i}{m} \eta_i (X_{a_i} - X_{ci})$$  \hspace{1cm} (3)
Where, $q$, $q_m$, and $q_a$ are the gas emission quantities of longwall face, mined seam and adjacent strata, in $m^3/t$, respectively; $k$ is the influencing coefficient of gas emission from mined seam, and it is control by three factors ($k = k_1 \cdot k_2 \cdot k_3$), where $k_1$ is coefficient of gas emission from surrounding strata, and $k_1=1.2$ for fully caving; $k_2$ is coefficient of gas emission from coal left in face, and it is the inverse of the face recovery ratio; $k_3$ is the influencing coefficient of pre-gas emission in the process of roadway development, and for retreat longwall mining, $k_3 = (L - 2b)/L$, where $L$ is face length, $b$ is the width of pre-gas emission in roadway); $m$ and $M$ are the mining height and seam thickness respectively, in $m$; $X_0$ is the initial gas content of mined coal seam, in $m^3/t$; $X_c$ is residual gas content of mined coal seam, in $m^3/t$; $M_i$ is thickness of the $i$ adjacent seam, $X_{0i}$ is the initial gas content of the $i$ adjacent seam, in $m^3/t$; $X_{ci}$ is the residual gas content of the $i$ adjacent seam, $m^3/t$; $\eta_i$ is the percentage of gas emitted from adjacent seam $i$.

Obviously, the accuracy of the two methods depends on the coefficients adopted which are different in different coal mines, and the quality of the gas data-set is critical to the prediction accuracy.

GAS CONTROL METHODS

It is acknowledged that ventilation is the most basic and common method for gas control, and for less gassy mines, ventilation is normally sufficient to dilute the gas concentration below the statutory safety limit; however, efficient drainage strategies are necessary for highly gassy coal mines where large quantities of gas may migrate into underground workings from adjacent strata and ventilation dilution capacity is limited in this situation. Therefore, a reasonable combination of a well-designed ventilation system and an efficient gas drainage system is critical to ensure a safe working environment. Figure 4 shows some of the gas control techniques used for multi-seam mining.

![Figure 4 - Gas control methods developed in Chinese coal mines for multi-seam mining](image)

‘U+L’ ventilation scheme based gas control technique

Based on the ‘U+L’ ventilation scheme, cross-measure boreholes can be drilled towards the fractured zone from the exterior tailgate (the second return roadway) with an interval of 15-35 m, as shown in Figure 5.

Firstly, gas released from the underlying strata and coal left in the goaf will flow through the cut through kept open behind face instead of coming back to the face which usually results in high gas concentration at the intersection of longwall face and return roadway. In other words, this ventilation scheme mainly deals with goaf gas emitted from underlying seams, mined seam and overlying seams within the caved zone. It is noted that the interval between cut throughs impacts on the performance of the exterior tailgate as a result of re-compaction in the goaf, and large quantities of goaf gas may report to face again when the cut through behind face is too far from face line.
Thus, cross measure boreholes are drilled to intercept gas desorbed from seams in the fractured zone before migrating into goaf and working area. It is suggested that the interval of boreholes should be in line with the periodic weighing interval of roof caving, ensuring that at least one borehole is working when large quantities of gas migrate into the working face immediately after periodical roof collapse. The effective drainage radius of boreholes should also be considered while designing intervals (Yu, 1992). The diameter of boreholes is generally between 73 mm and 300 mm according to available drilling equipment on site, the borehole length depends on the distance of the gas-bearing strata from the mined seam. The elevation angle, as shown in Figure 6, is another significant parameter, affecting performance of a borehole, and can be determined by,

$$\tan(\alpha + \beta) = \frac{h}{h\cot(\varphi + \alpha) + b}$$

(4)

Where, $\alpha$ is the seam angle, $\beta$ is the angle between borehole and horizontal, $h$ is the vertical distance between the adjacent seam and the mined seam, $\varphi$ is the de-stressing angle of strata after excavation, and $b$ is the pillar width.

Operational experiences indicates that borehole suction pressure should be kept between 6.7 and 13.3 kPa, and sealing length no less than 2-5 m.

![Figure 5](image-url) ‘U+L’ ventilation scheme combined with cross measure boreholes (You, et al., 2008a)

![Figure 6](image-url) Determination of elevation angle (Yu, 2005)

Field trials carried out at Yangquan mining area demonstrated that the gas recovery ratio of adjacent coal seams could reach 60-70% using boreholes with diameter of 200 mm (Zhang and Cheng). It should be pointed out that although the exterior tailgate has a great significance in the dilution of gas, this in turn leads to a major disadvantage of this technique, i.e., the self-heating or oxidation of coal in the goaf due to air leakage through the cut through behind the face, consequently, this technique is limited in its use for spontaneous combustion prone coal seams. Slotted casing must be used where soft rock especially water-swelling mudstone exists in the roof, the water discharge system is also critical in this case. When production of a longwall panel increases or gas emission from adjacent coal seams is too large and cross measure boreholes are not sufficient to prevent gas from migrating into working panels, a special roadway, which is driven as an inclined high level roadway for gas drainage, is developed from the exterior tailgate. While reaching the target drainage seam level in the fractured zone, it is developed along the seam and extended about 25 to 40 m with a cross section of 3-4 m$^2$. Then the roadway is sealed at the bottom and a pipeline installed. A typical layout and parameters of roadway involved in this technique can be found in Figure 7, from which the following formula can be obtained,

$$h_1 = b \tan{\beta} = b \times (h_c + \Delta h_c)/(a + b)$$

(5)
Where,  $h_3$ is the height that the inclined roadway exceeding the pillar,  $b$ is pillar width about 20-25 m,  $\beta$ is the elevation angle of roadway about 40-45°,  $h_c$ is height of caved zone,  $\Delta h_c$ is a safety height preventing the horizontal roadway from broken, and is taken as 1-1.5 times of mining height,  $L$ and  $L'$ are safety distance ensuring the inclined part to be intact,  $\varphi$ and  $\gamma$ are destressing angle and broken angle of strata after excavation respectively (You, et al., 2008a).

The interval between inclined roadways is much greater than that between cross-measure boreholes due to its large cross section and varies between 150 and 250 m. Field monitoring data of this drainage technique indicates that the average gas concentration captured could reach 71.79%, and the average gas flow rate $31.14 \, m^3/min$, leading to a 74.28% gas recovery ratio from adjacent coal seams.

Compared with the ‘U’ ventilation scheme, the ‘U+L’ ventilation scheme gas drainage technique is superior in terms of the large dilution capacity of the exterior tailgate, which could account for 57.67% of the total gas handled by ventilation, and the ease of drilling cross-measure boreholes or inclined high level roadway development and pipeline installation in the exterior tailgate. The occurrence of gas concentration exceeding the statutory limit at the upper corner has been improved greatly. However, a serious problem induced by this technique is the increased risk of oxidation in the goaf especially when cut-throughs behind the panel are not sealed tightly and timely.

‘U+I’ ventilation scheme based gas control technique

To overcome the shortcoming of ‘U+L’ ventilation scheme, the innovative ‘U+I’ ventilation scheme is adopted during the exploitation of No.15 coal seam which is a spontaneous combustion prone seam in Yangquan mining area. It can be seen from Figure 8 that the average thickness of No.15 coal seam is 6.52 m, and within 60 m above the seam, is the $K_3$, $K_4$ limestone, where fractures and Karst caves are well developed, and are filled with gas. The gas content of No.12 coal seam is up to $14.75 \, m^3/t$, which also poses a threat to the operation of No.15 seam. It should be pointed out that the top coal caving process is another factor influencing the gas control method in the panel. Under this condition, a strike high level roadway along the panel and interior tailgate are developed to deal with gas emitted from the overlying gas bearing strata and the top coal or goaf, respectively.

A general layout of this technique is illustrated in Figure 9. The interior tailgate is developed in the top coal along the roof about 15-25 m away from the return roadway, and the strike high level roadway parallel to the return roadway is developed in the fractured zone generally 40-60 m above the mined seam and about a third of the face length away from the return roadway horizontally (Zhu, et al., 1997).
The interior tailgate provides a new or negative pressure outlet besides the return roadway along the panel, and takes advantage of the collapse as the panel advances but is at least 5 m behind the coal caving line as it is developed in the top coal; consequently, this interior tailgate always performs better at collecting gas emitted from goaf compared with the return roadway and even the exterior tailgate developed in the ‘U+L’ ventilation scheme, and this has been verified by field data showing a dilution capacity of 75.24% total ventilation gas, 17% higher than that of the exterior tailgate [You, et al., 2008a]. In addition, the interior tailgate is easier to maintain than the exterior tailgate, cut throughs are neglected under this scheme. The strike high level roadway developed above the mined seam with a cross section of 4 to 5 m² is superior to the inclined high level roadway in terms of capture of gas desorbed from the adjacent gas-bearing strata as it works with a relatively stable and efficient drainage all the time during panel retreat.

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**Figure 8 - Typical stratigraphic column of the strata and coal seams in Yangquan mining area (You, et al., 2008a)**

<table>
<thead>
<tr>
<th>strata</th>
<th>gross thickness (m)</th>
<th>seam thickness (m)</th>
<th>column 1:200</th>
<th>rock name</th>
<th>petrographic description</th>
</tr>
</thead>
<tbody>
<tr>
<td>K4 limestone</td>
<td>2.00</td>
<td>1.20</td>
<td></td>
<td>gray, with much clay</td>
<td></td>
</tr>
<tr>
<td>11# coal seam</td>
<td>2.20</td>
<td>1.10</td>
<td></td>
<td>gray, with plant fossil</td>
<td></td>
</tr>
<tr>
<td>K3 limestone</td>
<td>3.30</td>
<td>2.30</td>
<td></td>
<td>gray, with much clay</td>
<td></td>
</tr>
<tr>
<td>siltstone</td>
<td>6.40</td>
<td>4.60</td>
<td></td>
<td>gray, white siltstone</td>
<td></td>
</tr>
<tr>
<td>gray mudstone</td>
<td>11.40</td>
<td>9.40</td>
<td></td>
<td>gray, black siltstone</td>
<td></td>
</tr>
<tr>
<td>siltstone</td>
<td>13.11</td>
<td>11.00</td>
<td></td>
<td>gray, white siltstone</td>
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</tr>
<tr>
<td>fine sandstone</td>
<td>16.04</td>
<td>14.70</td>
<td></td>
<td>gray, black siltstone</td>
<td></td>
</tr>
<tr>
<td>gray mudstone</td>
<td>22.64</td>
<td>20.60</td>
<td></td>
<td>gray, black siltstone</td>
<td></td>
</tr>
<tr>
<td>gray medium sandstone</td>
<td>37.37</td>
<td>35.00</td>
<td></td>
<td>gray medium sandstone, partially fine sandstone</td>
<td></td>
</tr>
<tr>
<td>gray mudstone</td>
<td>42.77</td>
<td>40.00</td>
<td></td>
<td>gray, black siltstone</td>
<td></td>
</tr>
<tr>
<td>gray mudstone</td>
<td>45.59</td>
<td>43.50</td>
<td></td>
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<tr>
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<td>46.60</td>
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<td></td>
</tr>
<tr>
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<td>49.71</td>
<td>47.70</td>
<td></td>
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</tr>
<tr>
<td>gray mudstone</td>
<td>52.94</td>
<td>50.90</td>
<td></td>
<td>gray, black siltstone</td>
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</tr>
<tr>
<td>gray mudstone</td>
<td>54.97</td>
<td>52.95</td>
<td></td>
<td>gray, black siltstone</td>
<td></td>
</tr>
<tr>
<td>gray mudstone</td>
<td>55.25</td>
<td>53.20</td>
<td></td>
<td>gray, black siltstone</td>
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<tr>
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<td>gray mudstone</td>
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<td>gray mudstone</td>
<td>69.87</td>
<td>67.50</td>
<td></td>
<td>gray, black siltstone</td>
<td></td>
</tr>
</tbody>
</table>

**Figure 9 - ‘U+I’ ventilation scheme based gas control technique (You, 2008b)**

The interior tailgate provides a new or negative pressure outlet besides the return roadway along the panel, and takes advantage of the collapse as the panel advances but is at least 5 m behind the coal caving line as it is developed in the top coal; consequently, this interior tailgate always performs better at collecting gas emitted from goaf compared with the return roadway and even the exterior tailgate developed in the ‘U+L’ ventilation scheme, and this has been verified by field data showing a dilution capacity of 75.24% total ventilation gas, 17% higher than that of the exterior tailgate [You, et al., 2008a]. In addition, the interior tailgate is easier to maintain than the exterior tailgate, cut throughs are neglected under this scheme. The strike high level roadway developed above the mined seam with a cross section of 4 to 5 m² is superior to the inclined high level roadway in terms of capture of gas desorbed from the adjacent gas-bearing strata as it works with a relatively stable and efficient drainage all the time during panel retreat.
Numerous field trials have been carried out in the Yangquan mining area and this technique has now developed into a main pattern of integrated exploitation of coal and coal bed methane for multi-seam mining in high gassiness and spontaneous combustion prone seams. In these field trials, gas recovery of the strike high level roadway could rise to 80-90%, and an average gas flow rate of 40-60 m³/min was achievable (Zhao, 1996). However, one limitation of the technique is that the interior tailgate will be difficult to maintain (or develop) in soft (or thin) coal seams where it should be developed, i.e., this technique is applicable for a seam thickness of 5.5 m. Meanwhile, it is strongly recommended that the mining height under the interior tailgate be carefully controlled and the flipper of chocks be extended in time to avoid its collapse ahead of the face line. Another factor that should be taken into account is the gas issues encountered in development of the strike high level roadway, the absence of a suitable adjacent seam where it can be developed also challenges this technique because of the large rock roadway development work and the relatively high cost, but it is justified for creating a safe working environment.

The 'U+I' ventilation scheme should be improved when the face width increases significantly. Field trials of K8206 panel in Yangquan 3rd mine, where the face width reached 252.2 m and gas concentration in the return roadway was always 1.0-1.2%, indicated that the single interior tailgate was not sufficient enough to dilute gas emitted from the goaf because of the increase of destressing zones both horizontally and vertically. The increased air quantity and ventilation pressure also leads to the flushing of a high volume of gas emission from the goaf. As a result, another interior tailgate can be developed, forming the 'U+II' ventilation scheme. Field trials had been carried out at K8205 panel, two interior tailgates with a distance of 28 m and 89 m away from the return roadway respectively were developed in the top coal along the immediate roof. Gas monitoring data demonstrated that exceeding the limit of gas concentration in the return roadway rarely occurred, and the dilution capacity of the tailgate was enhanced.

‘Y’ ventilation scheme based gas control technique

The gas control methods mentioned above may be not applicable in the Huainan mining area where coal and gas outbursts have become a major threat due to the high gas pressure (4-6 MPa) and burial depth (700-800 m). Low permeability (generally $2.75 \times 10^{-5}$ mD) and high geostress greatly limits the effectiveness of a pre-drainage strategy, thus, various post-drainage techniques have been developed. One of the most efficient and cost-effective methods was the ‘Y’ ventilation scheme based gas control technique without a coal pillar. From the perspective of outburst prevention, it is a protective mining method, which involves firstly mining a seam with low gas pressure and content to destress the outburst prone seams above and below, capture as much gas as possible, and finally remove the outburst risk of the protected seams (Yu, 1992).

It can be seen from Figure 10 that the ‘Y’ ventilation scheme is a variant of ‘U’ scheme, a bleeder road behind the panel is maintained for air return, and the two other roadways ahead of the panel are used for intake air. The gas flow and air leakage patterns in the goaf under the ‘Y’ ventilation scheme vary from the previously mentioned ‘U+L’ and ‘U+I’ scheme. Firstly, as the two roadways ahead of face act as intakes, there is less air exchange between the goaf and face, thus less gas migrates into the face compared with other schemes, protecting the panel from the threat of goaf gas. Gas accumulation issues around the tailgate corner, which generally accounts for a large proportion of production delay can be well handled (Yuan, 2008b); secondly, as a result of lower ventilation pressure and the goaf gas buoyancy effect, large quantities of gas accumulate along the retained roadway in the goaf, benefiting long term high purity gas drainage.

![Diagram of ‘Y’ ventilation scheme.](image)

**Figure 10 - ‘Y’ ventilation scheme based gas control technique (Yuan, 2008b)**
The retained roadway provides a gallery from which cross-measure boreholes can be drilled to capture large quantity of desorbed gas. Two or three upward boreholes can be drilled in each drilling gallery to reach target seams where vertical fractures are not well developed in the bending zone, so the borehole length depends on the vertical distance and elevation angle, both of which can be determined through Figure 5, borehole diameter is generally more than 90 mm, and the spacing between galleries about 20 m. Downward boreholes should also be drilled to drain gas desorbed from underlying seams; however, the destressing effect may not be as good as that for overlying seams the same distance from the mined seam. Field investigations show that the destressed region in the Huainan mining area may extend up to 130-150 m in the roof and 80-100 m in the floor of the mined seam respectively (Yuan, 2008b), therefore a shorter drainage gallery interval of 10 m is recommended. This ‘Y’ ventilation scheme based gas control technique has been adopted in the Xinzhhuangzi mine and the Guqiao mine.

Surface well drainage technique

Besides various gas control techniques conducted underground, surface well drainage has also recently been developed as an effective method in some mining areas, such as Tiefa, Huaibei and Huainan. Initially, the surface well drainage technique was limited in use as a kind of pre-drainage technique due to the low permeability of coal seams in most mines, however, recognising the effect of mining induced stress relief and permeability increase, surface wells are now widely used to capture stress relief gas desorbed from remote protected seams and goaf gas (sang, et al., 2010). Structurally, the surface wells are almost the same as shown in Figure 11, including surface casing, intermediate casing and slotted casing for gas production (Li and Liu, 2005; Sang, et al., 2010). Research results and field tests demonstrated that the ‘O’ shape compaction will be formed in the goaf as panel retreats, and it is suggested that the wells should be located close to the return roadway and the bottom should reach the ‘O’ circle where large volumes of gas accumulated (Xu and Qian, 2000). The bottom elevation of the well is usually 5 to 10 m above the mined coal seam. Depending upon specific site conditions, the diameter of the three borehole casing section is 299 mm, 177.8 mm, and 139.7 mm, respectively (Yuan, 2004; Li and Liu, 2005). It is accepted that the overlap of intermediate casing and slotted casing should be no less than 10 m, and the length of slotted casing should be 35-45 m. Spacing between surface wells is generally 200-400 m depending on their effective radius under specific site conditions, and the first surface well is usually 40-80 m away from the start up line, suction pressure is 40-60 kPa.

![Figure 11 - Typical surface well location and structure (Li and Liu, 2005; Sang, et al., 2010)](image)

Early trials of surface well gas drainage were conducted at N1405 panel of Daxing mine in the Tiefa mining area, where three surface wells with an interval of 150 m were implemented, and they were 50 m away from the return roadway, the first well was 60 m away from the start up line. Results showed that within 280 days about 2.9 Mm³ of methane was captured with a gas concentration of more than 83% (Li and Liu, 2005). However, the results were not always satisfying in some mines where well collapse is the main obstructive factor, for instance, two wells drilled in Zhangbei mine of the Huainan mining area collapsed as the panel passed them by 54 m and 49 m, respectively (Liang, 2007). Therefore, further research on both theory and practice of surface well gas drainage is necessary, especially the stability of the surface well.
Other gas control techniques

There are also some other gas control techniques being used in Chinese coal mines, for instance, cross-measure boreholes are conducted from drilling galleries in return roadways (Figure 12a) (Yu, 1992; Yuan, 2000), and this is applicable under close distance coal seam mining condition where desorbed gas may migrate through the goaf into the working environment; in some outburst prone mines, dedicated roadways are developed to eliminate the outburst risk (Figure 12b) (Huang and Yang); long horizontal boreholes parallel to roadway are also employed in certain mines for the purpose of replacing strike high level roadway or interior tailgate (Figure 12c) (Li and Hu, 2009). However, for various reasons, these gas control techniques are not widely adopted in China.

![Image of gas control techniques](image)

**Figure 12** - Some other gas control techniques (Yu, 1992; Huang and Yang; Li and Hu, 2009)

**CONCLUSIONS**

In order to effectively manage gas emission hazards, gas emission rates must be well understood and predicted for the choice of suitable gas control techniques. Generally, two main methods: statistical method and split-source method are adopted for gas emission prediction of coal mine and longwall face, and the accuracy of both methods is highly dependent on the coefficients adopted. Based on various ventilation schemes, both cross measure boreholes and dedicated roadways are widely employed to capture gas desorbed from overlying and underlying seams. For cross-measure boreholes, the interval, elevation angle, length and sealing are the main factors influencing its performance. As to the strike and inclined high level roadways, the horizontal and vertical distance from the return roadway and the mined seam are critical for their performance. Surface wells have been developed as an effective drainage method for stress-relief gas desorbed from the protected seam in the bending zone, where vertical fractures are not well developed, and for goaf gas.
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EFFECT OF MAGMA INTRUSION ON COAL GAS OUTBURST INDEXES IN CHINA

Liang Wang, Yuanping Cheng, Jingyu Jiang, Shengli Kong, Haina Jiang, Xiaolei Zhang

ABSTRACT: Numerous coal and gas outburst disasters in China were caused by magma intrusion. Several outburst accidents show that the affected region of magma intrusion is the key area with abnormal gas occurrence and gas outburst disasters. The intrusive igneous rock's occurrence, lithology and distribution form have an important impact on outburst indexes. Laboratory tests and field experimental research, analysed coal reservoirs adsorption characteristics under the function of high temperature pyrolysis and metamorphism caused by layered magma intrusion, and revealed internal relations between coal seam adsorption characteristics and outburst indexes, which were verified in the field. The results showed that under the additive effects of pyrolysis and metamorphism, the metamorphic degree of the coal seam got higher the closer the distance to the igneous rock, the adsorption characteristics of the metamorphic coal were positively correlated with gas outburst indexes. When the magma layer eroded the coal seam, along with the distance from the igneous rock, gas outburst indexes increased first, then decreased. When the layered magma intruded into the coal seam roof, coal seam gas was trapped by igneous rock, and gas outburst indexes were commonly higher, which decreased with the distance away from the igneous rock. When the layered magma intruded into the coal seam floor, the effect of high temperature pyrolysis was the strongest, and the gas outburst indexes were determined by the lithological characters of the roof.

INTRODUCTION

During the lengthy geologic history, magma action is quite frequent in China. Especially since the Yanshan movement in Mesozoic, crustal movement in eastern China became more intense, and magma action got more widespread. The magma intrusion provides a high temperature and high pressure environment for coal seams, which promotes the thermal evolution of coal seams, making low rank bitumite change to katogene metamorphism under the baking of magma, and speeding up the formation of gas, and bringing huge changes in metamorphism of coal, pore structure, adsorption-desorption characteristics, and coal structure.

Numerous coal and gas outburst disasters in China were caused by magma intrusion, in such mining areas as Hebi, Zhengzhou, Yaojie, Huaibe, Huainan, Hegang. Several outburst accidents show that the affected region of magma intrusion is the key area for abnormal gas emission and outburst accidents. There are two main types of occurrence of magma in China: dyke and sill. The dyke, formed by magma penetrated coal-rock mass through faults or fissures as flowing channels, is the vertical or skew intrusion with coal-rock bedding. The sill, formed by magma intruded into the roof, or the floor, even the coal seam through the bedding planes, are widely distributed in China, which can cause heavy damages to the coal seam in different shapes (Zhang, 2008). The size of magma, the intrusion place, the shape of magma, the structure of coal and coalfield, and such other factors could control the physical properties and characteristics of coal, gas occurrence and gas outbursts.

Scholars at home and abroad have done a lot research on the coal quality change, in the metamorphic aureole zone intruded by magma, Qin and Xu (1984); Yuan (2000); Wang and Zhang (2006); Liu et al. (2007); Shen (2008); Fredericks et al. (1985); Stewart et al. (2005); Dai and Ren (2007); Saghafi et al. (2008); Susan et al. (2009). They analysed the degree of coal metamorphism by using proximate analysis, elemental analysis, microscopic coal petrology characteristics and some other chemical process indexes, and considered that large quantities of volatile components and pressure caused by the coal thermal decomposition process would deteriorate the coal quantity. Golab and Carr (2004) compared the inorganic components of normal coal and the coal near the magma dyke in Australia’s Dartbrook Coal Mine. They found that metamorphic coal was rich in inorganic elements, mainly silicon aluminates, which could be used as a guidance to evaluate the affection scope of magma metamorphic
Underground Coal Operators' Conference
The AusIMM Illawarra Branch

Raymond and Murchison (1988) found that during the coal-forming, the degree of coal compaction and the water capacity of pore had a great impact on the metamorphic grade of coal intruded by magma. The lower the maturity and compaction, the less metamorphism action on humid coal. Wang et al. (2008) simulated the baking effect on coal seam produced by the tectonic-heat events by using a pyrolysis analyser, the results showed that the number and dimensions of pore in coal rock increased obviously after thermal baking. Gurba and Weber (2001), Saghaﬁ et al. (2008) found that, gas adsorption capacity, porosity, gas content and gas diffusion rates would correspondingly increased with the enhanced metamorphic degree of coal caused by magma intrusion, and the generating gas was stored in coal seams which were trapped by the igneous rock.

Based on geology research on gas emission sites recently, magma intrusion always caused gas outburst indexes to exceed critical values, such as the initial gas releasing rate $\Delta p$ and the coal sturdiness coefficient $f$, and to produce gas and CO$_2$, all of which increased the possibility of coal spontaneous combustion and gas outburst accidents (Golab and Carr, 2004). Gas emission intensity near intrusive igneous rock usually greatly increased, especially when the permeability of igneous rock was low, which made a large quantity of the generating gas trapped, and greatly increased gas pressure and gas content (Saghaﬁ, et al., 2008; Golab and Carr, 2004).

Nowadays, it could be found that the magma intrusion researches were mainly focusing on coal quality change, porosity evolution, and so on. The adsorption characteristics, variation of gas outburst indexes and the relation among the gas parameters were not discussed in detail, and the gas outburst index data measured in field and laboratory with pertinence were short, which result ed in that we were unable to master gas occurrence law in magma intrusion area.

The distribution of sills is widespread in China, and the sill intrusion contributes to the damage to coal seams. By laboratory tests and field experimental research, coal reservoirs adsorption characteristics under the function of high temperature pyrolysis and metamorphism caused by layered magma intrusion were analysed.

REGIONAL STRUCTURE EVOLUTION AND MAGMA INTRUSION CHARACTERISTICS IN HUAIBEI COALFIELD

There is an affinity between magma action and geological structure, especially the fracture structure that plays a controlling role in the distribution of a magma intrusion. Magma is a viscous lava, and it always moves to the direction of least pressure in the tectonic stress field. Tensional and wrench fractures with a good opening and small lateral pressure are beneficial to magma intrusion (Zhang, 2008). Coal is a macromolecular compound mainly consisted of carbon, hydrogen, oxygen and nitrogen. It has a low melting point and poor chemical stability, and is easily dissolved when heated, while the magma always intrudes along coal seams. Generally, magma intrudes along faults, and diffuses uphill into the coal seam.

Figure 1 - Regional structure outline of Huaibei coalfield (Modified according to Shi et al., 2007)

Test sites are in the central and south of Huaibei coalfield, which is located in the Xu-Huai sag in the southeast of the north China plate, and is adjacent to the east side Tan-Lu fault zone, clamping in
Feng-Pei and Bengbu apophysis with a nearly east-west trend. The place is at the end of the large region relating to the Subei fault in neighboring area, to the north of the Taihe-Wuhe fault, to the east of Feng-Wo fault and to the west of Guzhen-Changfeng fault, which is covered by Quaternary strata, which was shown in Figure 1 (Wu et al., 2009). Under the influence of Indosinian Movement and Yanshan Movement, the Paleozoic deposition of Huaibei was wrinkled, uplifted, broken and denuded (Yang, 1996), the Permian coal seam formation in the zone had experienced a complex tectonic evolution, and had received a second strong metamorphism by magma action since the Dabie-Sulu mountain building.

![Diagram](image1)

I —Intrusion area; II —Reverse fault; III —Normal fault

Figure 2 - Magma distribution and rock profile of No.10 coal seam in Wolonghu Coal Mine

![Diagram](image2)

I —Mine border; II —Intrusion area; III —Reverse fault; IV —Normal fault

Figure 3 - Magma distribution and rock profile in Haizi Coal Mine

The Huaibei mining area formed a series of drapes and fractures along NNE trend in the Yanshan Period, and extrusion was the main form, meanwhile a large-scale magma intrusion action happened. In this period, the magma, which intruded along the Subei fault up into the hanging wall and footwall, intruded the Linhuan mining area southward and the Suixiao mining area northward (Xu,1986). The Wolonghu Coal Mine is located in the west edge of Xusu arcuate structure, on the east side of the Kouziji fault in Feng county, on the north side of the Zhaozhuang anticline, and on the south side close to the Subei fault. Magma action is very universal from the No.2 coal seam to No.C3, and much intrusion is layered along the coal-rock bedding. The intrusion case of the No.10 coal seam is shown in Figure 2. Haizi Coal Mine is located between the NW trend fault (Subei fault and Guangwu-Guzheng fault) and the NE trend fault (Taihe-Wuhe fault and Guzhen-Changfeng fault), and in the northwest of the Tongting anticline. The magmatic rock is distributed as a sill which intrudes along the No.5 coal seam in the middle and western part of the mine, which is called the extremely thick igneous rock (Wang, et al., 2008; Wang, 2009; Wang, et al., 2010). The length of its strike is 6.5 km and the trend length is about 140 km. The distance in height between the No.10 and 9 coal seams is 84 m on average, and the No.7 coal seam is 115 m above the No.10 coal seam, which in turn, is 55 m above a 120 m extremely thick
igneous rock. The extremely thick igneous rock is distributed in a stable condition in the 1102 mining area above the No.7 coal roof; its thickness is usually more than 120 m. The magma distribution and rock profile is shown in Figure 3.

EXPERIMENTAL METHODS AND SAMPLING

Test parameters

The adsorption capability testing of coal is based on gas adsorption theory, usually the gas adsorption constants \((a, b)\) are used to measure the gas adsorption capacity. Adsorption constant \((a)\) is the extreme gas adsorption quantity, and its value reflects the maximum gas adsorption capacity. The adsorption constants \((a, b)\) of the coal sample are measured by using the HCA (High Capacity Method).

Gas outburst indexes are significant indicators of coal and gas outburst tendency, such as the initial gas releasing rate \(\Delta p\) and coal sturdiness coefficient \(f\). The initial gas releasing rate \(\Delta p\) indicates the speed of gas emission, which is related to gas content, porous structure and pore surface properties, and is measured by velocity of gas diffusion in the WT-1 type measuring instrument. Its critical value is 10mmHg in China. The coal sturdiness coefficient \(f\) reflects the physical and mechanical properties of coal seams. The greater the coefficient, the more likelihood outbursts occurring. In China, the dropping hammer method is adopted to measure the sturdiness coefficient, and its critical value is 0.5.

Coal industrial analysis includes the measurement of moisture, ash content, volatile content and carbon. It is measured by using the 5E-MAG6600 type automatic industry analyser. Coal industrial analysis could master the industrial quality and component of coal seams, which can also reflect the degree of coal seam metamorphism by volatile content.

Distribution of sampling locations

In order to research the metamorphism, adsorption characteristics and variation law of gas outburst indexes of the coal under the action of magma intrusion, continuous samples from the magma intrusion region are needed. The samples of Wolonghu Coal Mine were taken from 102 working face in the No.10 coal seam, where 6 samples were taken from the magma intrusion region to the normal region along 102 intake airflow roadway, which was shown in Figure 4. As magma intruded along the No.5 coal seam in Haizi Coal Mine, the samples were continuously taken from No.7, 8, 9 and 10 coal seams under the extremely thick igneous rock. Two samples were taken respectively from the No.7, 8 and 10 coal seams, and only one sample was taken from the No.9 coal seam, as it was near from the No.8 coal seam and they intercepted. Meanwhile, one sample was taken from the normal region in the No.10 coal seam in order to carrying on the comparative experiment.

![Figure 4 - Magma intrusion border and the distribution of sampling locations in No.10 coal seam of Wolonghu](image)
TESTING RESULTS

Related parameters were determined in the laboratory of National Engineering Research Center of Coal Gas Control, China University of Mining and Technology, and the results were shown in Tables 1 and 2. In order to analyse the relationship of the coal metamorphism characteristics, gas adsorption characteristics and gas outburst indexes at different distance away from the magma intrusion area analysis contrast charts were established. In the charts the distance away from magma intrusion region was the abscissa and the coal volatile content ($V_{idl}$), the adsorption constant ($a$), initial gas releasing rate ($\Delta p$), and the coal sturdiness coefficient ($f$) were $y$-coordinates. The testing results of Wolonghu Coal Mine are shown in Figure 5 and results of Haizi Coal Mine are shown in Figure 6. The samples of Wolonghu were taken along 102 intake airflow roadway, and the elevation did not largely change and its influence wasn’t considered. Two samples were taken from each coal seam under the extremely thick igneous rock in Haizi Coal Mine. Because the action of the igneous rock occupied the dominant position, during processing the date in Table 2, we took the average value of the two samples in the same coal seam for analyzing. At the same time, we tested the sample from the non-igneous rock covered region at the same elevation in the No.10 coal seam for comparison purposes.

Table 1 - Continuous coal samples testing results of No.10 Coal Seam from 102 intake airflow roadway

<table>
<thead>
<tr>
<th>Sample number</th>
<th>Coal seam</th>
<th>Distance away from magma intrusion /m</th>
<th>Elevation /m</th>
<th>Adsorption constant ($a$) /m³/t</th>
<th>Adsorption constant ($b$) /MPa</th>
<th>$\Delta p$ /mmHg</th>
<th>$f$</th>
<th>Mdaf%</th>
<th>Adaf%</th>
<th>Vad%</th>
<th>Fcdaf%</th>
</tr>
</thead>
<tbody>
<tr>
<td>1#</td>
<td>60m in the intrusion region</td>
<td>-497</td>
<td>16.0627</td>
<td>0.1958</td>
<td>1.6</td>
<td>0.76</td>
<td>2.49</td>
<td>42.42</td>
<td>42.00</td>
<td>50.89</td>
<td></td>
</tr>
<tr>
<td>2#</td>
<td>20m in the intrusion region</td>
<td>-503</td>
<td>35.9952</td>
<td>0.9339</td>
<td>3.0</td>
<td>0.80</td>
<td>4.98</td>
<td>34.17</td>
<td>7.11</td>
<td>53.74</td>
<td></td>
</tr>
<tr>
<td>3#</td>
<td>10m from the boundary of magma intrusion</td>
<td>-506</td>
<td>26.2635</td>
<td>0.0993</td>
<td>11.0</td>
<td>0.57</td>
<td>1.89</td>
<td>40.32</td>
<td>8.42</td>
<td>49.37</td>
<td></td>
</tr>
<tr>
<td>4#</td>
<td>30m from the boundary</td>
<td>-510</td>
<td>56.5327</td>
<td>1.6021</td>
<td>30.8</td>
<td>3.00</td>
<td>3.62</td>
<td>36.89</td>
<td>7.82</td>
<td>51.67</td>
<td></td>
</tr>
<tr>
<td>5#</td>
<td>90m from the boundary</td>
<td>-520</td>
<td>59.0229</td>
<td>1.5674</td>
<td>40.0</td>
<td>2.40</td>
<td>4.25</td>
<td>32.63</td>
<td>8.23</td>
<td>54.89</td>
<td></td>
</tr>
<tr>
<td>6#</td>
<td>150m from the boundary</td>
<td>-530</td>
<td>47.0921</td>
<td>1.3671</td>
<td>30.0</td>
<td>2.80</td>
<td>3.86</td>
<td>21.50</td>
<td>7.74</td>
<td>66.90</td>
<td></td>
</tr>
</tbody>
</table>

Table 2 - Testing results of the samples taken from the No.7, 8, 9 and No.10 coal seam in Haizi Coal Mine

<table>
<thead>
<tr>
<th>Sample number</th>
<th>Coal seam</th>
<th>Distance away from magma intrusion /m</th>
<th>Elevation /m</th>
<th>Adsorption constant ($a$) /m³/t</th>
<th>Adsorption constant ($b$) /MPa</th>
<th>$\Delta p$ /mmHg</th>
<th>$f$</th>
<th>Mdaf%</th>
<th>Adaf%</th>
<th>Vad%</th>
<th>Fcdaf%</th>
</tr>
</thead>
<tbody>
<tr>
<td>9#</td>
<td>7</td>
<td>55</td>
<td>-530</td>
<td>40.1800</td>
<td>1.0360</td>
<td>40</td>
<td>0.38</td>
<td>2.68</td>
<td>38.75</td>
<td>5.32</td>
<td>51.25</td>
</tr>
<tr>
<td>10#</td>
<td>7</td>
<td>55</td>
<td>-530</td>
<td>46.8459</td>
<td>1.0313</td>
<td>48</td>
<td>0.235</td>
<td>2.54</td>
<td>19.82</td>
<td>10.63</td>
<td>67.01</td>
</tr>
<tr>
<td>11#</td>
<td>8</td>
<td>79</td>
<td>-554</td>
<td>35.7932</td>
<td>1.3657</td>
<td>32</td>
<td>0.22</td>
<td>2.57</td>
<td>20.46</td>
<td>8.65</td>
<td>68.32</td>
</tr>
<tr>
<td>12#</td>
<td>8</td>
<td>79</td>
<td>-554</td>
<td>33.8680</td>
<td>1.1746</td>
<td>29</td>
<td>0.29</td>
<td>0.96</td>
<td>21.83</td>
<td>9.36</td>
<td>67.85</td>
</tr>
<tr>
<td>13#</td>
<td>9</td>
<td>84</td>
<td>-560</td>
<td>35.0500</td>
<td>1.0100</td>
<td>27</td>
<td>0.41</td>
<td>1.60</td>
<td>21.05</td>
<td>9.45</td>
<td>67.90</td>
</tr>
<tr>
<td>14#</td>
<td>8</td>
<td>84</td>
<td>-560</td>
<td>26.8075</td>
<td>1.1497</td>
<td>19</td>
<td>0.20</td>
<td>1.96</td>
<td>11.36</td>
<td>13.74</td>
<td>72.94</td>
</tr>
<tr>
<td>15#</td>
<td>10</td>
<td>Non-igneous rock covering</td>
<td>-630.0</td>
<td>19.3298</td>
<td>0.8713</td>
<td>9</td>
<td>0.41</td>
<td>0.79</td>
<td>9.07</td>
<td>19.67</td>
<td>70.47</td>
</tr>
</tbody>
</table>

ANALYSIS OF TESTING RESULTS

The affect on coal rank of magma intrusion

When magma intrudes into the coal seam, the coal seam lies in a high temperature and high pressure environment, under the effects of contact and regional thermal metamorphism, the coal molecular composition will change, and the condensation degree of aromatic thickening rings increases rapidly. The side chain of alkyl and oxygen containing functional groups exfoliates and decomposes, the volatile content of coal is reduced, vitrinite reflectance increases, and the metamorphic grade of coal improves. Metamorphic grade of coal is related to the distance with the intrusion region. When magma intrudes along coal seam bedding, contact and regional thermal metamorphism make the coal seam change to be natural coke, natural char and super anthracite coal. Over a distance, it could divide into several
zones according to the metamorphic grade, from near to far away from the intrusion region. These zones were described as natural coke zone - anthracite coal and other contact metamorphism coal zone-normal coal zone (Guo, et al., 2007). Coal quality depends on the size of the igneous rock and the rock layer thickness between the coal seam and the igneous rock, which determines the degree of coal metamorphism and the distribution area. The relationship between the size of igneous intrusion and the volatile content variation is shown in Figure 7 (Zhao, 1978), which showed that for the thicker igneous rock and the smaller thickness of rock layer between the igneous rock and the coal seam, the volatile content got lower, and, the coal metamorphic grade became higher.

Figure 5 - Variation relation among the coal volatile content, the gas adsorption constant and outburst indexes of the samples taken from Wolonghu Coal Mine

Figure 6 - Variation relation among the coal volatile content, the gas adsorption constant and outburst indexes of the samples taken from Haizi Coal Mine

Figure 7 - Variation curves of $V_{daf}$ with rock bed thickness and distance between rock bed and coal seam in certain coal mine
As shown in Figure 5, it was found that at the position of 60 m inside intrusion region, the coal volatile content was only 4.2%, and the coal was anthracite coal. From contact metamorphic zone to normal zone, the volatile content increased, but was less than 10% on the whole, which indicated that the coal was still in the anthracite stage. The metamorphic region was broad, which surpassed the boundary of 150 m in plane.

As shown in Figure 6, it can be seen that the volatile content of the No.7, 8, 9 and 10 coal seams gradually increased with the distance away from the igneous rock, the metamorphic grade of middle coal seam groups were deeper than the No.10 coal seam. The No.10 coal seam was lean coal while the corresponding middle coal seam groups were almost anthracite coal. From the testing results of the No.10 coal seam in Table 2, it was found that the No.10 coal seam with non-igneous rock covering was coking coal with a lower metamorphic grade, which proved that the high-temperature thermal baking effect of extremely thick igneous played a dominant role on coal quantity in the Haizi Coal Mine.

**Affection on coal adsorption characteristics of magma intrusion**

From a large accumulation of adsorption experimental data, it was found that there was no single value relationship between coal adsorption and metamorphic grade. But there was a general trend that the coal adsorption quantity of gas increased with the improving of coal metamorphic grade under the same gas pressure. It can be seen from Figure 5 that the adsorption constant \( k_a \) of super anthracite coal in the contact metamorphic aureole of Wolonghu Coal Mine was small, and the adsorption capacity of coal samples were the biggest in the transitional zone, and with increasing distance away from the intrusion boundary, the adsorption constant \( k_a \) declined on the whole. In Figure 6, the relation of the adsorption constant \( k_a \) and metamorphic grade changed correspondingly, namely the shorter the distance between the coal seam and the igneous rock, the higher the metamorphic grade of coal, and the stronger the adsorption capacity. Meanwhile, it was found that the adsorption capacity of the No.10 coal seam with non-igneous rock covering was far smaller than the coal under the extremely thick igneous rock in Tab.2. The high baking temperature increased the coal metamorphic grade and the number of micro-pores, and gas absorption capacity of the coal seam was enhanced which improved the capacity for gas storage.

**Affection on coal gas outburst indexes of magma intrusion**

Generally, the coal and gas outburst disasters happen in the soft coal seam which is called “tectonic coal” (Yu, 1992). The outburst coal seam has the following features: the mechanical strength is low and changes quickly, the permeability and humidity are low, the initial gas releasing rate is high, and the coal bedding is disturbed and broken by geological tectonic force. According to Figure 5, when magma intruded into the coal seam, the total coal seam would become natural coke, and the coal was broken in the metamorphic aureole, fractures developed and the coal sturdiness coefficient \( f \) was lower than the normal one. According to Figure 6, the coal sturdiness coefficients \( f \) caused by the high temperature baking under the extremely thick igneous rock were all lower than outburst critical value (0.5). The soft coal seams were often broken and presented the typical characteristics of “tectonic coal”. Meanwhile, from Figure 5 and Figure 6, it was found that the initial gas releasing rate \( \Delta p \) obeyed the same variation law with the adsorption constant \( k_a \). Generally, the coal seam with a large adsorption capacity always had a high initial releasing rate. However, the coal sturdiness coefficient \( f \) did not match the initial gas releasing rate \( \Delta p \). Gas outburst accidents in hard coal seams have happened at home and abroad at present, so the combination effect of the gas outburst indexes should be considered to judge the outburst tendency.

Besides, when layered magma intrudes into the coal seam floor, the pyrolysis action is the strongest. The distance away from the igneous rock decides the metamorphic degree. The covering lithology and thickness of overlying strata decides the gas occurrence and outburst index value.

**CONCLUSIONS**

The geological history of magma intrusion in the central and southern parts of Huabei coalfield, were analysed by using Wolonghu Coal Mine and Haizi Coal Mine as testing sites. By laboratory tests and field experimental research, coal reservoirs adsorption characteristics and gas outburst index variations under the function of high temperature pyrolysis and metamorphism caused by layered magma intrusion were analysed. The results showed that under the additive effects of pyrolysis and metamorphism, the
metamorphic degree of a coal seam increased the closer it was to the igneous rock. The adsorption characteristics of the metamorphic coal were positively correlated with gas outburst indexes. When the magma layer eroded the coal seam, along with the distance from the igneous rock, gas outburst indexes increased first, then decreased. When the layered magma intruded into the coal seam roof, coal seam gas was trapped by the igneous rock, and gas outburst indexes were commonly higher, which decreased with the distance away from the igneous rock. When the layered magma intruded into the coal seam floor, the effect of high temperature pyrolysis was the strongest, and the gas outburst indexes were determined by the covered strata above the coal seam.

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REGIONAL GAS DRAINAGE TECHNIQUES AND APPLICATIONS IN CHINESE COAL MINES

Yuanping Cheng, Haifeng Wang and Lei Wang

ABSTRACT: With China's rapid and sustained development of economy, the demand for coal supply has resulted in the extraction of coal in deeper coal mines, particularly in China's economically developed Eastern region where many mines have started mining at depths between 800 and 1500 m. As the depth increases, the geo-stress, pressure and gas content of coal seam also increases, and the geological conditions become more complicated with lower-permeability and softer coals, thus coal and gas outbursts and gas explosions and other hazards become increasingly serious. In order to effectively prevent gas disasters, the National Engineering Research Centre of Coal Gas Control, together with the Huainan and Huaibei Mining Group, have developed a regional system of gas extraction technology and conducted theoretical and experimental research. The results include: a) under multi-seam conditions, a relatively low gas disaster-prone coal seams can be mined as the first key protective layer, which can lead to stress-relief, and increased permeability by 400 to 3 000 times, thus facilitating the desorption and accumulation of gas flow; b) in a single seam, the use of enhanced extraction boreholes (crossing boreholes and in-seam boreholes) combined with hydro-fracturing greatly increases coal seam gas extraction rates up to 50%. Engineering practice has proved that the regional gas drainage technology not only eliminates the regional coal and gas outburst threat, but also enables gassy seam mining with low gas emissions. The research results have been adapted by China's national safety codes of production technology. This paper introduces the technology principles, engineering practices as well as the key parameters of regional gas drainage for engineering applications.

SERIOUSNESS OF COAL AND GAS OUTBURST DISASTERS

With the rapid and sustained development of China's economy, the demand for coal is increasing. China's coal production increased from 0.999 bt in 2001 to 3.03 bt in 2009, a rise of 203.3% over nine years with average annual growth of 22.6%. This has led to the rapid expansion of mining to greater depths, particularly in the economically developed eastern regions, where coal exploitation has a long history, the shallow coal resources are nearly exhausted, and many coal mines have to start mining at depths of 800~1500 m. In China the coal resources within 800~2 000 m of overburden is about 3.25 tt, accounting for 63% of the total reserves. With the increase of mining depth, increased geo-stress, gas pressure and gas content, have lead to coal and gas outburst associated disasters becoming increasingly serious. Extremely serious coal and gas outburst accidents of over 1 000 t of caol have occurred nine times since 2000. Serious gas explosion accidents with more than a hundred fatalities have occurred eight times. Extremely large outburst accidents happened in Luling Coal Mine in 2002, Zhengzhou Daping Coal Mine in 2004 and Huainan Xieyi Coal Mine in 2006, all were caused by high gas outburst emissions during the extension to deep mining. An in-depth study of gas disaster prevention issues associated with deep coal development will be of great significance to China's deep coal resource exploitation.

Huainan and Huaibei mining areas, located in China's rapid economic development eastern regions have a long mining history, and both are gradually entering a deep mining stage. The depth of many coal mining faces is up to 800~1500 m. The maximum gas pressure is up to 6.4 MPa, and the maximum gas content up to 30 m³/t. Most coal seams are of low permeability in the magnitude of 10⁻³ m/d. The hardness of coal, i.e., the f value, is only 0.1 to 0.4. Coal and gas outburst disasters are very serious. Of the Huainan coal mines, 85% are coal and gas outburst prone as are 60% of the Huaibei coal mines. On April 7th, 2002, China's second largest coal and gas outburst accident occurred in the Luling Coal Mine, accompanied by 10 500 t coal and rock and 1.23 Mm³ of gas.
CONCEPT OF REGIONAL GAS EXTRACTION

Classification of gas extraction methods

There is no uniform classification of gas extraction methods in coal mines. The gas extraction methods have been divided into in-seam extraction in working seams, gas extraction from adjacent seams, and gas extraction in goaf and surrounding stata (Yu, 1992). A new classification of coal mine gas extraction methods with three levels is proposed, as shown in Figure 1. The first level is based on the time of coal mining, which includes pre-drainage, drainage during mining and post-mining drainage. The second level is based on the spatial locations of gas in relation to the mining seam, which includes gas extraction from adjacent seams, gas extraction in working face, gas extraction during development and from goaf. The third level refers to more specific methods of gas extraction, such as surface well gas extraction, cross-measure borehole extraction, and buried pipe in goaf gas extraction.

In coal mine production, the adoption of a single method of gas extraction usually could not solve mine gas problems. Instead, a combination of several of these methods has to be applied to achieve the goal of comprehensive gas extraction (Cheng, et al., 2009).

Purpose of regional gas extraction

Regional gas extraction must be used in the dangerous zone of outburst prone coal seams before coal mining. After regional gas extraction, the gas content and gas pressure must satisfy the requirement of Chinese safety code Basic Indexes of Coal Mine Gas Extraction. Regional gas extraction is an outburst prevention program consisting of the setup of gas control projects from safe zones to unsafe zones during the mining process. It can reduce the gas content of unsafe areas, eliminating the regional risk of coal and gas outburst and making the unsafe zones to be safe.

Long-term theoretical research and outburst coal seam mining have proven that protective seam mining incorporating pre-drainage of seam gas is the most effective regional prevention approach to reduce coal and gas outbursts. This method can reduce the gas content of unsafe areas, minimizing the risk of coal outburst, improving the safety and reliability of controlling measures for coal and gas outburst, and increasing the level of safe mining of outburst coal seams. In addition, the regional gas extraction methods could drain a large volume of high purity gas and improve the utilization of clean energy, thus reducing the emissions of greenhouse gas.

Figure 1 - Classification of mine gas drainage method

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Figure 1 - Classification of mine gas drainage method
Scope and standards of regional gas drainage

Protection area of the roadway in coal seams

When cross-measure boreholes or in-seam boreholes are used in coal roadways as outburst prevention measures, they have different protection effects in different coal seams. Considering the effect of gravity on outbursts, boreholes should control the upper side of roadway for at least 20 m, the down side for at least 10 m and the two sides for at least 15 m in inclined coal seams or half-edge coal seams (SACMS, 2009). As Figure 2 shows, the borehole control range is the distance along the coal seam.

![Diagram showing the protection area of the pre-drainage crossing boreholes for outburst prevention](image)

Figure 2 - Diagram show the protection area of the pre-drainage crossing boreholes for outburst prevention

Protection area of working face

When crossing boreholes or in-seam boreholes are used to pre-drain the disturbed (or the whole mining section) coal seam, they should control the whole mining coal block (or the whole mining section). The locations of these boreholes are shown in Figures 3 and 4. In Figure 3, the method of dense crossing borehole for pre-draining gas applies to the coal seam with high coal and gas outburst risk but without a protective layer. In Figure 4, this method is applied in the situation where there are roadways in a coal seam. For thick coal seams, pre-drainage boreholes should have a control extraction range within 20 m in the roof and within 10 m in the floor (SACMS, 2009).

![Diagram showing the protection scope of dense crossing boreholes as regional outburst prevention measures](image)

Figure 3 - The protection scope of dense crossing boreholes as regional outburst prevention measures

![Diagram showing the protection scope of in-seam boreholes as regional outburst prevention measures](image)

Figure 4 - The protection scope of in-seam boreholes as regional outburst prevention measures
Protection domain during cross-cut coal and shaft-sinking

When cross-measure boreholes are used for pre-drainage in the driving of main roadways or in shaft sinking (shaft and drift entry) before intercepting coal seams, the drilling should be completed at least 7 m (in vertical distance) between the drivage cut-through uncovering the working face and the coal seam. As shown in Figure 5, the lowest controlling ranges of boreholes are: 12 m beyond the boundary of the roadways which are intercepting the seam, at least 5 m for the distance between the edge of protection domain and the working level of the roadway. When the boreholes cannot penetrate the entire coal seam, a crossover distance of 15 m should be kept, and a distance of at least 12 m should be left for the pre-drainage of the coal gas beyond the working roadway and 15 m ahead the working face, otherwise, the work of drilling and inspection should be carried out without any delay.

![Figure 5 - The use of cross-measure borehole pre-drainage as regional outburst prevention measures](image)

Codes compliance

When assessing the effect of pressure-relief gas extraction of the protective layer, the outburst prevention measures in the pre-drainage coal gas regional needs to be examined, firstly including whether the layout of boreholes complies with the design requirements. The assessment mainly uses the residual gas pressure test, residual gas content, deformation of coal seam expansion and other indicators and methods. It can also be combined with secondary indicators such as coal seam permeability. The key parameters to test coal seam gas pre-drainage area outburst prevention measures are the residual gas pressure and residual gas content. The critical index for regional outburst prevention measures is shown in Table 1.

<table>
<thead>
<tr>
<th>Gas pressure (MPa)</th>
<th>Gas content (m$^3$/t)</th>
<th>Coal seam expansion deformation (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt; 0.74</td>
<td>&lt; 8</td>
<td>&gt; 3</td>
</tr>
</tbody>
</table>

Implementation of regional outburst prevention measures

The implementation process of regional outburst prevention measures includes four steps, as shown in Figure 6. These are:

- Step one - carrying out outburst forecast of the regional coal seams, and according to the regional forecast, coal seams are divided into outburst risk and non-outburst risk regions.
- Step two - regional outburst prevention measures implemented prior to mining the outburst region to eliminate outburst risks. These measures include the exploitation of protective seam and the pre-drainage of coal seam gas. Wherever conditions permit, protective seam mining technology should be adopted as a priority.
- Step three - the effectiveness of regional outburst prevention measures are assessed. The main indicators are the residual gas pressure and residual gas content, which requires that the maximum residual value must be less than the critical value after prevention measures. If the residual gas content and pressure value are higher than the critical value, then the outburst prevention measures are insufficient and have to be enhanced further until the outburst risk is eliminated.
- Step four - regional validation will be carried out to ensure the safety of mining operations. If the working face is classified as in an outburst risk region, additional outburst prevention measures should be adopted to eliminate its outburst potential.
Regional forecast outburst prevention measures

Regional outburst prevention measures

Disqualified

Regional measures effect test

Qualified

No bursting hazardous zone

Regional validation

BASIC PRINCIPLES OF REGIONAL GAS EXTRACTION

Protective seam mining and pressure-relief gas drainage

As shown in Figure 7, after the protective seam mining, the overlying strata will form three zones, i.e., the caving zone, the fractured zone and the sagging (deformation) zone. The underlying rock will form two zones, i.e., floor heave fractured zone and floor heave deformation zone. The mining condition of the protected seam must not be damaged when mining the lower protective seam, so the protective seam should be in the fractured zone and sagging zone. The coal measure rocks in the fractured zone produce both parallel and vertical bedding fractures. Under the effect of drainage pressure, the pressure-relief gas can flow along these fractures. A range of efficient gas extraction methods can be used, including the roof or floor boreholes high level borehole along the panel, under or overlying gas drainage tunnels and vertical surface boreholes (Cheng, 2003a; 2003b; 2004; Yu, 2004; Liu, 2010; Wang, 2010 and Wang, 2010).

Enhanced gas extraction of a single seam

The enhanced gas extraction of a single coal seam refers to using intensive cross-measure boreholes and in-seam drainage to conduct high-intensity and longer term extraction of the in situ coal seam, through which to eliminate the outburst danger. China firstly carried out the enhanced gas extraction of a single coal seam in the Liangshan coal mine. From the analysis of the drainage results, the outburst prevention mechanism can be explained in the following three aspects (Li, 1981): (1) The shrinkage of coal caused a stress drop in the coal and the reduction of storage strain energy. Stress reduction can be up to 1.88~3.25 MPa, accounting for 23-40% of the total stress. The reduced amount of strain energy is 0.042-0.127 MJ/m³; (2) Through the extraction of coal seam gas, gas content and pressure can be dropped substantially, thus reducing the potential for outburst. Coal seam gas content is reduced from 20 m³/t to 10.1 m³/t, gas pressure reduced from 2.8 MPa to 0.4 MPa, and gas potential reduced from...
0.77 MJ/t to 0.12 MJ/t; (3) The stiffness factor of coal is increased to 0.25-0.3 from 0.1 after extraction. The combined effect of increased coal hardness and reduced gas pressure gradient significantly minimise the likelihood of coal and gas outbursts.

To improve the single coal seam gas extraction effect, the water-powered induction nozzle technology can be used to enhance the coal bed permeability around the boreholes. This method uses the water pressured high power induction control nozzle in the drilling hole to form a number of fan-shaped boreholes. The coal around the boreholes flows to the adjacent holes, swells, and deforms, with pore fracture system expansion, forming a group of macro pores within the fracture network, with much increased permeability. Field results showed that the seam permeability increased 200 times, significantly reducing gas extraction lead time (Zhou, 2008). Hydraulic fracturing, and deep hydraulic cutting vibrating by blasting can also be used, but large-scale applications of these measures requires further study.

**ENGINEERING APPLICATIONS OF REGIONAL GAS DRAINAGE**

**Gas extraction in Huainan coal seams**

In the Huainan mining area, the coal measures system is Carboniferous-Permian with multiple seams divided into A, B and C groups. The B and C groups are the main mining groups, including 10 to 19 layers, and the total thickness is 23-36 m. The typical stratigraphic section is shown in Figure 8. With the increase of mining depth, the gas pressure and content of the main mining coal seams are increasing. With the increase of mining depths, the majority of the current coal mines are extracting outburst prone coal seams. The gas control issue of outburst coal seams has become the primary problem to be addressed in the Huainan mining area.

![Figure 8 - Histogram of Huainan Coal Mining](image)

The Huainan mining area has a typical condition of coal seams that are suitable for protective seam mining. Firstly, the coal seam of non-outburst risk or low outburst risk is selected as the first key protective layer, mining of which eliminates the outburst risk of adjacent coal seams. Then, the coal seam whose outburst risk has been eliminated is selected as a protected seam, and the mining of this seam leads to pressure relief and increased permeability to eliminate the outburst risk. The protective seams can be repeatedly mined until safely mining the last coal seam.
As shown in Figure 8, the thickness of B10 coal seam is 0.1-0.85 m with an average of 0.9 m, and the outburst risk is relatively small compared with other coal seams. The interburden between the upper B11b coal seam is 30 m, with the lower B8 seam at 40 m. Through comparative analysis, the B10 coal seam was selected as the first key protective layer to mine. After mining the B10 seam, the upper caving zone can protect the safe mining of the B11b coal seam, with the lower fractured zone protecting the safe mining of the B8 coal seam. Then the B11b protective seam at the top was mined to protect the mining of C13 coal seam, and B8 protective seam in the lower was mined to protect the mining of C6 coal seam and so on.

Enhanced gas extraction for single outburst seams in Huaibei coal mine

Because all the mid-group seams in the Huaibei coal mining area are outburst prone seams and most of the mines do not have the conditions for protective seam mining, the use of intensified gas extraction technology to eliminate the danger of outburst is required. Cross-measure boreholes combined with in-seam drilling gas extraction technology is the most common method for enhanced gas extraction. Firstly a rock roadway is driven in the floor, then cross-measure boreholes are drilled from it to pre-drain the strip of coal roadway in order to ensure the construction safety of roadways; Secondly in-seam boreholes from seam roadways are drilled to extract the gas of the mining area so that the outburst danger can be eliminated. This method has been used to control the No. 7 coal seam in Qinan coal mine and achieved good performance. The No. 713 working face was selected as a testing face. The working seam has an average thickness of 3.5 m, an average dip of 5°, gas pressure of 2.42 Mpa and gas content of 12.3 m³/t. The drilling layout is shown in Figures 9 and 10. The spacing between crossing boreholes is 5 m, and 7 holes along the dip are arranged. The control width of coal strip is 35 m. The space between in-seam boreholes is 2-3 m.

![Figure 9 - Floor roadway network crossing holes layout diagram](image)

![Figure 10 - In-seam borehole layout diagram](image)

In this testing face, a total of 1230 cross-measure boreholes are drilled with a total length of 58,837.1 m, including 14,501.4 m of coal holes and 44,335.7 m of rock holes. The average gas extraction rate from cross-measure boreholes is 2.6 m³/min. A total of 1,700,500 m³ gas was extracted and the gas extraction reached an average of 63.4%. Average residual gas content was reduced to 4.5 m³/t, showing that the coal strip has eliminated the outburst danger.

CONCLUSIONS

Measures of regional gas extraction must be taken to eliminate the outburst danger before coal mining due to the seriousness of outburst danger in China’s coal mines. Through the engineering practice in Huainan and Huaibei coal mines, regional gas extraction measures are shown to effectively reduce the gas pressure and gas content of the coal seams, and completely eliminate outburst danger, to achieve the mining safety of highly gassy seams with reduced gas condition.
Although the regional gas extraction measures have achieved remarkable results, there are still many technical problems need to be studied, such as the stress field caused by protective seam mining, fracture field development and the non-steady state desorption flow of pressure-relief gas, and enhancement of low permeability coal seams.

ACKNOWLEDGEMENTS

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REFERENCES

PRESSURE RELIEF GAS EXTRACTION BASED ON STRATA MOVEMENT OF MINED UPPER PROTECTIVE SEAM

Haifeng Wang, Yuanping Cheng and Qingwei Zhai

ABSTRACT: Consensus on protective seam mining technology to solve the problems of coal and gas outburst has been reached. For upper protective seam mining, the effect of pressure relief and permeability in the lower protected layers is determined by the distance between the seams. If the distance is small, the effect of pressure relief and permeability on the lower protected seams is better, but there will be more relief gas flows to the working faces of the protective seam, and gas control of the working faces will be more difficult.

INTRODUCTION

Most high gas coal seams in China have low permeability. The Huainan mining area for example, has a permeability coefficient of 2.5×10^{-3}-2×10^{-2} m/d (0.001~0.008 m^{2}/(MPa·d)) (Zhao and Liao, 2007) and the gas extraction of the original coal seam is very poor. Protective seam mining is a kind of pressure relief gas extraction technology. Through protective seam mining, the pressure relief and permeability increases in the adjacent coal seams, and thus gas extraction is improved (Cheng, 2009; 2007; 2003). Protective seam mining is divided into upper and lower protective seam mining. Mining the upper protective seam will not damage the mining conditions of the protected seams, so mining the upper protective seam should be preferred (SACMS, 2007; 2009). The pressure relief and permeability increase in the protected seams is determined by the distance between the seams (seam spacing). The smaller the distance, the better. Close range upper protective seam mining is ideal for regional gas extraction technology. However, there are some problems in application. The most significant problem is that if gas cannot be extracted quickly from the protected seam it will flow into the working faces of the protective seam, and cause a great security risk.

APPLICATION BACKGROUND

Huainan Xinzhuangzi Mine coal seam C13 is a seam in which many coal and gas outburst dynamic phenomena have happened. There is a thin and unstable coal seam C14 on top of the seam C13, and the coal seam histogram is shown in Figure 1. The seam C14 is planned to be adopted to protect the seam C13 and eliminate its outbursts. A test working face in the upper protective layer of C14 has a coal thickness of 0.1 to 1.2 m, average 0.6 m, and mining height 1.5 m. There is no outbursts potential for seam C14, and the gas content is 6 m^{3}/t. The average thickness of coal seam C13 the underlying protected seam is 6.2 m, the average gas pressure is 4.6 MPa, and the average gas content is 14.9 m^{3}/t. In this geological block, the distance between seams C14 and C13 is the 14 ~ 20 m, with an average of 18 m.

MOVEMENT AND FRACTURE EVOLUTION OF COAL-ROCK MASS IN FLOOR

After mining of an upper protective seam the floor strata will move up under integrated stresses, and floor heave will appear in the goaf (Qian and Liu, 1984). According to the simulation and field tests and previous research, the floor strata affected by mining can be classified as floor heave fracture zone and floor heave deformation zone, as shown in Figure 2. The lower limit of the floor heave fracture zone is 15~25 m below the goaf, the fracture in the zone are mainly bedding fractures parallel to the seam and penetrating fractures vertical and skewed to the seam. The penetrating fractures connect the coal seams within the fracture zone and the goaf, the relief gas can flow into the protective seam goaf along the fracture. If the protected seam is in the zone, it is called the close range upper protective seam mining. The lower limit of the floor heave deformation zone is 50~60 m below the goaf, the fracture in the zone are mainly bedding fracture and penetrating fracture are lacking. The protected seam in the zone

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will expand and deform, the permeability of the coal seam will increase, which will create favorable conditions for gas extraction. The number of fractures decreases as seam spacing increases.

<table>
<thead>
<tr>
<th>Thickness /m</th>
<th>Seam</th>
<th>Rock property</th>
</tr>
</thead>
<tbody>
<tr>
<td>Average</td>
<td></td>
<td></td>
</tr>
<tr>
<td>37–41</td>
<td></td>
<td>Upper part are mudstone, sand-mudstone, refined banding sandstone; central part are aluminum soil rock, sand-mudstone, sandstone; lower part are mudstone coarse sandstone</td>
</tr>
<tr>
<td>39.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>19–20</td>
<td></td>
<td>Gray sand-mudstone</td>
</tr>
<tr>
<td>19.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0–1</td>
<td></td>
<td>Seam C15</td>
</tr>
<tr>
<td>0.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>2–4</td>
<td></td>
<td>Sand-mudstone</td>
</tr>
<tr>
<td>2</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0–2</td>
<td></td>
<td>Seam C14</td>
</tr>
<tr>
<td>1</td>
<td></td>
<td></td>
</tr>
<tr>
<td>14–20</td>
<td></td>
<td>Upper part are sand-mudstone, mudstone; central part are medium-coarse sandstone; lower part are sand-mudstone</td>
</tr>
<tr>
<td>18</td>
<td></td>
<td></td>
</tr>
<tr>
<td>4.38–7.4</td>
<td></td>
<td>Seam C13</td>
</tr>
<tr>
<td>6.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>1.5–3.5</td>
<td></td>
<td>Gray mudstone, local part is sand-mudstone</td>
</tr>
<tr>
<td>2.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0–0.6</td>
<td></td>
<td>Seam C12</td>
</tr>
<tr>
<td>0.3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>24–35</td>
<td></td>
<td>Layered to thick layered mudstone and coarse sandstone</td>
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<tr>
<td>29.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>3–21</td>
<td></td>
<td>Gray-white medium-coarse sandstone</td>
</tr>
<tr>
<td>10</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.4–19.7</td>
<td></td>
<td>Gray sand-mudstone</td>
</tr>
<tr>
<td>6.3</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**Figure 1 - Coal seam histogram of Xinzhuangzi Mine**

Coal seams will expand and deform when moving up, which is the macroscopic performance of fracture development. The greater expansion deformation, the more significant will be the permeability increase, and the stronger the gas desorption effect. The typical vertical deformation curve of the close range lower protected seam is shown in Figure 3, which was obtained from an adjacent Xieyi Mine working coal seam C13. It can be seen from the figure, in the upper protective seam mining period, the compression deformation of the protected seam C13 occurs first and then expansion deformation. The maximum relative amount of compression deformation is 1.14‰, and the maximum relative expansion deformation is 4.00‰.

**Figure 2 - Fracture zones of floor stratum**

**Figure 3 - Typical vertical deformation curve of the close range lower protected seam**
RELIEF GAS MIGRATION LAWS AND EXTRACTION EFFECTS OF LOWER PROTECTED SEAMS

Gas release and migration trends of lower protected layers

In the process of close range upper protective seam mining, favourable pressure relief and permeability increased of the lower protected seams can be obtained (Yu, 1992). Under the common effect of the negative pressure of ventilation in the working faces of the protective seams and high-pressure gas in the protected seam, much relief gas from the protected seams flows into the faces of the protective seams along the penetrating fractures. When the spacing is less than 10~15 m, the desorption gas emission rate of the lower protected seams can reach over 80% (SACMS, 2006). If a large amount of gas of the lower protected seams flows into the working faces of the protective seams, it will cause a great security risk, or even cause mining to be stopped. So during close range exploitation of the upper protective seams, effective face ventilation ways and related extraction measures must be selected. Using negative pressure of ventilation and extraction could change the gas migration direction, and ensure safe mining of the protective seams.

Positive effects of Y-ventilation on the safe mining of the close range upper protective seams

The Y-ventilation is a “two-intake and one-return” ventilation. The leaking air of the goaf does not flow through the upper corner, gas accumulation in the upper corner can be eliminated completely, and then the gas overflow problem can be solved. In addition, there are two inlet air tunnels for the Y-ventilation, and the total air quantity is more than that of U-ventilation. With the same gas concentration, the gas emission capacity is higher than that of U-ventilation. In summary, the Y-ventilation is more suitable for the working face of the close range upper protective seams than is U-ventilation.

Effect of gas extraction on the relief gas migration direction

To ensure safety of the working faces of the protective seams and reduce the gas contents of the protected seams, boreholes must be drilled to form a new free space for rapid desorption and flow of gas. Through sunction of the boreholes, the flow direction of relief gas of the protected seams can be changed. Under normal circumstances, the technique of grid-type penetrating boreholes from the floor roadway is used to extract the relief gas in the lower protected seam, as shown in Figure 4. First the floor roadway needs to be constructed under the protected seam along the strike, and then a set of equally spaced boreholes are made at regular intervals in the roadway. Each set shows a fan-shaped arrangement of boreholes that should cross the protected seams, with hole diameters not less than 90 mm.

Figure 4 - Layout inclination diagram of the grid-type penetrating borehole in floor roadway

The negative pressure of the penetrating borehole causes the relief gas of the protected seams to flow into the boreholes along the bedding fractures; however, it can flow into the goaf of the protective seams along the penetrating fracture. The gas flow direction is determined by the frictional drag formed by gas flow in the cracks and the negative pressure of the borehole extraction. The permissible negative pressure range of the free space (protective seam goaf and the boreholes) changes little, so the flow direction is mainly determined by the frictional drag (Wang, et al., 2010).

To control the relief gas of the protected seams flowing to the boreholes, valid borehole spacing must be selected to reduce the frictional drag in the fracture and improve the sealing quality of the boreholes, thus a certain negative pressure extraction capacity within the holes can be gained. The gas flow with the best extraction effect is shown in Figure 5. Most of the protected seam gas is extracted from the boreholes. According to the theoretical analysis and field test results, in close range upper protective
In the mining process of the protective seam working of face 62114, besides the grid-type penetrating boreholes, a high level drainage roadway was used for gas extraction.

**EFFECT ANALYSIS**

**Ventilation and gas extraction of the protective seam working faces**

During mining of the protective seam working face 6214, proper gas extraction measures were taken to ensure working face safety. The amount of air distribution to the working face was about 1500 m³/min, gas concentration below 0.6%, and the gas emission by ventilation 4-9 m³/min.

In the protective layer mining period, a total gas extraction was 21 520 000 m³. Three-quarters of the total gas extraction occurred by penetrating boreholes in floor roadway, by high level drainage roadway and by ventilation. The remainder one quarter was removed by other means. According to the analysis of the results, the relief gas extraction rate of the protected seams in protection range should reach 77.5%, the gas pressure of the protected seams reduces from 4.6 MPa to 0.34 MPa, and the gas content from 14.9 m³/t to 3.3 m³/t. This level of reduction in the gas should completely eliminate outbursts of the protective seams.

**Outburst elimination verification in drivage face of lower protected layers**

In protection range the gate road excavation work was carried out safely. From the field records, the maximum weight of drilling cuttings “S” was 3.2 kg/m, the maximum cuttings desorption index \( K_1 \) was 0.28 ml/g·min⁰.². These values were far less than their outburst critical values. The gas concentrations of the drivage face were all less than 0.3%. The above three indicators can fully show that the gas content and pressure of the coal seam C13 had a significant decline through the mining of the coal seam C14 and corresponding gas extraction, which completely eliminated outbursts of the coal seam.

**CONCLUSIONS**

1) The lower limit of the floor heave fracture zone was 15~25 m below the goaf. The penetrating fractures connected the coal seams within the fracture zone and the goaf, and the gas of the protected layer in the zone had the tendency to flow into the protective seams.

2) In the process of close range upper protective seam mining. Gas control measures with a combination of extraction and emission is necessary. The borehole spacing for relief gas extraction of the lower protected seams should be less than or equal to the seam spacing. If necessary, the roof high level drainage roadway extraction can be used.
3) Through the close range upper protective seam mining and protected seam extraction, the relief gas extraction and emission rate of the protected seams in protection range reaches 77.5%, the gas pressure reduces from 4.6 MPa to 0.34 MPa, and the gas content from 14.9 m$^3$/t to 3.3 m$^3$/t. This reduces the coal seam gas content, and completely eliminates outbursts of the protected seam C13.

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REFERENCES

EVALUATION OF OUTBURST POTENTIAL AT SIHE COAL MINE, CHINA

Mingju Liu¹,², Hani Mitri², Jianping Wei¹, Wang Xiao¹ and Zhihui Wen¹

ABSTRACT: The coal and gas outburst accident on May 20, 2007 at Sihe Coal Mine is unique in that there is no or very little soft coal in the coal seam and that the coefficient of coal strength is very high. It was also indicated in the outburst that there is a very high gas content of 20-30 m³/t in the coal seam nearby the outburst site. Based on field test and observations, together with laboratory experiments, detailed analysis was conducted on controlling factors of coal and gas outburst potential at Sihe Coal Mine. Stress data available from Sihe Coal Mine, northern China and other parts of world, were also taken into account to facilitate the analysis. The results show that the stress level is not abnormal compared to other coal fields in China prone to outbursts, and that the gas contained in the coal seam is the main factor that triggered the outburst. The results show that outbursts are limited to only a small part of the whole coal seam and that the East part of coal field is not prone to outbursts.

INTRODUCTION

Sihe Coal Mine, owned by Jincheng Anthracite Mining Group, is located in Jincheng City. It currently has the capability to produce annually 10.8 Mt of coal. There are three coal seams and all were formed in Permian. One of them, namely the number 3 coal seam, is the only one believed to be outburst prone, and its thickness varies in the range of 4.45 to 8.75 m, with an average of 6.31 m. The coal seam strikes NNE and dips to the NWW at an angle of 2 to 10°. Geological structures are simple, and the coal seam has not been tectonically disturbed. Underground observations indicate that the seam keeps its original sedimentary structure. On average, the coal seam has an overburden thickness of 300 to 600 m.

Geologically, Sihe Coal Mine is located at the southeast end of the composite syncline of Qinshui Basin, North China. Geologic structure in the area consists of faults, joints, and igneous dykes. The most significant geologic structure is the north–east oriented Panhe Syncline that divides the Sihe coal field into Main West and Main East. The mine was connected underground before the outburst on May 20, 2007, and it has been isolated as two independent parts by permanent shields approximately along the Syncline. Generally speaking, geological structures in Main West are more complicated than that in Main East, which contributes to the potential of outburst occurrence in Main West area.

OUTBURST AT SIHE COAL MINE

On May 20, 2007, an outburst occurred at Sihe Mine, Jinchang Mining Group, and four coal mine workers were buried in the ejected coal and killed. Approximately 370 t of coal were blowout with the release of about 87 000 m³ of gas in the outburst.

Mining and development of entries

Because of the outbursts in neighbour privately owned coal mines, the mine is prudent in dealing with outburst potential. Typically, “four-in-one” measures are employed to prevent and control outbursts in entry development whenever there is obvious evidence of outburst potential, including gas surge emission from boreholes, dramatic change of coal seam, high level of gas content in ventilation air etc. Coal Mine Safety Regulations provide that the “four-in-one” measure be taken with high alert to prevent outburst occurrence, which consists of four steps. The first step is the assessment of outburst potential by \( K_1 \) index at cross cut No 6, which is an indicator of gas emission velocity from coal drilling yields. The critical value is 0.5, and if it is higher than 0.5, the heading face of the entry is believed to be outburst prone and then over 20 boreholes of 89 mm in diameter are drilled into the heading face to destress and release gas in coal seam. When all boreholes are completed, \( K_1 \) index is again tested to make sure it is

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less than 0.5 and the danger of outburst has been eliminated. Then normal operations on the heading face are executed with rescue chamber and rescue ventilation system set up nearby.

Outburst on May 20

The #6 cross cut was driven seven days per week during three 8-hour shifts. On May 9, 2007, the morning shift crew tested both $K_1$ and $S$ (borehole drilling yields), and the maximum $K_1$ and $S$ were 1.77 ml/(g·min$^{1/2}$) and 2.5 kg/m respectively, which indicated an impending outburst potential. Then the development operations were stopped and measures against outbursts were executed to eliminate the potential. According to regulations of Sihe, tests on $K_1$ and $S$ were carried out on May 18, and they were 0.49 ml/(g·min$^{1/2}$) and 3.2 kg/m respectively, surely suitable for resuming development operations. Before mid shift on May 20, 6m were driven, with 2m left allowed to be finished by mid shift. At about 13:22, May 20, an outburst occurred, with the ejection of 370 t of coal and the release of 87 000 m$^3$ of gas, mainly methane. Four coal mine workers were killed in the outburst and several others were injured.

Figure 1 - Normal coal from Sihe outburst site (left) and tectonically disturbed coal from outburst cavity at Menin Mine, Henan, China (right)

The site of the outburst was at the heading face of #6 cross cut, which was planned to connect two main airways in Main West. The outburst is characterized by its no connection with tectonically disturbed or soft coal and by its release of huge volume of gas. Worldwide experiences (Shepherd, et al., 1981; Frodsham and Gayer, 1999; Liu, et al., 2008) have shown that almost all outbursts are accompanied by tectonically disturbed coal or soft coal. Usually, the soft coal is formed by crushing coal seams into coal fines by thrusting, folding or sliding in coal seams millions of years ago.

After the accident, the authors made an underground visit to the outburst site and made some observations. Onsite observation (Liu, et al., 2008) showed that coal seam close to the outburst site is normal without soft coal, and sedimentary bands can be clearly seen as is indicated by the left side photo shown in Figure 1.

UNDERGROUND AND LABORATORY TESTS

After the accident, the authors conducted underground onsite observations on tectonically disturbed coal, underground tests on indexes for assessment of outburst potential and laboratory test, with aims to analyse the causes of the outburst and to assess outburst potential in the Main West and Main East.

Distribution of tectonically disturbed coal

Tectonically disturbed coal or soft coal is coal whose original sedimentary structure is totally damaged by geological movements, with very low strength and high gas emission rate. Traditionally, it is a primary condition for a coal seam to be outburst prone when the overburden is less than 600 m. Therefore, it is of critical value to have a clear picture of its distribution if an accurate assessment of outburst potential is to be made. Two methods are employed to obtain the distribution of soft coal: one is direct underground observations; another is to calculate by borehole logging data.
The results obtained from both methods show that soft coal layers are not well developed and their total thickness is generally under 0.5 m, the majority of which, approximately 76%, are under 0.3 m. According to our research at Huainan coal field, Anhui Province and Hebi coal field, Henan Province, at least 0.90 m of soft is necessary for a coal seam to be prone to outburst occurrence. It is worth noting that there is no soft coal in the vicinity of the outburst on May 20.

Laboratory test on gas desorption from coal

An experimental system (see Figure 2) was set up to test gas emission rules and outburst potential assessment indexes $K_1$ and $\Delta h_2$, of which $K_1$ was used to assess outburst danger before the May 20 accident. The system has the capacity to test sample of 2.5 kg of coal, allowing more accurate test results than that carried out underground with only 10 g of coal.

Tests were conducted on coal samples from the ejected coal of the outburst. Coal samples from other coal fields such as Jiaozuo and Yima were also tested for comparison. Test method was described by Wen in details (Wen, 2008). The results show that there exists a linear relationship between gas pressure and index $K_1$. Although coal samples are different, their trends for the relations between index $K_1$ and gas pressure are approximately the same; and the only differences are their intersections with $K_1$ axis.

For hard coals with less volatile gas contents and higher strengths such as Jincheng and Zhaogu, their respective values of index $K_1$ are much lower than those from coals with higher volatile contents and less strengths such as Yima and Jiulishan. It can be estimated that for coal from Jincheng, when $K_1$ is higher than 0.5 ml/(g·min$^{1/2}$) the gas pressure can be as high as 1.5 MPa, and if $K_1$ reaches 1.77 ml/(g·min$^{1/2}$), as is the case of the outburst accident, the gas pressure will be as high as 6.9 MPa.

In addition, the same tests were conducted for two coal samples from the Main East to further analyze gas desorption law at Sihe coal mine. It is clearly seen from both Figure 3 and Figure 4 that whatever equilibrium pressure is, gas initial desorption velocity of the soft coal is faster than that of the hard coal with little difference in gas desorption quantity.
Indexes of coal properties

Because the majority of the coal seam is hard coal, two important indexes of coal properties are tested for hard coal samples at Sihe Coal Mine that keeps its original sedimentary structure perfectly. These indexes are strength coefficient \( f \) and Gas Emission Index \( \Delta P \). The test results show that values of strength coefficient \( f \) are 1.5, which is much higher than the critical value shown in Table 3 in section 4 of this paper; and the Gas Emission Index \( \Delta P \) ranges from 20 to 44.

ANALYSIS AND DISCUSSION

Since geological structures in Sihe are generally simple when compared to other coal mining fields, the three most important factors, namely stress level, gas content (pressure) and coal properties, will be considered to analyse and discuss the outburst.

Overburden and stress levels

The maximum overburden in the cross cut is 458 m, which is not uncommon in outburst coal mines in most parts of China. Many experts have the opinion that the outburst may have been caused by very high horizontal tress levels developed in ancient geological periods. Stress levels in Main West and Main East have been measured jointly by Coal Research Institute (2004), Beijing and Sihe Coal mine and are listed in Table 1.

<table>
<thead>
<tr>
<th>Area</th>
<th>Overburden (m)</th>
<th>( \sigma_v ) (MPa)</th>
<th>( \sigma_{H_{\text{max}}} ) (MPa)</th>
<th>Direction of ( \sigma_{H_{\text{max}}} )</th>
<th>( \sigma_{\text{min}} ) (MPa)</th>
<th>( \sigma_{H_{\text{max}}} / \sigma_v )</th>
</tr>
</thead>
<tbody>
<tr>
<td>Main East</td>
<td>303.0</td>
<td>8.03</td>
<td>16.585</td>
<td>N83.4ºW</td>
<td>8.53</td>
<td>2.07</td>
</tr>
<tr>
<td></td>
<td>384.0</td>
<td>10.17</td>
<td>18.264</td>
<td>N69.2ºW</td>
<td>8.67</td>
<td>1.80</td>
</tr>
<tr>
<td></td>
<td>376.0</td>
<td>9.96</td>
<td>18.928</td>
<td>N71.2ºW</td>
<td>9.76</td>
<td>1.90</td>
</tr>
<tr>
<td></td>
<td>439.7</td>
<td>11.65</td>
<td>13.00</td>
<td>N17.4ºW</td>
<td>6.74</td>
<td>1.12</td>
</tr>
<tr>
<td></td>
<td>399.2</td>
<td>10.58</td>
<td>13.20</td>
<td>N63.8ºW</td>
<td>6.98</td>
<td>1.25</td>
</tr>
<tr>
<td></td>
<td>410.5</td>
<td>10.88</td>
<td>13.81</td>
<td>N39.0ºW</td>
<td>7.92</td>
<td>1.27</td>
</tr>
<tr>
<td></td>
<td>459.6</td>
<td>12.18</td>
<td>10.89</td>
<td>N62.7ºE</td>
<td>5.61</td>
<td>0.89</td>
</tr>
<tr>
<td></td>
<td>414.6</td>
<td>10.99</td>
<td>10.80</td>
<td>N65.7ºE</td>
<td>5.92</td>
<td>0.98</td>
</tr>
<tr>
<td></td>
<td>409.4</td>
<td>10.85</td>
<td>12.43</td>
<td>N79.2ºE</td>
<td>6.62</td>
<td>1.15</td>
</tr>
<tr>
<td></td>
<td>410.2</td>
<td>10.87</td>
<td>13.67</td>
<td>N63.5ºW</td>
<td>7.05</td>
<td>1.26</td>
</tr>
<tr>
<td></td>
<td>310.0</td>
<td>8.22</td>
<td>11.34</td>
<td>N49.7ºE</td>
<td>6.40</td>
<td>1.38</td>
</tr>
<tr>
<td></td>
<td>292.4</td>
<td>7.75</td>
<td>11.74</td>
<td>N54.2ºE</td>
<td>6.70</td>
<td>1.51</td>
</tr>
<tr>
<td></td>
<td>285.5</td>
<td>7.54</td>
<td>10.90</td>
<td>N43.5ºE</td>
<td>5.71</td>
<td>1.45</td>
</tr>
</tbody>
</table>

The data in Table 1 show clearly that stress levels in Main West and Main East are pretty normal when compared with worldwide stress levels (Brown and Hoek, 1978) and that within China (Cai, et al., 2002;
Zhao, et al., 2007). According to Cai et al. (2002), they are in the medium category and maybe lower than those in some coal fields such as Shenhuan Mining Group, east Henan Province, China, where the stress level is higher than 25.0 MPa with same overburden and there is no outburst occurrence.

The data in the table indicate that stress level in Main West is higher than that in Main East, which may not explain why the outburst occurred in Main West. However, this situation and the comparison thus made show obviously that stress level is not the dominating factor that caused the outburst on May 20.

Gas pressure and gas content

It is well established that coal seam gas pressure or gas content is one of the most important factors controlling the occurrence of outbursts. As is shown in Table 3, it is provided by Detailed Regulations of Outburst Coal Mine that coal seam with gas pressure higher than 0.74 MPa should be considered as outburst prone if the coal seam is tectonically disturbed. Thresholds of gas contents for outbursts to occur depend mainly on the geological conditions and the nature of coal seams. These gas content thresholds for the coal seams in Huainan, Anhui Province, Hebi and Zhengzhou, Henan Province are 9.0 m³/t, 12.0 m³/t and 8.0 m³/t, respectively. And in Australia, the threshold is 9.0 m³/t when the gas involved is methane (Liu, et al., 2008).

Data from coal field prospecting period show that coal seam pressures range from 0.8 to 2.12 MPa and gas contents from 10.70 to 26.80 m³/t. Borehole gas content closest to the outburst site is 16.60 m³/t, which is probably 20-50% less than its true value according to China’s empirical data obtained from prospecting borehole drilling. Although immediate onsite data of gas contents or gas pressures are not available to evaluate exactly how the outburst was influenced by gas storage status, however, the gas pressure may be as high as 6.90 MPa, as was indicated by our previous laboratory test and estimation. Chongqing Coal Research Institute (2005) has tested some adsorption parameters of gas on coal samples from Main West, and the results show an average Langmuir parameter a=40 m³/t, b=1.6 MPa⁻¹. Based on the previously estimated pressure and adsorption parameters of gas on coal at Sihe, it is estimated the gas contents are higher than 36.68 m³/t.

<table>
<thead>
<tr>
<th>Date</th>
<th>Coal Mine</th>
<th>Coal Ejected (t)</th>
<th>Gas Released (m³)</th>
<th>Gas Released Per Ton of Coal (m³/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>04-19</td>
<td>Dashucun Mine, Hebei</td>
<td>1270</td>
<td>93 000</td>
<td>73.20</td>
</tr>
<tr>
<td>04-18</td>
<td>Taer Mine, Hebei</td>
<td>475</td>
<td>65 000</td>
<td>136.8</td>
</tr>
<tr>
<td>05-20</td>
<td>Sihe, Shanxi</td>
<td>370</td>
<td>87 000</td>
<td>235.13</td>
</tr>
<tr>
<td>06-07</td>
<td>Jinjia Mine, Guizhou</td>
<td>204</td>
<td>8 000</td>
<td>39.80</td>
</tr>
<tr>
<td>11-08</td>
<td>Qunli Mine</td>
<td>402</td>
<td>37 283</td>
<td>92.74</td>
</tr>
<tr>
<td>11-12</td>
<td>No.10, Henan</td>
<td>2000</td>
<td>40 000</td>
<td>20.00</td>
</tr>
</tbody>
</table>

The unusually high gas content and gas pressure can be established by the large volume of gas released by the May 20 outburst. Table 2 is a list of six most important outbursts that occurred in 2007 in China, which indicates that gas release from the Sihe outburst is the highest, with 235.13 m³ of gas per ton of coal. From Table 2 and other outburst experiences from China and other countries, for the majority of outbursts, the average gas release per ton of coal is in the range of 30 to 150 m³. The high volume of gas release from the outburst on May 20 indicates high gas content and high gas pressure in coal seam. Nevertheless, the outburst occurred after gas drainage of almost 10 days. If it had occurred without the gas drainage, much more gas release could be expected because of its unusually high gas content and gas pressure.

Coal properties

Various works (Frodsham and Gayer, 1999; Liu et al., 2008; Wen, 2008) found that the nature of coal seams plays important role in outburst occurrence. In China, coal is classified into 4 categories (Liu et al., 2008), of which category I and II are considered as not outburst prone, category III and IV are tectonically disturbed soft coal and are considered outburst prone. Based on approximately 1000 laboratory tests and observations on coal samples from more than 10 coal fields in Guizhou, Anhui and
Henan, for about 95% of disturbed coal samples, their values of \( f \) and \( \Delta P \) are above their corresponding thresholds, that is \( f \) less than 0.5 and \( \Delta P \) more than 10 (Table 3).

**Table 3 - Thresholds of outburst potential parameters in coal seams**

<table>
<thead>
<tr>
<th>Coal Strength Coefficient of ( f )</th>
<th>Gas Emission Index ( \Delta P )</th>
<th>Gas Pressure (MPa)</th>
<th>Coal Structure</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt;0.5</td>
<td>&gt;10</td>
<td>&gt;0.74</td>
<td>Tectonically disturbed</td>
</tr>
</tbody>
</table>

From the laboratory test results, coal samples from site of the outburst on May 20 are not outburst prone because they are not tectonically disturbed and their strength coefficient are much higher than 0.5.

**Cause of outburst on May 20**

As is shown in Table 3, it is provided by Detailed Regulations of Outburst Coal Mine that coal seams with gas pressure higher than 0.74 MPa should be considered as outburst prone if the coal seam is tectonically disturbed. For coal seam in Main West at Sihe Coal Mine, its stress level is normal, its overburden is 458 m, and it was not tectonically disturbed. Compared to other normal coal seams, which are not outburst potential at all, the only and obvious difference is that the coal seam at Sihe Coal Mine has unusually high gas pressure and high gas content.

It is, therefore, concluded that the main cause of the outburst at Sihe Coal Mine is the large volume of gas stored in coal seam. This conclusion is the basis for the future assessment of outburst potential at Sihe Coal Mine.

**ASSESSMENT OF OUTBURST POTENTIAL**

Based on the analysis and discussion made previously on factors that may affect outburst potential at Sihe Coal Mine, it is surely possible to assess outburst potential by utilizing data of soft coal distribution in coal seam and data of gas contents.

**Standards for outburst potential assessment**

Compared to coal fields that are liable to outbursts, the #3 coal seam is unique in that it demonstrates very thin layers of tectonically disturbed coal or none at all and that it has unusually high gas pressures or gas contents. Currently, we are not sure exactly what is the lowest gas pressure or content for the coal seam to be safe from outburst occurrence. It is, however, certain that when gas content is lower than 12 m\(^3\)/t, the coal seam will not be outburst prone at all.

**Table 4 - Gas contents in Main East**

<table>
<thead>
<tr>
<th>Measurement Sites</th>
<th>Gas Content ( (m^3/t) )</th>
<th>Components(%)</th>
<th>( N_2 )</th>
<th>( CH_4 )</th>
</tr>
</thead>
<tbody>
<tr>
<td>Outby 38 m, 3# Cross Cut, 23053 Entry</td>
<td>9.25</td>
<td>4.95</td>
<td>95.05</td>
<td></td>
</tr>
<tr>
<td>Left, Outby 6 m, 5# Cross Cut, Belt Entry</td>
<td>6.35</td>
<td>7.88</td>
<td>92.12</td>
<td></td>
</tr>
<tr>
<td>Right, Outby 6 m, 5# Cross Cut, Belt Entry</td>
<td>4.22</td>
<td>27.40</td>
<td>72.60</td>
<td></td>
</tr>
<tr>
<td>Entry 43013</td>
<td>9.74</td>
<td>7.64</td>
<td>92.36</td>
<td></td>
</tr>
<tr>
<td>Entry 41014</td>
<td>8.08</td>
<td>0.97</td>
<td>99.03</td>
<td></td>
</tr>
<tr>
<td>Inby 42 m, #1 Cross Cut, Entry 41013</td>
<td>10.97</td>
<td>7.31</td>
<td>92.69</td>
<td></td>
</tr>
<tr>
<td>Outby 27 m, #3 Cross Cut, Entry 43013</td>
<td>7.82</td>
<td>7.64</td>
<td>92.36</td>
<td></td>
</tr>
<tr>
<td>Outby 16 m, #5 Cross Cut, Entry 43014</td>
<td>3.27</td>
<td>48.9</td>
<td>51.10</td>
<td></td>
</tr>
<tr>
<td>Outby 45 m, #5 Cross Cut, Entry 43012</td>
<td>6.38</td>
<td>10.40</td>
<td>89.60</td>
<td></td>
</tr>
<tr>
<td>Outby 30 m, 15 Cross Cut, Entry 43013</td>
<td>7.44</td>
<td>8.89</td>
<td>91.11</td>
<td></td>
</tr>
</tbody>
</table>

The safe gas content level of 12 m\(^3\)/t can be established by Table 4. Table 4 is valid for low gas contents and overburden less than 600 m. For coal from Sihe, If the gas pressure is 0.74 MPa, the corresponding gas content is approximately 21.68 m\(^3\)/t, as is calculated by Langmuir's formula with parameters.
a=40.0 m$^3$/t and b=1.6 MPa$^{-1}$. Based on the principle that predictive indexes of coal seam outburst upgrading should guarantee clear and safe alert of outbursts, when gas content is lower than 12 m$^3$/t, the coal seam will not be outburst prone at all, when gas content falls in the range of 12 to 20 m$^3$/t, the coal seam should be managed as outburst threaten area and when gas content higher than 20 m$^3$/t, the coal seam will be determined as outburst potential.

Outburst potential assessment

Coal seam gas contents in the Main East obtained from prospecting period range from 6.88 to 11.28 m$^3$/t, and gas pressure from 0.1 to 0.29 MPa. Table 4 provides the gas contents measured underground, and the highest gas content is 10.97 m$^3$/t. There, the Main East is not prone to outbursts.

In the Main West, 19 gas pressures were measured by Chongqing Coal Research Institute (2005). The maximum gas pressure is 2.12 MPa, and the average gas pressure is 1.33 MPa. The gas contents obtained during coal field prospecting period are in the range of 10.7 to 26.80 m$^3$/t. According to the standards set up previously, as a whole, outbursts are limited to only a small area of the coal seam.

CONCLUSIONS AND SUGGESTIONS

The principal conclusions drawn from this study are as follows:

- The outburst on May 20, 2007, in Main West at Sihe Coal Mine was caused mainly by unusually large volume of gas stored in the coal seam;
- A gas content of 12 m$^3$/t has been established as the reliable and secure standard for assessment of coal seam outburst potential in Sihe Coal Mine;
- Main East at Sihe is not outburst prone; and to the contrary, Main West is evaluated as potential to outbursts;
- Collaborative research among universities and research institutes is necessary to determine and clarify the exact conditions of outburst occurrence in coal seams similar to Sihe Coal Mine.

REFERENCES

Wen, Z H, 2008. Experimental study on gas desorption laws of tectonically disturbed coals, Master’s Degree at Henan Polytechnic University, Jiaozuo.
CO₂ STORAGE IN ABANDONED COAL MINES

Paria Jalili, Serkan Saydam, Yildiray Cinar

ABSTRACT: Emissions CO₂ into the atmosphere have become a serious concern worldwide. Despite efforts to reduce the CO₂ concentration, global CO₂ emissions are increasing, and are estimated to be almost twice the current emissions by 2040 unless mitigating techniques are adopted. Many methods have been introduced to mitigate CO₂ emissions to the atmosphere. One of them is to capture CO₂ from stationary sources and transport, and store it in abandoned coal mines. CO₂ can be stored in abandoned coal mines in three states: free in empty spaces, adsorbed on the remaining coal or dissolved in mine water. The amount of storage could be significant depending on geological characteristics of mines and engineering design parameters. It would also be possible to recover some CH₄ which could offset the cost of the project. A review of the previous studies completed on CO₂ sequestration in abandoned coal mines around the world and preliminary assessments of the potential in Australia are presented.

INTRODUCTION

The concentration of CO₂ and other greenhouse gases, as a result of anthropogenic activities, is on the rise. The concentration of CO₂ had not changed in the past 650,000 years, varying between 180-300 ppm. However, in the last two centuries, because of using fossil fuels as a source for cheap and available energy, the amount of CO₂ in the atmosphere has increased significantly (389 ppm in 2010). Given the fact that fossil fuels will dominate the energy generation for the foreseeable future, the combustion of fossil fuels will continue to output significant amount of greenhouse gases.

Australia ranks first, among developed countries, with CO₂ emissions of 20.6 t per year, before the United States, which is producing 19.8 t per year (Maplecroft, 2009 and International Energy Agency - IEA, 2005). The power generation industry has the highest share for CO₂ emissions. Australia uses coal heavily for power generation which causes around 37% of the country’s total emissions (Department of Climate Change, 2007).

Initiatives such as the Kyoto Protocol have been put into the place in an attempt to technically and socially evolve society for less dependency on fossil fuels. The Kyoto Protocol between 39 industrialised countries and European Community (called Annex I countries) planned to reduce the greenhouse gases by 5.2% from the 1991 level over the five year period (2008-2012) (Kyoto Protocol, 1997).

Carbon Capture and Storage (CCS) is one of the proposed options for minimising global greenhouse gas emissions in the long-term. It involves capturing, compressing and injecting CO₂ deep underground. For deep underground storage sites, the options for CO₂ storage include unmineable coal seams, deep saline aquifers, depleted oil and gas reservoirs and abandoned coal mines.

THE CONCEPTS OF STORAGE

CO₂ can be stored in an abandoned coal mine through three physical mechanisms: by adsorption on the remaining coal, by solution in the mine water and by compression in the empty space of the mine. The adsorption of CO₂ on coal is the result of the van der Waals forces between the adsorbate (CH₄ or CO₂) and adsorbent (coal) (Piessens and Dusar, 2003b). It depends on the type of coal and also the amount of coal available for adsorption. Calculating the amount of coal available for adsorption is very important as most of the CO₂ would be adsorbed through this mechanism.

CO₂ is also expected to dissolve in ground water which has the less contribution amount compared to the other modes of storage (in most cases, less than 10% of the total sequestration amount). Residual space is the mined-out volume that remains after collapse and subsidence. The mined-out volume can easily be estimated from the amount of exploited coal and its density, augmented with the volume of extracted host rock. The volume lost due to collapse and subsidence is too difficult to estimate. It can be calculated from the total subsidence, if accurate data are available, or with a site-specific ratio between

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mined-out volume and residual space, depending on geology, mining and back-filling techniques. The latter approach requires data from drill cores or other in situ measurements (Piessens and Dusar, 2003b). The sum of these three amounts (adsorption on the remaining coal, dissolution in the mine water and free space storage) should represent the total sequestration capacity for an abandoned coal mine. The amount of storage for each mechanism varies from mine to mine depending on coal properties and the mine pressure.

CO₂ adsors on coal more than CH₄ (almost twice). By injecting CO₂ into the abandoned coal mine CH₄ may be released which can offset some of the cost of CO₂ sequestration.

Houtrelle (1999) introduced some basic conditions for gas storage in abandoned mines:

- There should be no lateral communication with other mines. This would allow the gas to migrate to places where it becomes difficult to retrieve.
- There should be no communication between the mine reservoir and the surface. This would allow the gas to leak to the atmosphere.
- The amount of the influx of water into the mine should preferably be low. The mine has to be kept dry by pumping for storage of CO₂ to prevent gas pressures which may cause leakage of the gas.

Piessens and Dusar (2003a) also suggested two criteria for CO₂ sequestration in abandoned coal mines:

- The top of the mine workings should be at least 500 m underground.
- The reservoir pressure should be 30% more than hydrostatic pressure to prevent water influx.

PROJECTS FOR CH₄ STORAGE IN ABANDONED MINES

Abandoned coal mines had been used to store natural gas since 1961. Three examples around the world have been reported. Two of these are located in Belgium (Peronnes and Anderlus Abandoned Coal Mines) and the other one is in the USA (Leyden Abandoned Coal Mine). These projects have been set up for meeting peak demands and avoiding the peak pricing of natural gas. All of these pilot projects faced some unforeseen problems like sealing of the shafts and production wells, flooding, compositional and pressure changes due to adsorption and desorption. However, the Leyden Coal Mine was positively evaluated, leading to an expansion of its capacity. The Peronnes and Anderlus Coal Mines were terminated due to an increase in local taxes and unforeseen costs (Piessens and Dusar, 2004).

Leyden coal mine (USA)

The Leyden gas storage operation was commenced in 1961 and is still in operation. The Leyden Mine was near Denver, Colorado, USA and was in operation from 1903 till 1950. During its operation, 6 Mt of sub-bituminous coal were produced. The mine consisted of four shafts providing the access to two horizontal seams from 240 to 260 m below the surface, in the upper cretaceous Laramie formation. The public Service Company of Colorado (PSCo) began storing gas at Leyden in the late 1950’s as a means of optimizing natural gas supplies. The original purpose of the Leyden storage facility was to ensure that PSCo could provide gas to the Denver area during peak demand times. It has still been used for this purpose; moreover they have used this as a key role in optimizing PSCo’s year-round gas buying and selling strategy.

The cap rock consists of 20 m claystone in the Leyden Mine. The leak off test shows that the Leyden Mine’s cap rock can only withstand 75% of the hydrostatic pressure which would be 1.8 MPa. During the initial development of the storage system, water was pumped from the mine as gas was being injected. Two active water wells are currently used to continuously remove approximately 50,000 m³ of water from the mine each year. It was a room and pillar operation and PSCo reports that the extraction efficiency was about 35%. This means still 65% of the original coal remained in place after the mining ended, primarily in the pillars. The EPA (Environmental Protection Agency - USA) estimates that the sorption capacity of the coal within the mine ranges from 85 to 127 Mm³ at the Leyden Mine facility’s, average operating pressure of 1.1 MPa and from 100 to 140 Mm³ at the facility’s maximum operating pressure of 1.8 MPa (Schultz, 1998).
Anderlus and Peronnes Mines (Belgium)

The Anderlus and Peronnes Mines, located in the gassy Hainaut Coalfield in Southern Belgium (between Mons and Charleroi) have been used for seasonal storage of natural gas. Figure 1 shows the locations of mines in Belgium. Anderlus mine operated between 1857 and 1969. 25 Mt of subbituminous coal were produced during its operation. Storage operation began in 1980 at depths from 600 to 1100 m. The overburden is almost 50 m thickness, but varies from South to North. A thrust fault acts as a primary hydrogeological barrier (Piessens and Dusar, 2004a). The reservoir volume assumed to be 6 to 10 Mm$^3$ which could be considered to store 180 Mm$^3$ of CH$_4$. Peronnes Mine is also located at the same area and it was in operation from 1860 until 1969. This mine could store 120 Mm$^3$ of CH$_4$.

![Figure 1 - The location of the mines in Belgium](image)

The storage operation for both Peronnes and Anderlus Mines were stopped in 1996 and 2000, respectively as an unforeseen cost for sealing the shafts was required.

PLANNING FOR STORAGE

An abandoned coal mine differs from other reservoirs in shape, height, permeability, initial pressure state and modes of storage. Hence it requires a specific tool to predict the reservoir performance. Piessens and Dusar (2003a) introduced a software package, named CO$_2$-VR, for calculating the storage capacity in abandoned coal mines. CO$_2$-VR is a reservoir simulator which is the combination of Micro-Excel and Visual Basic. It calculates the reservoir pressure and density of CO$_2$ at each depth. CO$_2$-VR is the only available software used for the simulation of CO$_2$ sequestration in abandoned coal mines. Below are the examples of the mines, for which CO$_2$-VR was used for their simulation for CO$_2$ storage.

Campine basin

The Campine Basin is located in the north of Belgium. The depth of the Campine Basin is reported to vary from 350 to 1090 m. The Basin contains seven mines, which had been operated between 1917 and 1992. The average temperature gradient for the Campine Basin is assumed to be 0.0035 °C/m with an average ambient temperature of about 9.8 °C. The water temperature at the deepest level of the mine was assumed to be 48 °C. All the shafts were filled with the reinforced concrete from 560 m to the surface. The cap rock is assumed to consist of cretaceous chalks and marls.

The Beringen is a typical colliery among these seven collieries for the Campine Basin. Almost 80 Mt of medium to high volatile A type bituminous coal was extracted. 85 Mm$^3$ is assumed as the mined-out volume by the 7% residual volume fraction (Van Tongeren and Laenen, 2001). This gives a residual volume space of 5.95 Mm$^3$. The maximum surface recovery is assumed to be 3-5 % of the original subsidence which is around 25-30 cm.
The ascertained sequestration capacity for a reservoir pressure being equal to 130% and the hydrostatic pressure is around 2.7 Mt CO₂ with the potential to sequester a total 5 Mt. When combined with the neighbouring and probably interconnected collieries, the combined ascertained capacity would be about 7 Mt CO₂, extendable to 13 Mt. This is sufficient to sequester between 0.3 to 0.5 Mtpa for 25 years (Piessens and Dusar, 2003a). The total ascertained capacity for all the Campine collieries including Zolder and Houthalen is 1 Mtpa for more than 15 years, with a potential to sequester for over 30 years at this rate (Piessens and Dusar, 2004a).

The case study for the Belgian collieries shows that sequestration is a viable option, even for the ascertained capacity alone.

### Table 1 - Estimated residual volume and sequestration capacity for the abandoned Campine coal mines (Piessens and Dusar, 2004a)

<table>
<thead>
<tr>
<th>Residual Volume (m³)</th>
<th>Ascertained Sequestration Capacity (Mt)</th>
<th>Potential Sequestration Capacity (Mt)</th>
<th>Total Sequestration Capacity (Mt)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Beringen</strong></td>
<td>(5.9 ± 2.3) 10⁶</td>
<td>3.0</td>
<td>2.5</td>
</tr>
<tr>
<td><strong>Zolder</strong></td>
<td>(6.2 ± 2.4) 10⁶</td>
<td>3.2</td>
<td>2.6</td>
</tr>
<tr>
<td><strong>Houthalen</strong></td>
<td>(1.8 ± 0.6) 10⁶</td>
<td>0.9</td>
<td>0.8</td>
</tr>
<tr>
<td><strong>Subtotal</strong></td>
<td><strong>13.9 ± 10⁶</strong></td>
<td><strong>7.1</strong></td>
<td><strong>5.9</strong></td>
</tr>
<tr>
<td><strong>Eisden</strong></td>
<td>(5.5 ± 2.1) 10⁶</td>
<td>2.8</td>
<td>2.3</td>
</tr>
<tr>
<td><strong>Waterschei</strong></td>
<td>(5.1 ± 1.9) 10⁶</td>
<td>2.6</td>
<td>2.2</td>
</tr>
<tr>
<td><strong>Winterslag</strong></td>
<td>(4.8 ± 1.1) 10⁶</td>
<td>2.4</td>
<td>2.0</td>
</tr>
<tr>
<td><strong>Zwartberg</strong></td>
<td>(3.0 ± 1.1) 10⁶</td>
<td>1.5</td>
<td>1.3</td>
</tr>
<tr>
<td><strong>Subtotal</strong></td>
<td><strong>18.4 ± 10⁶</strong></td>
<td><strong>9.4</strong></td>
<td><strong>7.8</strong></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>32.3 ± 10⁶</strong></td>
<td><strong>16.4</strong></td>
<td><strong>13.7</strong></td>
</tr>
</tbody>
</table>

**A typical mine from Upper Silesian Coal Basin, Poland**

For the purpose of study, one of the active Polish coal mines located in the Upper Silesian Coal Basin was selected and its storage capacity has been estimated (Lutyński, 2010). The shafts had been sealed from 440 m upward and the reservoir pressure at that depth was assumed to be around 5.85 MPa (not exceeding 30% more than hydrostatic pressure).

The results show that, at a reservoir pressure of 5.43 MPa, 3.5 Mt of CO₂ can be stored in the empty space as compressed gas and in mine water as dissolved. Almost a similar amount (3.53 Mt) of CO₂ can be stored in the remaining coal as adsorbed at the given pressure. In the calculations, the “worst case” scenario was taken into account deliberately. Moreover, there is a huge possibility of connections to remaining proven reserves. Both facts suggest this amount should be increased by 30% to determine the ascertained potential. This yields a storage capacity of 8.09 Mt of CO₂ for this coal mine at 5.43 MPa.

**POTENTIAL FOR CO₂ SEQUESTRATION IN AUSTRALIAN ABANDONED COAL MINES**

Australia has a huge source of coal and abundant coal mines, which can make the sequestration viable and bring the opportunities for CO₂ sequestration. With collaboration between the Schools of Mining Engineering and Petroleum Engineering at the University of New South Wales, a study has recently been initiated to assess the potential for CO₂ sequestration in abandoned coal mines located in New South Wales and Queensland.

Many factors have to be considered in a simulation of CO₂ sequestration in abandoned coal mines. These include depth, coal reserve, sealing, mine condition, mine water, and existent faults. The most important ones are assumed to be depth and total coal reserves. The deeper abandoned coal mines have higher reservoir pressures resulting in high capacity for CO₂ storage. The total coal reserve is important as almost half of the CO₂ would be adsorbed on the remaining coal compared to solution storage and compression storage.

Figure 2 shows the total coal reserve versus depth for the abandoned coal mines in New South Wales and Queensland. The results indicate that, although Queensland’s abandoned coal mines have higher coal reserves compared to the New South Wales’s coal mines, they do not have sufficient depth for CO₂ sequestration. Similarly, a few mines in New South Wales are deep enough but their reserves are not as...
large as Queensland’s coal mines. It is clear that more investigation needs to be done as the depth and coal reserve are not the only parameters given that many other factors like sealing, mine condition, mine water, existent faults may potentially affect the capacity. Investigation regarding these issues is ongoing and will be published soon.

Figure 2 - Total coal reserve versus depth data for the abandoned coal mines in New South Wales and Queensland (International Longwall News, 2010; Geoscience Australia, 2010)

CONCLUSIONS

According to Maplecroft (2009) and IEA (2005), Australia ranks first in per capita CO₂ emissions in the world just before the United States and Canada. Given the concerns that are rising about greenhouse gas emissions, the emissions need to be mitigated. CO₂ sequestration in abandoned coal mines can be one of the new methods as Australia has significant coal mines. Examples around the world have shown that this technique can be a viable option to reduce the amount of greenhouse gases liberated. The reasons for this include low initial investment cost as no transport is needed from the power plants and the possibility of incremental methane recovery which can offset some of the cost of CO₂ sequestration.

Like other methods, sequestering CO₂ in abandoned coal mines has several difficulties. Sealing of the shafts and faults is one of the first issues and has to be fully studied. Water influx is also very important as it could pressurize the CO₂ and may cause CO₂ leakage through the seals to the surface. In order to prevent water influx into the mine, the reservoir pressure should be more than the hydrostatic pressure (normally 30% more than hydrostatic pressure but may vary from mine to mine). This will cause an overpressure in the mine and may cause the leakage of CO₂ through the seal as well. The other issue is the lack of the data for abandoned coal mines. In most of the abandoned coal mines the leeway of water influx is not monitored after the closure. Most of the data using for simulations are not available as a certain value which may result in under or overestimation of the storage capacity. As a result, care must be taken in the screening studies.

The screening studies for CO₂ sequestration in abandoned coal mines in New South Wales and Queensland are currently ongoing at the University of New South Wales with the aim of determining the storage capacity and evaluating the possible methane recovery.

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REFERENCES


NUMERICAL SIMULATION OF THE DE-STRESSED DEFORMATION OF MINING COAL STRATA AT DIFFERENT PORE PRESSURES

Hongyong Liu, Yuanping Cheng, Haidong Chen, Shengli Kong and Qingwei Zhai

ABSTRACT: Gas extraction from mining induced de-stressed areas is one of the most efficient and cost-effective methods for eliminating coal and gas outbursts in China. In the paper, the characteristics of fracture induced pressure relief at different pore pressures and confining pressures on the de-stressed deformation of strata are analysed by using FLAC software. A strain-softening constitutive relation was used and the confining pressure was unloaded after the model was loaded to reach a static equilibrium state. The numerical results show that under the same confining pressure, and with the increase of pore pressure, the deformation of strata due to de-stressing as well as the fracture zones also increase significantly, and the characteristics of fissure induced pressure relief becomes more apparent. For seams with high gas pressure, the impact of pore pressure on the deformation of mining disturbed strata cannot be neglected. The numerical results are in good agreement with field observations, and improve the development of the coupled gas-solid theory in mining induced de-stressed strata.

INTRODUCTION

Coal seams characterized by low gas permeability usually less than $2.5 \times 10^{-4}$ md (0.01 m$^{-2}$/ (MPa$^{-2}$m$^{-1}$)), are regarded as the hard-to-drain coal seams in China. In order to drain coal gas efficiently, the coal seam permeability must be improved significantly. Various workers (Yu, 1986; Yu, 2005; Cheng, et al., 2003; 2004; Yu and Cheng, 2007) and practices proved that pressure-relief gas drainage, especially the use of the protective seam and pressure-relief gas drainage from the protected layer, was one of the most effective and cost-effective regional methods to eliminate coal and gas outbursts and reduce gas content. The Rules of Coal and Gas Outburst Prevention (hereinafter referred to as The Rules) were published in 2009 by the State Administration of Coal Mine Safety of China (SACMSC, 2009). The Rules require that the regional methods must be used as a priority for coal and gas outburst prevention, and local methods could be used as supplemental methods. The regional comprehensive methods, with the protected layer as priority selection, must be taken prior to the extraction of seams with outburst risk.

For different geologic conditions, the effects of gas control methods based on de-stressing can be different in the same coal field. Gas control methods should be selected to meet the depressurized effect and the requirement of The Basic Indicator of Gas Drainage in Coalmine (hereinafter referred to as The Indicator). Yuan (2005) showed that dilatational deformation could be used as the indicator of the depressurized effect for the protected layer, which related to the mining height of protective seam, roof control method, the dimension of the coal face and mining depth.

Much of the research work on the protective seam are based on physical-scale modelling or numerical calculations that are associated with geo-stress without due consideration of the pore pressure. When the gas pressure is high, the effective stress is much less than the total stress. Liang et al., (1995) showed gas pressure could affect the mechanical strength of the coal mass. Thus, pore pressure could directly influence the mechanical response of coal and rock mass, and consequently the simulations without taking pore pressure into consideration could not reflect the real depressurized effect on the protected layer. This makes it difficult for the use of the protective layer, especially when choosing the protective layer and designing the drainage holes and other projects for coal and gas outburst prevention, which also influences the effectiveness, safety and economical efficiency of pressure-relief gas drainage. Various workers (Liang, et al., 1995; Yao and Zhou, 1988; Zhao, 1992; Jin, et al., 1991; Xu, et al., 1993 and He, et al., 1996) found that the changing of mechanical strength, mechanical response and deterioration is influenced by pore pressure. Lu et al. (2001) updated the Karl Terzaghi

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effective stress formula of porous media, found that the regulation of effective pore pressure coefficient changes the complete stress-strain process. By using shear strain gradient plasticity theory, Wang and Pan (2001) established the relationship of stress and strain under the confining pressure and pore pressure. Yin and Wang (2009) and Wang et al. (2010) set up a coupled elastic-plastic damage constitutive model, and described the mechanical characters and behaviour of gas coal and rock at different loads. Using FLAC, Wang et al. (2009) numerically modelled a rock specimen with random defects in uniaxial plane strain compression, and investigated the effects of pore pressure on the failure processes, overall deformational characteristics and precursors.

In this paper, the characteristics of fissure induced pressure relief of mining coal mass at different pore pressures and confining pressures are analysed by using FLAC software. A strain-softening constitutive relationship is used and the confining pressure is unloaded after the model is loaded to reach a static equilibrium state based on the geological condition of Xinjing coal mine in Yangquan coal field.

**RESEARCH METHODS**

The hybrid discretization, dynamic relaxation method and explicit difference scheme was adopted. FLAC has the superiority for handling strain softening problems. It has been extensively used in mining coal and rock fracture analysis by various workers (Lan, et al., 2008; Peng, 2008 and Gao, et al., 2010). In FLAC, the pore pressure is described by effective stress law, and a positive value would be taken under the compressive condition. The relationship between effective stress \( \sigma' \) (negative when compressed), total stress \( \sigma_0 \) and pore pressure \( p \) could be described as:

\[
\sigma' = \sigma_0 + \alpha p
\]

Where, \( \alpha \) is the effective gas stress coefficient.

Assuming that the mechanical behaviour of coal and rock mass follows the strain-softening constitutive relationship, coal and rock mass is in isotropy linear elasticity during elasticity stage; after the peak strength, the constitutive relationship of coal and rock mass is M-C shear and tensile damage combined strain-softening model. The frictional angle and cohesion become lower with the elastic shear strain, until the frictional angle and cohesion reach the residual values.

**Numerical models**

The computation model is a standard cylinder, with a radius of 25 mm and 100 mm in height. Divided into 8 000 units, the vertical displacement of the cylinder bottom is fixed, and a radial load is imposed around the cylinder and on the top, corresponding to the larger friction between the sample and the press heads.

The model calculation uses the strain-softening model by FLAC software. The physical properties of the model are defined as in Table 1. The initial cohesion is set at 2.0x10^6 Pa and the initial frictional angle is 41°, the relationship of cohesion, frictional angle with plastic strain is shown in Figure 1. The numerical models are divided into six schemes with pore pressures respectively set as 0, 0.2 MPa, 0.4 MPa, 0.6 MPa, 0.8 MPa, 1.0 MPa, 1.2 MPa, and 1.4 MPa.

**Table 1 - Physical mechanics parameters of the numerical model**

<table>
<thead>
<tr>
<th>Density ( \rho ) (kg \cdot m^{-3})</th>
<th>Young modulus ( E ) (GPa)</th>
<th>Poisson ratio ( \mu )</th>
<th>Frictional angle ( \phi ) (°)</th>
<th>Cohesion ( c ) (MPa)</th>
<th>Tensile strength ( \alpha ) (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2 400</td>
<td>1.63</td>
<td>0.21</td>
<td>41</td>
<td>2.0</td>
<td>1.41</td>
</tr>
</tbody>
</table>

**Calculation steps**

1. (1) After creating the unexcavated model, the constitutive relation, boundary conditions and load conditions (confining pressure and internal pore pressure) are defined, and numerical computations started until the maximum unbalance force is less than 1.5x10^3 N.

2. (2) Using the Fish function to load the pressure on the sample top progressively, 100 kPa per section, and the static equilibrium would be disturbed by the loads.
(3) A new calculation process starts (FLAC adds a damping automatically) until a new static equilibrium state is reached.

**Axial strain calculation**

To study the effect of pore pressure on depressurized deformation of coal and rock mass, the numerical simulation will check and calculate the axial strain $\varepsilon_a$ as in Figure 1 during the calculation process:

$$\varepsilon_a = \frac{u}{L} \tag{2}$$

Where, $u$ is the displacement of rock sample top, in m; $L$ is the sample height, in m; $\varepsilon_a$ is the axial strain, if it is positive, the sample is compressed.

![Figure 1 - Relations of cohesion and frictional angle among plastic strain](image)

**SIMULATION RESULTS**

The compression experiments by cylindrical specimens are performed at different pore pressures under 1.0 MPa confining pressure, and induce the damage deformation at the pore pressures of 0, 0.4 MPa, 0.6 MPa, 0.8 MPa, 1.0 MPa, 1.2 MPa and 1.4 MPa. In order to compare the damage deformation at different confining pressures, another set of compression experiment is computed at the pore pressures of 1.2 MPa and 1.4 MPa under 2.0 MPa confining pressure.

**Influence of pore pressures on failure process under fixed confining pressure**

Figure 2 shows the relationship between axial stress and axial strain of testing a specimen at different pore pressures under 1.0 MPa of confining pressure.

![Figure 2 - Relationships between axial stress and axial strain of testing specimen at different pore pressures under 1.0 MPa confining pressure](image)

It can be seen from Figure 2 that the peak stress of damage failure and its corresponding axial strain decreases with the rising pore pressure. The higher the pore pressure, the smaller the declining rate.
after peak stress, and the mechanical characteristics become brittle from plastic. When the pore pressure is zero, the peak stress reaches 6.92 MPa and its corresponding axial strain is $3.94 \times 10^{-3}$. After the peak stress, the residual stress drops down to 5.44 MPa, falling 21.4%. While the pore pressure is increased to 1.4 MPa, the peak stress reaches 3.17 MPa and its corresponding axial strain $1.6 \times 10^{-3}$. But the residual stress is only down to 3.15 MPa after the peak stress, falling 0.63%. After the specimen has damaged, some plastic strain is caused by the larger load increment. Those observations demonstrated some related conclusions by Wang (2007) and Liang et al. (1995).

**Influence of pore pressures on failure process under different confining pressures**

The relationship between axial stress and axial strain of the testing specimen at different confining pressures under the same pore pressures 1.0 MPa and 2.0 MPa, can be shown in Figure 3.

![Figure 3 - Relationships between axial stress and axial strain of testing specimen at different confining pressures under the same pore pressures](image)

As it can be seen in Figure 3 the peak stress and the brittleness of specimen increases with increasing confining pressure, but the axial strain corresponding to peak stress becomes lower. Under 1.4 MPa pore pressure, when the confining pressure increases from 1.0 MPa to 2.0 MPa, the peak stress reaches 3.78 MPa from 3.58 MPa, and its corresponding axial strain drops to $1.59 \times 10^{-1}$ from $1.87 \times 10^{-1}$. While the pore pressure is 1.4 MPa, the peak stress rises to 3.45 MPa from 3.18 MPa, and its corresponding axial strain also drops to $1.31 \times 10^{-1}$ from $1.82 \times 10^{-1}$, and residual stress is 2.92 MPa and 2.99 MPa respectively.

**Influence of pore pressure on depressurized deformation of coal mass**

Plastic zone contains shear yield zone and tensile yield zone in FLAC. The plastic zone distribution at 0 and 1.4 MPa pore pressure under 2.0 MPa confining pressure is shown in Figure 4 (a) and (b).

![Figure 4 - Distributions of yielded elements at different pore pressure and time step under 2.0 MPa confining pressure](image)
The results indicate that when the confining pressure is fixed, the specimen only has shear fracture under the 0 pore pressure in the compression experiment. In Figure 4 (a), the specimen has tensile failure caused by stress concentration in adjacent units, while the pore pressure rises to 1.4 MPa from 0. The tensile failure appears at the ends of fissure under the pore pressure, caused by the tensile stress concentration for gas-filled coal containing cracks, according to the Griffith Strength Theory. Therefore, those simulation results agree with the theory and some related calculations as shown by Liang et al. (2010).

**SIMULATION CASE STUDIES**

In order to investigate the influence of pore pressure on depressurized failure and deformation, the remote protective layer exploitation was simulated at different pore pressures.

**Computation model**

The computation models are based on the geological features of a Xinjing coalmine. The geometrical characteristics, meshing and boundary conditions are shown in Figure 5.

![Figure 5 - Geometry features and boundary conditions of model](image-url)

The computation model has a dimension of 500 m × 210 m with an excavation length of 300 m in the No.3 coal seam. The horizontal displacement is fixed except the bottom and the top. The numerical calculation uses a large deformation mode. The burial depth of No.15 coal seam is 380 m, and the physical properties of the coal and strata in the simulation are shown in Table 2.

<table>
<thead>
<tr>
<th>No.</th>
<th>Name</th>
<th>Thickness (m)</th>
<th>Summation (m)</th>
<th>Bulk modulus K (GPa)</th>
<th>Shear modulus G (GPa)</th>
<th>Frictional angle ϕ (°)</th>
<th>Cohesion c (MPa)</th>
<th>Tensile strength T (MPa)</th>
<th>Cohesion softening slope Ck (GPa)</th>
<th>Frictional angle softening slope Fk (°)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Medium sandstone</td>
<td>27</td>
<td>210.3</td>
<td>27.4</td>
<td>18.8</td>
<td>38</td>
<td>6.4</td>
<td>4.17</td>
<td>0.573</td>
<td>2.0</td>
</tr>
<tr>
<td>2</td>
<td>Mudstone</td>
<td>3.4</td>
<td>183.3</td>
<td>1.80</td>
<td>0.929</td>
<td>30</td>
<td>2.67</td>
<td>2.50</td>
<td>0.426</td>
<td>1.5</td>
</tr>
<tr>
<td>3</td>
<td>Sandy mudstone</td>
<td>7.8</td>
<td>179.9</td>
<td>2.03</td>
<td>1.04</td>
<td>33</td>
<td>3.2</td>
<td>3.34</td>
<td>0.573</td>
<td>2.0</td>
</tr>
<tr>
<td>4</td>
<td>Coal</td>
<td>2.6</td>
<td>155.5</td>
<td>2.58</td>
<td>1.19</td>
<td>26</td>
<td>1.23</td>
<td>1.28</td>
<td>0.426</td>
<td>1.5</td>
</tr>
<tr>
<td>5</td>
<td>Limestone</td>
<td>3.2</td>
<td>66.7</td>
<td>10.5</td>
<td>6.91</td>
<td>37</td>
<td>4.7</td>
<td>6.83</td>
<td>0.573</td>
<td>2.0</td>
</tr>
</tbody>
</table>

Using the FISH function, the units in the mining area are deleted step by step. After deleting the units, the model will not maintain the equilibrium state under the pore pressure and the confining pressure. The model is recalculated until the unbalanced force is less than the defined value. The simulations are divided into three schemes with pore pressure 0, 1.0 MPa and 1.7 MPa.

**Simulation results**

The vertical stress on No.3 coal seam floor and the vertical relative deformation of the whole No.3 coal seam at different pore pressures are shown in Figures 6 and 7, when the protective layer has advanced 200 m.
As discussed previously, the peak stress of yield damage and its corresponding axial strain decrease with the rising pore pressure, and the mechanical characteristics become brittle from plastic. The peak stress concentration is increased with the rising pore pressure after a certain distance of the protected layer being mined, shown in Figure 6. As shown in Figure 7, the range of the dilatational deformation area and the deflection also increased, which causes significant compression in the coal block.

After the protected layer advanced 200 m, the depressurized length of protected layer is 160 m at the pore pressure 0. In the most depressurized zone, the stress became tensile from compressive at 0.364 MPa, and corresponding relative deformation is 1.3‰. The peak concentration stress in the both sides of the coal block reaches -2.45 MPa, and the stress concentration factor is 1.63, and corresponding compression deformation -1.4‰ at 20 m to the edge of coal seam. While the pore pressure rose to 1.7 MPa, the depressurized length increased to 220 m, the minimum stress is -0.417 MPa, i.e., about 27.8% of the initial value, and the depressurized deformation reached 81.2‰ in the most depressurized zone. Meanwhile, the peak concentrated stress at 45 m to the edge of coal block reached -2.62 MPa, and the stress concentration factor 1.75, the max compression deformation is -1.0‰, and the value of above parameters is in the same order of magnitude as the 0 while the pore pressure is increased to 1.0 MPa.

These results showed that the depressurized deformation and the damaged area increased with the rising pore pressure. And the characteristics of depressurized fissure are also becoming more obvious. The stress concentration in both coal seams becomes higher, and the distance from the stress concentration zone to the edge of coal block is also larger. If the pore pressure is not considered, the depressurized effect and the position of peak stress concentration would be underestimated. This will have a hidden danger to the design of gas control projects and the safe mining of protected layer. Obviously when the pore pressure reaching a certain value, the effect of pore pressures on the depressurized deformation of coal mass cannot be neglected.

Simulation results also indicated that the height of the water-conducting crack zone (zone and fissure zone) is over 135 m, which means that the No.3 coal seam, locating above the fissure zone of No.15 coal seam, will have well-developed fractures and increased permeability, therefore the effect of pressure-relief gas drainage can be achieved.
CONCLUSIONS

Based on the systematic studies of mechanical characteristics of coal and rock mass due to mining, the following conclusions can be reached:

1. The peak stress of damage failure and its corresponding axial strain decreases with the rise of pore pressure, when the confining pressure is fixed. The higher the pore pressure, the smaller the rate of reduction after peak stress, and the mechanical characteristics become brittle from plastic.

2. When the confining pressure is fixed, on shear fracture occurs under the 0 pore pressure in the compression experiment. The specimen has tensile failure caused by stress concentration in adjacent units, as the pore pressure increases to 1.4 MPa from 0.

3. The simulation based on protective layer exploitation in Yangquan Xinjin coal mine shows that the depressurized area, the peak stress concentration and the position increase with the rising pore pressure after a certain distance of protected layer have been mined. When the pore pressure reaches a certain value, the effect of pore pressure on the depressurized deformation of coal mass must be considered.

4. As the numerical simulation shown, No.3 coal seam located above the fissure zone of No.15 coal seam. After the mining of protective seam, mining induced cracks can be well-developed with enhanced permeability in the protected seams, thus facilitating the practice of efficient gas drainage.

5. The numerical results are in good agreement with field observations, which improve the gas-solid coupled theory of mining coal mass.

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REFERENCES


GOAF INERTISATION AND SEALING UTILISING METHANE FROM IN-SEAM GAS DRAINAGE SYSTEM

C Claassen

ABSTRACT: In the past the process for sealing longwall goafs at Mandalong has been to simply seal access points to the goaf and monitor the goaf as it self inertised. Due to changes in legislative requirements an improved method was required. A number of different methods utilising N\textsubscript{2} or CO\textsubscript{2} have been utilised at other mine sites. However, at Mandalong it was decided to use methane from the in-seam gas drainage system to purge and inertise the goaf. According to our knowledge this is the first time this method of utilising methane for goaf inertisation and sealing has been implemented.

The aim of the sealing process is to seal the goaf in a safe manner without disruption to other parts of the mine. This is achieved by controlling the inertisation process through the introduction of methane from the gas drainage system with the intent of purging the critical zone of the unsealed goaf of any oxygen.

Methane from the mine’s gas drainage system and existing pipe arrangement is re-directed to the seals behind the longwall take-off face and injected into the goaf fringe under seam pressure. The goaf atmosphere is monitored via a tube bundle system and is allowed to enter and exit the explosive range under controlled conditions. When the tube bundle monitoring shows the goaf atmosphere is inert, final sealing of the goaf is carried out.

The principal hazard associated with the sealing of a goaf area in a gassy mine is the ignition of an explosive atmosphere resulting in an explosion. To reduce this risk to as low as reasonably possible (ACARP) numerous controls are implemented.

Mandalong has successfully utilised this method four times since February 2008. Longwall 5 (LW5), Longwall 6 (LW6), Longwall 7 (LW7), Longwall 8 (LW8) and Longwall 9 (LW9) were sealed in this manner, and it is intended future longwall goafs be sealed utilising the same methodology. The results of these will be presented and discussed.

INTRODUCTION

Mandalong Mine is located 50 km south of Newcastle, New South Wales, Australia. The mine operates a 150 m wide retreat longwall system in the West Wallarah Seam of the Newcastle Coalfield. The seam varies in thickness from 3.5 to 6.5 m and has a gas content up to 6 m\textsuperscript{3}/t. The predominant seam gas constituent is methane. In-seam gas drilling and drainage is applied to the seam to lower the gas content to sufficient levels to prevent statutory limits being exceeded in the mine general body gas make.

The Fassifern seam underlies the West Wallarah seam at 4 to 8 m with a gas content of 4 to 6 m\textsuperscript{3}/t. During LW retreat the interburden between the West Wallarah and Fassifern seam is fractured, liberating methane from the Fassifern seam to the active LW goaf. This in turn creates self-inertisation of the active goaf at a distance of approximately 1 000 m inbye of the working LW face.

GOAF SEALING AND RESULTS

Sealing LW1 – LW4

Sealing of the first LW goafs consisted of constructing 138 kPa (20 psi) seals (with man doors in the seals) in the maingate and tailgate. These seals were constructed in parallel to LW face recovery. Upon completion of face recovery, a 138 kPa (20 psi) seal was constructed at the take-off cut through location and upon completion of the seal in the take-off cut through, the man doors in the seals in maingate and tailgate were closed, effectively sealing the goaf. The goaf was then allowed to self-inertise over time, as can be seen from the gas trend attached in Figure 1.
Legislative sealing requirements

Since 2007 the requirements of the Department of Industry and Investment (previously Department of Primary Industries) is that goaf areas be inertised prior to final sealing. Coal Mine Health and Safety Regulations 2006 clause 49 relating to High Risk Activities (NSW Govt, 2006) require that notification be submitted to the Department of Industry and Investment at least 30 days prior to sealing.

Inertisation options available

To achieve an inert atmosphere in the goaf area, different mines have adopted different methods; either nitrogen or carbon dioxide is used, with nitrogen appearing to be the more commonly used method. Mandalong assessed the options available and opted to utilise methane (CH\textsubscript{4}) from its in-seam gas drainage system.

Seal specifications

All seals constructed for purposes of goaf containment are constructed to minimum 138 kPa (20 psi) rating. Line seals with a life span of less than 18 months are concrete mesh block type, while life-of-mine (LOM) seals are of a type able to withstand a degree of floor heave and roof convergence evident in the Mandalong workings.

Sequence and methodology

Monitoring has shown that self-inertisation of the goaf generally occurs approximately 1000 metres behind the LW operating face. Self-inertisation occurs primarily due to the liberation of CH\textsubscript{4} from the underlying Fassifern seam via floor cracks between the Fassifern seam and West Wallarah seam subsequent to LW retreat, and is assisted to some small degree by CH\textsubscript{4} liberation from coal left in some parts of the goaf area.

In lieu of this, in February 2008 Mandalong utilised the method of injecting methane (CH\textsubscript{4}) into the goaf for purging and inertisation of the goaf for the first time. CH\textsubscript{4} is injected at seals inbye of the recovery
face. Methane is conducted under seam pressure via the existing gas drainage range and 4 inch (100 mm) diameter pipes installed from the gas range to the seals during the pre-work phase.

Once the E-frame is established in the tailgate, construction of the final maingate (MG) and tailgate (TG) seal commences. Both seals on the MG and TG side are equipped with man doors to allow for continued airflow via these sites as required. Upon completion of the TG seal, CH₄ is directed to the seals on the maingate (MG) side inbye of the recovery face, with flows of 300 – 600 L/s achieved under normal seam pressure. Ventilation flow across the recovery face is maintained at 30 - 40 m³/s to allow for operation of diesel equipment within prescribed statutory limits. CH₄ injected on the maingate side of the goaf serves to displace oxygen from the goaf atmosphere during face recovery.

Once face recovery is completed, the airflow across the recovery face is regulated to 10 m³/s. A 138 kPa (20 psi) seal at the take-off face cut through is constructed while face ventilation is maintained via man doors in the MG and TG seals as shown in Figure 2.

Figure 2 - Methane injection on MG side purging oxygen from goaf atmosphere, while maintaining ventilation across the recovery face

Upon completion of the seal in the take-off cut through, the man door in the MG seal is closed and flow at the TG seal is reduced to < 2 m³/s. CH₄ injection via the take-off cut through seal commences simultaneously as shown in Figure 3.

The quantity of air/gas mixture flowing from the TG seal is reduced incrementally with the aim of achieving a higher volume of gas input on the MG side than air/gas bleeding from the TG seal, thus pressurising and purging oxygen from the goaf atmosphere.

Prior to final sealing a comprehensive checklist is completed to ensure that all actions are completed. The spontaneous combustion TARP (Trigger Action Response Plan) applies before, during and post the inertisation and sealing process. Should any spontaneous combustion indicators be present, actions as per the TARP are implemented.

Monitoring

Monitoring of the goaf atmosphere occurs via tube bundle points installed in the TG seal and selected MG seals. The tube bundle gas analyser is situated on surface and gas analysis results are accessible via a number of terminals.

Routine bag samples are also taken and analysed by gas chromatograph. Sealing/purging progress via tube bundle is monitored via Safegas and the results trended utilising either Ellicott’s or Coward’s triangle, both facilities of which are incorporated in the Safegas/Segas trending software. Final sealing does not occur until such time as the monitoring results indicate that the goaf atmosphere has reached at least < 80% explosibility as shown by Figure 4. This allows for a “barrier” to compensate
for fluctuations in goaf atmosphere composition due to barometric fluctuations and migration of CH$_4$ away from low lying areas in the seam, ingress of oxygen into the goaf via seals due to pressure differentials created by operational conditions, etc.

Figure 3 - CH$_4$ injection via MG seals continues while airflow via TG is reduced

Figure 4 - Coward's triangle showing 80% explosibility parameter

Results

The Safegas trend in Figure 5 shows a typical CH$_4$ versus oxygen trend over time for the TG seal during the inertisation process while CH$_4$ in injected on the MG side.
The objective during inertisation is to achieve CH$_4$ content >25% and oxygen content less than 10%. What is achieved in reality is a CH$_4$ content of 30-70% and an oxygen content of <2%, with the balance being nitrogen. Some low level carbon monoxide (CO) is present behind the immediate LW face during LW retreat, due to low level oxidation of fractured coal exposed to oxygen. In addition to this varying levels of ethane (C$_2$H$_6$) is detected by gas chromatography, and hydrogen (H$_2$) of up to 30 ppm have been detected in select locations in the older goafs, due to the hydrogen being in isolated pockets it is presumably the result of chemical reaction of water and galvanised pipes.

Figure 6 shows the corresponding CO level at the TG seal pre- and post-sealing. As mentioned before, CO levels in the goaf atmosphere are relatively low. CO presence and trending can thus be used as an accurate early spontaneous combustion indicator.
CONCLUSIONS

CH₄ is used to displace oxygen from the goaf atmosphere, thus creating an inert atmosphere. The process is closely monitored for signs of spontaneous combustion and a well defined TARP applies during and after sealing. A rigorous monitoring system is in place in the form of both tube bundle monitoring and bag sampling/gas chromatography.

REFERENCE

ANALYSIS OF ETHANE EMISSION TRENDS FROM ACTIVE GOAF SEALS AT MANDALONG MINE

M Leal¹, B Beamish¹,² and C Claassen³

ABSTRACT: Monitoring of active goaf seals at Mandalong Mine shows anomalous levels of ethane concentrations. Currently ethane concentrations exceeding 250 ppm have been experienced from 30 to 32 C/T seals of MG8 with the remainder of the panel averaging 50 - 100 ppm. Active goaf seals for MG7 and MG9 average 50 - 100 ppm and 80 - 100 ppm, respectively. For most mines these concentrations of ethane would be assumed to indicate a spontaneous combustion event; however no carbon monoxide is being recorded and a sympathetic relationship with methane indicates that the measured ethane is due to gas desorption. This paper presents the results from the mine showing trends in ethane emissions from the active goaf seals to date, and shows how this historical data can be used to predict expected ethane concentrations in the active goaf of MG10.

INTRODUCTION

Mandalong Mine is located 50 km south of Newcastle, New South Wales, Australia. The mine operates a 150 m wide retreat longwall system in the West Wallarah Seam of the Newcastle Coalfield. The seam varies in thickness from 3.5 to 6.5 m and has moderate gas content up to 6 m³/t. The predominant seam gas constituent is methane, but ethane is also present as a subordinate component in appreciable amounts. In-seam gas drilling and drainage is applied to the seam to lower the gas content to sufficient levels to prevent statutory limits being exceeded in the mine general body gas make.

Anomalous concentrations of ethane experienced in recent active goaf areas has been a cause of concern to the operations as ethane is generally linked to a possible heating of coal (Beamish and Jabouri, 2005; Claassen and Beamish, 2010). This has prompted an investigation into the source and trends of ethane emissions within the active goaf. Data for the first three months of monitoring each cut-through seal has been analysed to determine localised trends and averaged to determine panel wide trends. This paper presents the results of this investigation including a model of the ethane emissions in the active goaf.

GAS SAMPLING AND RESULTS SUMMARY

Summary of gas data

Gas chromatographic results of gasbag samples from active seal samples at Mandalong Mine have been analysed to determine and interpret trends in ethane emission data. To standardise the results, data from the first three months of monitoring for each seal was analysed. The active goaf areas investigated were Maingate 7 (MG7), Maingate 8 (MG8) and Maingate 9 (MG9). The average ethane concentrations for the first three months of monitoring 32 C/T to 19 C/T of MG7, MG8 and MG9 are contained in Table 1. The corresponding values for 18 C/T to 3 C/T are contained in Table 2.

Active goaf trends

Maingate 7 active goaf trends

Average ethane concentrations for 32 C/T to 30 C/T are around 100 ppm and remain relatively stable in the first three months of monitoring (Figure 1). For the outbye cut-throughs the ethane concentrations decrease to values between 50 and 80 ppm (Figure 1). Methane and ethane concentrations along MG7 cut-through seals show a sympathetic relationship, as shown for 29 C/T to 24 C/T in Figure 2, while carbon monoxide is generally absent.

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² B3 Mining Services Pty Ltd, Kenmore QLD 4069, b.beamish@uq.edu.au
³ Centennial Coal, Mandalong Mine, Mandalong NSW 2264
### Table 1 - Average ethane results for maingate cut through seals, 32 C/T - 19 C/T

<table>
<thead>
<tr>
<th>Maingate seal</th>
<th>MG7 Ethane (ppm)</th>
<th>MG8 Ethane (ppm)</th>
<th>MG9 Ethane (ppm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>32 C/T</td>
<td>104 ± 3</td>
<td>(31/03/09 – 15/05/09)</td>
<td>173 ± 19</td>
</tr>
<tr>
<td>31 C/T</td>
<td>105 ± 7</td>
<td>(31/03/09 – 21/05/09)</td>
<td>163 ± 28</td>
</tr>
<tr>
<td>30 C/T</td>
<td>98 ± 8</td>
<td>(1/02/09 – 2/05/09)</td>
<td>195 ± 22</td>
</tr>
<tr>
<td>29 C/T</td>
<td>63 ± 18</td>
<td>(31/03/09 – 22/05/09)</td>
<td>99 ± 10</td>
</tr>
<tr>
<td>28 C/T</td>
<td>68 ± 12</td>
<td>(2/03/09 – 2/06/09)</td>
<td>90 ± 12</td>
</tr>
<tr>
<td>27 C/T</td>
<td>67 ± 19</td>
<td>(31/03/09 – 22/06/09)</td>
<td>81 ± 3</td>
</tr>
<tr>
<td>26 C/T</td>
<td>61 ± 19</td>
<td>(31/03/09 – 16/06/09)</td>
<td>105 ± 9</td>
</tr>
<tr>
<td>25 C/T</td>
<td>56 ± 12</td>
<td>(31/03/09 – 20/05/09)</td>
<td>64 ± 10</td>
</tr>
<tr>
<td>24 C/T</td>
<td>54 ± 8</td>
<td>(31/03/09 – 16/06/09)</td>
<td>54 ± 3</td>
</tr>
<tr>
<td>23 C/T</td>
<td>56 ± 4</td>
<td>(31/03/09 – 22/06/09)</td>
<td>77 ± 24</td>
</tr>
<tr>
<td>22 C/T</td>
<td>54 ± 12</td>
<td>(2/03/09 – 2/06/9)</td>
<td>51 ± 4</td>
</tr>
<tr>
<td>21 C/T</td>
<td>47 ± 9</td>
<td>(31/03/09 – 22/06/09)</td>
<td>53 ± 5</td>
</tr>
<tr>
<td>20 C/T</td>
<td>55 ± 6</td>
<td>(1/09/09 – 22/06/09)</td>
<td>55 ± 4</td>
</tr>
<tr>
<td>19 C/T</td>
<td>51 ± 19</td>
<td>(31/03/09 – 22/06/09)</td>
<td>54 ± 8</td>
</tr>
</tbody>
</table>

### Table 2 - Average ethane results for maingate cut through seals, 18 C/T - 3 C/T

<table>
<thead>
<tr>
<th>Maingate seal</th>
<th>MG7 Ethane (ppm)</th>
<th>MG8 Ethane (ppm)</th>
<th>MG9 Ethane (ppm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>18 C/T</td>
<td>60 ± 7</td>
<td>(31/03/09 – 16/06/09)</td>
<td>57 ± 8</td>
</tr>
<tr>
<td>17 C/T</td>
<td>57 ± 3</td>
<td>(31/03/09 – 3/05/09)</td>
<td>63 ± 10</td>
</tr>
<tr>
<td>16 C/T</td>
<td>59 ± 12</td>
<td>(2/04/09 – 16/06/09)</td>
<td>56 ± 6</td>
</tr>
<tr>
<td>15 C/T</td>
<td>63 ± 14</td>
<td>(2/04/09 – 22/06/09)</td>
<td>55 ± 3</td>
</tr>
<tr>
<td>14 C/T</td>
<td>58 ± 7</td>
<td>(20/04/09 – 16/06/09)</td>
<td>59 ± 6</td>
</tr>
<tr>
<td>13 C/T</td>
<td>67 ± 5</td>
<td>(25/04/09 – 2/06/09)</td>
<td>53 ± 7</td>
</tr>
<tr>
<td>12 C/T</td>
<td>77 ± 16</td>
<td>(18/05/09 – 22/06/09)</td>
<td>53 ± 2</td>
</tr>
<tr>
<td>11 C/T</td>
<td>69 ± 16</td>
<td>(25/04/09 – 17/06/09)</td>
<td>67 ± 3</td>
</tr>
<tr>
<td>10 C/T</td>
<td>69 ± 26</td>
<td>(22/06/09 – 10/09/09)</td>
<td>57 ± 6</td>
</tr>
<tr>
<td>9 C/T</td>
<td>77 ± 17</td>
<td>(18/05/09 – 16/06/09)</td>
<td>58 ± 4</td>
</tr>
<tr>
<td>8 C/T</td>
<td>74 ± 24</td>
<td>(8/05/09 – 22/06/09)</td>
<td>60 ± 1</td>
</tr>
<tr>
<td>7 C/T</td>
<td>69 ± 17</td>
<td>(20/05/09 – 16/06/09)</td>
<td>67 ± 6</td>
</tr>
<tr>
<td>6 C/T</td>
<td>78 ± 17</td>
<td>(21/05/09 – 22/06/09)</td>
<td>68 ± 1</td>
</tr>
<tr>
<td>5 C/T</td>
<td>60 ± 23</td>
<td>(16/06/09 – 20/10/09)</td>
<td>60 ± 5</td>
</tr>
<tr>
<td>4 C/T</td>
<td>81 ± 15</td>
<td>(21/05/09 – 10/09/09)</td>
<td>81 ± 9</td>
</tr>
<tr>
<td>3 C/T</td>
<td>102 ± 22</td>
<td>(12/01/10 – 20/04/10)</td>
<td>81 ± 9</td>
</tr>
</tbody>
</table>
Maingate 8 active goaf trends

Average ethane concentrations for 32 C/T to 30 C/T are significantly higher in MG8 seals compared with MG7 and it is not until 27 C/T before the ethane concentrations begin to decrease to the 50 – 80 ppm range of the outbye seals in MG7 (Figure 1). A second anomalous zone of high ethane also occurs at 4 C/T.

Maingate 9 active goaf trends

Average ethane concentrations for 30 C/T to 25 C/T are elevated in the range of 80 – 100 ppm before returning to the 50 – 80 ppm range at the outbye seals, except for 3 C/T (Figure 1). Once again a sympathetic relationship is present between ethane and methane emissions.

Figure 1 - Average ethane concentrations for MG7, MG8 and MG9

Figure 2 - Maingate 7 methane and ethane concentrations on air free basis (29 C/T – 24 C/T)
ETHANE PREDICTION MODEL

All of the average ethane values obtained from the active goaf seals in MG7, MG8 and MG9 have been used to create a grid model in the surface modelling package Surfer® 8 using the Kriging method. The gridded values have been contoured to generate a 3D surface model of the ethane concentrations (Figure 3). The anomalous ethane zone shows as a localised area in LW8, extending from 32 C/T to 28 C/T. The kriged values obtained using Surfer can also be used to predict the ethane concentrations for each of the active seals in MG10. These values are contained in Table 3 and they show that ethane concentrations range from 68 to 97 ppm.

![Figure 3 - 3D surface model of ethane concentrations for Mandalong Mine active goaf areas](image)

**Table 3 - Maingate 10 predicted ethane concentrations**

<table>
<thead>
<tr>
<th>Location</th>
<th>Ethane (ppm)</th>
<th>Location</th>
<th>Ethane (ppm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>3 C/T</td>
<td>97</td>
<td>18 C/T</td>
<td>78</td>
</tr>
<tr>
<td>4 C/T</td>
<td>93</td>
<td>19 C/T</td>
<td>79</td>
</tr>
<tr>
<td>5 C/T</td>
<td>86</td>
<td>20 C/T</td>
<td>80</td>
</tr>
<tr>
<td>6 C/T</td>
<td>80</td>
<td>21 C/T</td>
<td>79</td>
</tr>
<tr>
<td>7 C/T</td>
<td>74</td>
<td>22 C/T</td>
<td>80</td>
</tr>
<tr>
<td>8 C/T</td>
<td>71</td>
<td>23 C/T</td>
<td>82</td>
</tr>
<tr>
<td>9 C/T</td>
<td>70</td>
<td>24 C/T</td>
<td>85</td>
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<tr>
<td>10 C/T</td>
<td>69</td>
<td>25 C/T</td>
<td>87</td>
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<td>11 C/T</td>
<td>68</td>
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</tr>
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<td>12 C/T</td>
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<td>27 C/T</td>
<td>85</td>
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<tr>
<td>13 C/T</td>
<td>70</td>
<td>28 C/T</td>
<td>79</td>
</tr>
<tr>
<td>14 C/T</td>
<td>72</td>
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</tr>
<tr>
<td>15 C/T</td>
<td>74</td>
<td>30 C/T</td>
<td>70</td>
</tr>
<tr>
<td>16 C/T</td>
<td>75</td>
<td>31 C/T</td>
<td>71</td>
</tr>
<tr>
<td>17 C/T</td>
<td>77</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

**CONCLUSIONS**

The presence of ethane in recent active goaf areas appears to be due to seam gas desorption and not coal oxidation due to the lack of consistent carbon monoxide readings, as well as the sympathetic relationship with methane. The 32 C/T to 28 C/T area of MG8 experienced a localised spike in ethane.
concentrations, possibly due to an increased level of desorbed ethane, or an increase in seam thickness resulting in a higher quantity of coal in the goaf. Based on ethane trends for the first three months of each cut through seal of MG7, MG8 and MG9, a gridded model has been developed using kriging to produce a 3D surface model of the ethane concentrations at each seal. This model has also been used to predict the expected ethane concentrations in the first three months for each cut-through seal of MG10. As these values become available they will be used to validate and refine the model.

ACKNOWLEDGEMENTS

The authors would like to thank Centennial Coal and in particular Mandalong Mine for supplying the goaf data for analysis.

REFERENCES


EXPERIENCE WITH USING A MOIST COAL ADIABATIC OVEN TESTING METHOD FOR SPONTANEOUS COMBUSTION ASSESSMENT

B Beamish\(^1,2\) and R Beamish\(^2\)

ABSTRACT: Many small-scale coal spontaneous combustion tests exist, but none of these appear to be comprehensive and definitive in terms of assessing the self-heating behaviour of coal at low temperatures from ambient to thermal runaway. Within this low temperature region there are a number of competing influences on the tendency of the coal to continue to gain heat. One of the key influences is the presence of moisture in the coal as this contributes to heat loss from evaporation. The presence of disseminated pyrite can contribute to heat gain from the additional exothermic reaction with oxygen in the presence of moisture. A new moist coal adiabatic oven test has been developed that can measure the influence of these parameters on the low temperature self-heating of coal. Examples of the experience gained from using this test clearly show the effects of moisture and pyrite on coal self-heating leading to thermal runaway. The results obtained have major consequences for the operational planning and management of the spontaneous combustion hazard.

INTRODUCTION

The influence of moisture on coal self-heating has been investigated in a number of studies. It is generally accepted that there are competing influences of heat of wetting and moisture evaporation depending on the environmental circumstances of the coal (Hodges and Hinsley, 1964; Guney, 1971; Bhattacharyya, 1971, 1972; Bhat and Argarwal, 1996). Numerical model studies by Akgun and Essenhigh (2001) showed that moisture effects on self-heating in a broken coal pile situation are two-fold. In the case of low moisture content coals, the maximum temperature increases steadily with time. In the case of high moisture coals, temperature increases rapidly initially before evaporation dominates and the temperature reaches a plateau value (generally around 80-90°C). Once the coal becomes dry locally the temperature will increase rapidly towards thermal runaway. However, if the coal pile has been in a prolonged drying phase that is interrupted by a rain event and the water penetrates into the pile then additional heat can be generated from the heat of wetting effect as the coal re-adsorbs the moisture available to it. This effect can also lead to premature thermal runaway in the coal pile. The presence of pyrite may also complicate the heat balance situation and is identified under certain circumstances as a heat contributor.

Development of a standard laboratory test to benchmark moisture or pyrite effects on coal self-heating has not been achieved to date. Instead a number of tests have been developed to rate the propensity of coal for spontaneous combustion (Nelson and Chen, 2007). In the Australian and New Zealand coal industries there is one test that is routinely used. This is the adiabatic oven R\(_{70}\) self-heating rate test (Beamish, Barakat and St George, 2000, 2001; Beamish and Arisoy, 2008a), which has been used to show the effects on coal self-heating rate of rank (Beamish, 2005), type (Beamish and Clarkson, 2006) and mineral matter (Beamish and Blazak, 2005; Beamish and Sainsbury, 2008; Beamish and Arisoy, 2008b). The R\(_{70}\) self-heating rate is a low temperature oxidation spontaneous combustion index parameter that is measured on dried coal from a start temperature of 40 °C. The relationship of this parameter to thermal runaway performance of as-mined coal has been interpreted on an inferred basis by comparison with coals that have similar R\(_{70}\) values and coal characteristics. As such a reactivity rating scale has been developed for both New South Wales and Queensland conditions using this parameter.

Recent studies by Beamish and Hamilton (2005) and Beamish and Schultz (2008) have re-emphasised the importance of moisture on coal self-heating. As a result of these studies, Beamish and Beamish (2010) proposed a new moist coal adiabatic oven test that can be used to benchmark laboratory performance against actual site performance of a range of coals from Australia and overseas that cover the rank spectrum from sub-bituminous to high volatile A bituminous. Since introducing this test to the

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coal industry a number of additional benchmark coals have been added to the database and tests have been conducted that show this new method is extremely accurate and definitive for assessing the spontaneous combustion risk of coal in a range of environmental situations. Some specific examples of this more recent testing technique and highlights the significance to operational planning and management are presented.

**ADIABATIC OVEN TESTING**

**Coal samples**

Details of the samples used in this study are contained in Table 1. The three major benchmark coals are Kideco (Indonesia), Spring Creek (New Zealand) and Rocglen (Australia). The Bowen Basin test samples were received as fresh core samples from exploration boreholes. They were appropriately sealed in gladwrap to prevent oxidation and tightly bound with duck tape to maintain sample integrity as solid cores. Representative splits were taken from each core length for testing.

Sample BBHVB01 was from the uppermost part of the seam in one borehole and sample BBHVB07 was from the equivalent horizon in another borehole from the deposit. Samples BBHVB08 and BBHVB10 were from lower horizons in the seam of the same borehole as BBHVB07. All of these samples have an ASTM rank of high volatile C bituminous based on the volatile matter and calorific value of the coal.

**Table 1 - Coal quality data and test parameters for benchmark and Bowen Basin coal samples**

<table>
<thead>
<tr>
<th>Sample</th>
<th>$R_{70}$ (°C/h)</th>
<th>Volatile matter (% dmmf)</th>
<th>Calorific value (Btu/lb, mmmf)</th>
<th>ASTM rank</th>
<th>Ash content (% db)</th>
<th>Sulphur content (% db)</th>
<th>Moisture content (%)</th>
<th>Start temperature (°C)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Benchmark coals</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
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</tr>
<tr>
<td>Kideco</td>
<td>28.57</td>
<td>51.6</td>
<td>9755</td>
<td>subC</td>
<td>1.8</td>
<td>0.10</td>
<td>24.0</td>
<td>24.4</td>
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<tr>
<td>Spring Creek</td>
<td>5.87</td>
<td>41.3</td>
<td>13749</td>
<td>hvBb</td>
<td>1.2</td>
<td>0.30</td>
<td>11.7</td>
<td>27.0</td>
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<td>Rocglen</td>
<td>3.18</td>
<td>45.8</td>
<td>14664</td>
<td>hvAb</td>
<td>4.9</td>
<td>0.45</td>
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</tr>
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<td><strong>Bowen Basin high volatile bituminous coal</strong></td>
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<td>2.3</td>
<td>0.38</td>
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<td>12885</td>
<td>hvC</td>
<td>7.3</td>
<td>4.54</td>
<td>9.8/9.3*</td>
<td>27.2</td>
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<tr>
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<td>12845</td>
<td>hvC</td>
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<td>27.0</td>
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<td>34.7</td>
<td>12882</td>
<td>hvC</td>
<td>5.9</td>
<td>0.45</td>
<td>6.9*</td>
<td>29.1</td>
</tr>
</tbody>
</table>

# Moisture contents of initial/repeat test; * 5% moisture removed from sample

**Self-heating test procedure**

The $R_{70}$ testing procedure essentially involves drying a 150 g sample of <212 μm crushed coal at 110 °C under nitrogen for approximately 16 h. Whilst still under nitrogen, the coal is cooled to 40 °C before being transferred to an adiabatic oven. Once the coal temperature has equilibrated at 40 °C under a nitrogen flow in the adiabatic oven, oxygen is passed through the sample at 50 mL/min. A data logger records the temperature rise due to the self-heating of the coal. The time taken for the coal temperature to reach 70 °C is used to calculate the average self-heating rate for the rise in temperature due to adiabatic oxidation. This is known as the $R_{70}$ index, which is in units of °C/h and is a good indicator of the intrinsic coal reactivity towards oxygen.

The major changes from the normal $R_{70}$ method for moist coal testing are, testing the coal with its as-received moisture content from the ambient mine start temperature, an increased sample size of approximately 200 g and a decreased oxygen flow rate of 10 mL/min. Increasing the sample size to 200 g provides a greater mass of coal to react that is still manageable without modifying the reaction vessel. Decreasing the oxygen flow rate to 10 mL/min reduces any cooling effect experienced by the coal from moisture evaporation as it self-heats. Effectively, these changes optimise the worst case scenario of developing a heating from as-mined coal.
RESULTS AND DISCUSSIONS

R70 self-heating rate values

The R70 self-heating curves for each sample are shown in Figure 1. Their respective R70 values are contained in Table 1. It can be seen that Bowen Basin samples have a very high intrinsic spontaneous combustion reactivity rating that is slightly higher than Spring Creek. These values and rating are generally consistent with the rank and coal type of the Bowen Basin coal.

Figure 1 - Adiabatic self-heating curves for samples tested using the normal R70 test procedure, showing intrinsic spontaneous combustion reactivity ratings based on Queensland conditions (H = High, VH = Very High, UH = Ultra High, EH = Extremely High)

Sample BBHVB07 has a significantly higher sulphur content compared to the other samples (Table 1). Analysis of the forms of sulphur for this sample shows that 42% is due to the presence of pyrite, 54% is organic sulphur and 4% is sulphate sulphur. This equates to less than 2% pyrite present in the sample as tested. A visual inspection of the sample revealed traces of fine pyrite stringers spread throughout the core. The R70 values of the cores do not appear to make any clear distinction in the self-heating performance of this sample compared to the others. In fact it has the lowest R70 value, presumably due to its higher mineral matter content (ash content). Given the long held view that presence of pyrite increases spontaneous combustion propensity, this result seems to be contradictory. However, it must be remembered that the key exothermic pyrite reaction takes place with oxygen in the presence of moisture. As the R70 test is performed on a dry basis this additional reaction does not take place and hence the additional exothermic reaction that may be attributable to the presence of pyrite is not measured by the test.

Moist coal performance and benchmark comparison

Results of tests using the new moist coal adiabatic method are shown in Figure 2. The results for the typical coal cores from the deposit (BBHVB01 and BBHVB08) show that the moisture content of the coal significantly inhibits the coal self-heating. To emphasise this, sample BBHVB10 had 5% of its moisture content removed and when tested under the same start temperature conditions showed a rapid progression to thermal runaway. At this level of moisture, no temperature plateau was recorded. Therefore the moisture state of the coal from the mine face to the stockpile will be an important parameter in determining the overall spontaneous combustion risk of the coal. A site investigation of this effect is underway.

The most important result of this study is the self-heating behaviour of sample BBHVB07. This result is the fastest of any sample tested to date. It reaches thermal runaway in half the time taken by the low rank Kideco coal and indicates the possible risk of the coal creating a spontaneous combustion event in approximately five days based on the benchmark comparison (Figure 2). The accelerated self-heating of this sample compared to all of the others can only be attributable to the presence of the pyrite in the coal reacting with the oxygen and moisture. It is quite a dramatic effect and one that has not been documented so clearly in experimental work before. To prove that the test was not just an artefact, a repeat sample was tested and the results are shown in Figure 3. This clearly shows that the
experimental procedure is very repeatable and that differences between samples can be identified with confidence.

**Figure 2** - Moist coal adiabatic self-heating curves for high volatile bituminous coal samples from the Bowen Basin compared with benchmark coals (Note: the case history typical minimum number of days to reach thermal runaway for each of the benchmark coals is shown)

**Figure 3** - Repeat moist coal adiabatic testing of a pyrite-rich coal ply from a Bowen Basin coal deposit

The identification of this reactive layer in the deposit is a key feature that can be used for spontaneous combustion hazard mapping purposes. Further detailed geotechnical investigation is underway to assess both the vertical and lateral extent of the influence this may have on mine planning and stockpile management of the coal. This work will commence in early 2011 as the geological model of the deposit is refined and further testing is warranted to identify the extent of the pyrite effect on the coal self-heating performance.

**CONCLUSIONS**

The new moist coal adiabatic testing method is proving invaluable to the assessment of coal spontaneous combustion for hazard management planning. Coals of similar intrinsic reactivity show quite dissimilar behaviours in terms of time taken to reach thermal runaway from low ambient temperatures due to the moderating influence of moisture present in the coal. This effect has been documented in earlier moist coal testing on low rank coals and it is clearly shown for a high volatile bituminous coal from the Bowen Basin. It is therefore quite important to understand and document any moisture changes in coal from the mining face to the stockpile environment.
Pyrite has often been cited as an accelerant in the coal self-heating process. This effect is dramatically illustrated by the moist coal adiabatic oven test as the pyrite reaction involving oxygen and moisture is measured. Tests on borehole cores from a Bowen Basin coal deposit have identified a particular seam horizon where the pyrite accelerated coal self-heating is prevalent. This feature of the coal deposit is under further investigation and will be incorporated into the hazard management planning of the operation.

ACKNOWLEDGEMENTS

The authors would like to thank the Australian Coal Industry for their continued support of spontaneous combustion benchmarking and UniQuest Pty Limited for granting permission to publish this paper.

REFERENCES


DEVELOPMENT OF A WEB-BASED UNDERGROUND COAL MINING INFORMATION MANAGEMENT SYSTEM

Ian Porter, Ernest Baafi and R Stace

Abstract: The development of the Australian Coal Association Research Program (ACARP) funded “one stop shop” information management system for the coal mining industry is described. The site www.undergroundcoal.com.au went live on 28 August 2009 with various sub-modules including an introduction to underground coal mining practices and a handbook of roadway development practice. The site also includes presentations from ACARP six-monthly workshops on current roadway development practice. These workshops provide an invaluable opportunity to share state of the art knowledge in roadway development practice, to learn of emerging Research and Development (Rand D) programs and development of equipment and technology by Original Equipment Manufacturers (OEMs).

INTRODUCTION

A review of current roadway development practice (Gibson, 2005) found that there was little information transfer across underground coal mines, even for mines within the same company, and that mines were unaware of developments in roadway practice, equipment and technology. The review recommended that the coal industry develop “a body of knowledge” as a means of capturing current roadway development practice and sharing it across the industry. This provided the impetus for the establishment of a professional website with the domain name www.undergroundcoal.com.au. The website was developed and is maintained by the Faculty of Engineering’s Mining Unit in collaboration with the Centre for Academic Systems and Resources (CASR) at the University of Wollongong (UoW). Located on a single domain website, this visually interactive learning environment provides appropriate material for undergraduate mining engineering students through to highly experienced practicing mining engineers. The three basic elements of the website are:

(i) Information delivery

- Links to OEMs, Government organisations and mine sites;
- Video/audio content of presentations from experts;
- Knowledge of coal mining through practicing engineers;
- Industry calendar of events.

(ii) Web – Interaction (Internet forum)

- This is a message board feature where users can engage in an online discussion in the form of posted messages.

(iii) Research support

- Conference and journal articles;
- Proceedings of industry workshops such as Roadway Development Operators’ Workshops. These workshops had been conducted six monthly since 2006 and provide an invaluable opportunity to share knowledge and improvements in roadway development practice, to learn of emerging R&D programs and development of equipment and technology by OEMs, and to network across the industry.

DOMAIN ARCHITECTURE

The domain’s homepage and a simplified sitemap are shown in Figures 1 and 2, respectively.
Main/Tier 1 navigation

The following six links are available from the Main or Tier 1 Navigation section of the homepage:

1) Calendar - Keeps the dates, venues of future coal events including conferences and workshops.


3) Links - Hyperlink to a database of contact details of coal mining consultants and other service providers.

4) Benchmark Database - In responding to why some longwall roadway development rates are different among mines, ACARP’s Roadway Development Task Group (RDTG) initiated a benchmarking study to survey each longwall mine to determine the correlation between roadway development practices and roadway development performances (Baafi and Gibson, 2010). This sub-module of the site is a web-based relational database system which was developed purposely to survey Australian longwall roadway development performance. Participating longwall
operators are invited annually to supply information online via this sub-module on the following topics:

(i) Mine parameters;
(ii) Development parameters;
(iii) Gas and ventilation;
(iv) Shifts and personnel;
(v) Management of development performance, and
(vi) Development performance.

5) Roadway Workshops - This is a hyperlink to the presentations of the Roadway Development Operators’ Workshops which were initiated in September 2006 as part of ACARP’s overall roadway development improvement project. The primary objective of the workshops is to provide all personnel involved in roadway development a forum to:

(i) learn of emerging best practice and roadway development initiatives;
(ii) learn of developments in equipment and technology;
(iii) network with peers and share their experience and learnings, and
(iv) identify areas for targeted research.

The workshops are held in the three major coal mining regions each six months, Pokolbin (Newcastle/Hunter Valley region NSW), Penrith (Southern/Western NSW) and Mackay (Central Queensland), so that mine based roadway development personnel have the opportunity to attend a workshop in their region without losing time to attend an interstate venue and incurring the associated travel and accommodation expenses.


Local navigation

The following four sub-modules are also accessible from the Local Navigation of the site:

1. **Fundamentals** - the www.undergroundcoal.com.au website is to be of use to all levels of expertise from people with no or, at most, a very limited knowledge of mining through to practicing mining engineers seeking specific information on some aspect. The sub-module is not intended to re-write information which is already available in the public domain but to provide references and where possible web links so that the website can be used as a rapid means of sourcing required information. As a result, most of the text provided in this section of the website covers very basic descriptions of aspects of underground coal mining while the more technical details are available through the links or references. The following topics are presented in this sub-module:

   (i) Basic mining terminology;
   (ii) Exploration;
   (iii) Access to seam from surface;
   (iv) Mine development;
   (v) Pillar extraction using continuous miners;
   (vi) Longwall mining;
   (vii) Ventilation;
   (viii) Gas drainage/Outburst;
   (ix) Spontaneous combustion;
   (x) Coal haulage;
   (xi) Personnel and material transport;
   (xii) Subsidence;
   (xiii) Gas utilisation;
   (xiv) Strata control, and
   (xv) Mine services.

2. **Longwall Mining** - This section discusses the elements of longwall mining methods, including mine planning, equipment selection, automation and cutting methods as well as how to avoid and correct practical longwall related problems.

3. **Roadway Development** - This sub-module is devoted to roadway development practices. The following topics are presented:
(i) Improving roadway development performances;
(ii) Roadway development research updates;
(iii) Roadway development benchmark database, and
(iv) Presentations of ACARP Roadway Development Operators' Workshops.

4. Outburst - This sub-module of the website is a modified version of the original outburst website of ACARP Project C14015 (Aziz, et al., 2007). The original website has been reviewed and modified with the content now being within www.undergroundcoal.com.au. All the figures have been enhanced by professional graphic designers and incorporated into the website using javascript library enhanced modal dialog windows. The site uses "prettyPhoto" a jQuery driven lightbox clone (jQuery, 2010). This results in thumbnail images of the figures being enlarged in a stylish, cross-browser compatible, animated popup display when clicked by the user. By using such a plugin for the jQuery javascript library a broad, robust and maintainable range of browser compatibility is ensured. The original outburst site can still be accessed at http://www.uow.edu.au/eng/outburst/, and is also available via http://research.uow.edu.au/coal.

THE DOMAIN HARDWARE AND SECURITY

The server

This domain is hosted on a dedicated web server installed in the University of Wollongong Information Technology Services (ITS) data centre. The details of the domain’s hardware are:

- Dell(TM) PowerEdge (TM) R710 Rack Mount Server - 3.5-Inch (8.89 cm) Chassis;
- Processors: 2 X Intel(R) Quad Core E5506 Xeon(R) CPU, 2.13GHz, 4M Cache, 4.86 GT/s QPI;
- Memory: 4GB Memory (2x2GB), 1333MHz Dual Ranked RDIM;
- OS drive 146GB - 2 x 146GB SAS - Raid 1;
- Data drive 1.5TB - 4 x 500GB SATA - Raid 5.

This drive configuration allows the operating system to be on one drive and data on the other. Both logical drives are made up of hot swappable drives allowing continuous server operation and data protection in case of a single physical drive failure.

The software used to develop and maintain the site are:

- Windows Server 2008 Enterprise;
- Internet Information Services (IIS) version 7;
- Net Framework versions 2 and 3.0;

The search engine

A mini search engine has been incorporated into the website to allow a rapid and easy search with the inclusion of Google search functionality at a local level. This is achieved by incorporating into the site Google Mini (Google, 2010), an integrated hardware and software solution designed to make the most of digital resources. The Mini achieves this by delivering the power and productivity of Google by searching across all documents within the website quickly and easily. The Mini includes support for multiple document collections, search across file servers and reporting functions. It works with over 220 different file formats and is accessed by a search box on www.undergroundcoal.com.au.

Security access

The server of www.undergroundcoal.com.au is hosted in the University of Wollongong Information Technology Services (ITS) machine room and physical access to the server is restricted to authorised ITS personnel. UoW information technology infrastructure incorporates the provision of computer, network and communication services to the campus community for academic teaching and research and for general administrative functions. The server conforms to the UoW server security process and guidelines. This implies the website is behind the UoW firewall and public access is only allowed to the web content through the UoW hypertext transfer protocol (http) through port 80. Windows file sharing is, however, allowed from a number of nominated machines in UoW CASR for content update and management. Location of the machine within ITS allows automatic backup and recovery of all site content in the event of a system or hardware malfunction.
CONCLUSIONS

The ACARP funded website www.undergroundcoal.com.au provides easy access to current information on underground coal mining, especially roadway development technology and systems for Australian longwall operators, OEMs, researchers and developers. The site provides a ‘one stop shop’ for information on Australian coal mining practices, and has become an invaluable tool - for mining engineering students and practitioners worldwide. The website www.undergroundcoal.com.au also serves as devolution of outcomes from ACARP’s RDTG. Its current content includes:

- Fundamentals of coal mining practice sub-module;
- Outburst module;
- Online forum sub-module to engage web-based discussions in real time;
- Roadway development benchmarking online survey;
- Handbook of Roadway Development Practice – Improving Roadway Development sub-module;
- A directory of active consultants serving the industry;
- Presentations of Roadway Development Operators’ Workshops since 2006;
- Calendar of future events;
- Research updates and useful information, and
- Industry links.

An editorial panel comprising experienced mining professionals and UoW mining academics has been charged with the responsibility for not only developing the basic structure of the site but also reviewing and updating the content.

ACKNOWLEDGEMENTS

The authors would like to thank Robert Newman for his contributions, especially the contents of the sub-modules Coal Mining Fundamentals, Longwall Mining and Gas Drainage. The support from members of ACARP’s RDTG, especially Gary Gibson, is also acknowledged for peer reviewing the site. The authors would like to thank OEMs, especially Joy Mining Machinery and Sandvik Mining and Construction for making various images of mining equipment available for the website. The authors acknowledge the developers of the original outburst website of ACARP Project C14015 http://www.uow.edu.au/eng/outburst/.

REFERENCES


IMPROVING ACCESS TO UNDERGROUND COAL OPERATORS’ CONFERENCE PAPERS

Michael Organ, Naj Aziz and Jan Nemcik

ABSTRACT: The annual Underground Coal Operators’ Conference (UCOC) has been held at the University of Wollongong since 1998. In February 2010 it celebrated its 10th anniversary. During that period a total of 332 conference papers were published within the ten printed volumes of proceedings. From December 2008 digital copies were made available online via the UCOC website (http://ro.uow.edu.au/coal) within the University of Wollongong’s open access repository Research Online (http://ro.uow.edu.au). The UCOC website is linked to other relevant sites available to mining engineering researchers at Wollongong and all papers are easily located through an internal search engine, using author, title and keyword searches. Download of individual conference papers is free of charge and usage statistics are monitored by the conference conveners. Since going live, visitors to the site from more than 130 countries have downloaded over 85 000 individual conference papers. Online availability has greatly improved access for students, researchers and mine professionals world-wide. It has also enhanced the reputation of the University of Wollongong as the centre of excellence in teaching and research in mining engineering, with a keen interest in promoting mining technology nationally and internationally. This paper discusses the national and international impact of free online access to the UCOC papers.

INTRODUCTION

The annual Underground Coal Operators’ Conference (UCOC) has been held at the University of Wollongong since 1998. Jointly organised by the Mining Engineering Group of the School of Civil Mining and Environmental Engineering, University of Wollongong, the Illawarra Branch of The Australasian Mining and Metallurgy (AusIMM) and the Mine Managers Association of Australia, it is an international conference drawing practitioners in the field from all corners of the globe. Printed volumes of proceedings have seen limited circulation via dispersal to conference attendees and placement in various library collections.

During the second half of 2008 the opportunity to improve access to this significant collection of research material by making it freely available online was presented to the conference conveners by the University Library at the University of Wollongong. A new conference module within Research Online (http://ro.uow.edu.au) – the University of Wollongong’s open access repository – was being rolled out and the Manager Repository Services sought content from past and future conferences which could be digitally archived. Following campus-wide promotion of this new feature, he was approached by Professor Naj Aziz, convenor of UCOC, who was interested in the possibilities offered by providing online access to the substantial collection of UCOC papers. Digital files were subsequently secured and from December 2008 through to March 2010 a collection of 332 papers were uploaded to a new UCOC website (http://ro.uow.edu.au/coal) located within Research Online.

The previous UCOC websites – comprising the University of Wollongong Coal Mining Science and Technology website (http://research.uow.edu.au/coal/) and a dedicated conference information site (http://www.coalconference.net.au/) - contained a wealth of information on the conference and related topics, however they did not provide access to copies of the conference papers. The present UCOC section of Research Online remedies this omission and also maintains linkage to up-to-date mining websites at the Faculty of Engineering websites through a ‘Conference website’ link. These sites cover topics such as longwall mining (http://www.uow.edu.au/eng/longwall/), bord and pillar mining (http://www.uow.edu.au/eng/pillar/) and coal and gas outburst (http://www.uow.edu.au/eng/outburst/), and provide an important adjunct to the archived conference papers and current conference websites.

UCOC on Research Online is, as far as the authors are aware, the only coal mining website of its type that is freely available for the use of students, engineers and professionals in the field. Since its creation at the end of 2008, UCOC papers have been accessed more than 100 000 times via internet search
engines such as Google or direct links. This represents a major increase on the perhaps hundreds of researchers who would have had access to this material through its printed form. A more detailed assessment of the success of the UCOC website in exposing conference papers to global researchers is outlined below.

RESEARCH ONLINE

Research Online ([http://ro.uow.edu.au](http://ro.uow.edu.au)) is an open access digital archive promoting the scholarly output of the University of Wollongong. It makes use of the Bepress Digital Commons software platform ([http://www.bepress.com/ir/](http://www.bepress.com/ir/)). Research Online allows visitors to browse research material by:

- Faculty or administrative unit;
- Series and journals;
- Conferences;
- Thesis;
- Authors;
- Image Galleries;
- Document Type.

The UCOC website is a series within Research Online and is accessed from the home page under the Faculty of Engineering heading within the Faculty of administrative unit link. The site arranges the material chronologically for ease of browsing and contains a search engine which allows identification of individual conference papers by author, title or keyword.

Research Online facilitates open access to digital copies of research publications. It does this by using various internet protocols and programming tools to ensure that material placed on Research Online is visible to search engines such as Google. Searches on topics of relevance to the UCOC will return results which are clear and concise, thereby encouraging researchers to visit the site, assess the content via the information contained on the metadata pages, and download a digital copy of the conference paper itself. Every UCOC paper on Research Online contains an individual descriptive page, whilst the downloaded documents themselves include a system-generated cover page, which provides detailed citation information in the following form:


Research Online makes available a substantial amount of bibliographic and descriptive information to search engines such as Google. Not only are author, title and publication details visible, but also abstracts, and in many cases – where optical character recognition (OCR) has been applied to the pdfs - the full text of the paper itself is searched by Google. With such a wealth of data available, the chances of a researcher in the field coming across a UCOC paper are substantially increased. Evidence for this is found in the number of people accessing the site, by how many conference paper pdfs are downloaded, and by subsequent citation rates.

Research Online also contain a statistics package which can be used in conjunction with Google Analytics to gauge the level of access to various sections of the site, including individual UCOC papers. These statistics can be access via the local system administrator and conference conveners. They provide basic information on “hits” to the site (i.e. the number of times a link to the metadata page or pdf file is opened), the number of full text downloads, and the various countries and domains from which site access is sought. A detailed statistical analysis of use of the UCOC site to date is presented below.

STATISTICAL SURVEY OF ACCESS TO THE UCOC WEBSITE

Access and downloads

According to statistics generated by the Research Online system, as of 1 January 2011 the 332 conference papers on the UCOC website had been accessed, or ‘hit’, over 100,000 times. This included both browsing of the individual metadata or descriptive pages for each conference paper, plus full text
downloads of pdf copies of the papers themselves. All items received at least 50 hits, with 80% receiving more than 100 hits and 11% more than 500 hits. The five most popular papers had each been accessed over 1 000 times. These figures reveal the high visibility of material placed on Research Online and made accessible via the internet.

Perhaps of most significance is the fact that there have been more than 85 000 full text downloads of conference papers to the end of 2010. This indicates that individual researchers and professionals in the field have actually made the effort to download the full paper and, it can be assumed, read it, rather than merely browsed the citation and abstract. Such downloads can lead to increased citation rates for individual authors, further referrals to the UCOC site and the opening of communication between individual authors and researchers. The latter can of course lead to future collaboration. By increasing online visibility of the conference papers, and enabling easy access to digital copies, Research Online is supporting the aims of the conference conveners in dispersing research findings relating to the area of coal mining operations. It is clear that Research Online has greatly increased the impact of the conference, which was previously limited only to those able to attend in person, or perhaps access printed copies of the proceedings.

Paper popularity

Access and full text download statistics for UCOC papers reflect both the time papers are available online and individual popularity. It is obvious that the longer an item is on open access, then the greater the number of hits and downloads it receives. However this alone cannot account for the number or variation in such statistics. Popularity is an overriding factor, though it is a difficult term to define and may be affected by a number of factors, including content of the conference paper, title, abstract information and authorship. For example, two papers with non-descriptive titles - ‘Improving Relations’ and ‘Contemporary developments in training’ - figure at number 4 and 14 (viz. 71 and 46 hits) in the list of least accessed papers (Table 1), though other papers with more specific and relevant titles (e.g. ‘Mining gas initiative North Rhine-Westphalia’, 68 hits and ranked number 3) also figure low on the list. It is therefore difficult to ascertain the effect of a descriptive title on popularity. A concise abstract may be a more significant factor than the title.

Table 1 - Paper accessed ranking between 1st December 2008 and 31st December 2010

(a) Ten most accessed papers

<table>
<thead>
<tr>
<th>No.</th>
<th>Title</th>
<th>URL</th>
<th>Conference Year</th>
<th>Total Hits</th>
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<tr>
<td>5</td>
<td>Gate Road Development in High Gas Content Coal Seams at Karaganda Basin Coal Mines, Kazakhstan</td>
<td><a href="http://ro.uow.edu.au/coal/85">http://ro.uow.edu.au/coal/85</a></td>
<td>2009</td>
<td>1097</td>
</tr>
<tr>
<td>6</td>
<td>Advances in Surface Seismic Acquisition, Processing and Interpretation</td>
<td><a href="http://ro.uow.edu.au/coal/41">http://ro.uow.edu.au/coal/41</a></td>
<td>2008</td>
<td>1077</td>
</tr>
</tbody>
</table>
In regards to length of online availability as a factor, the most popular paper – ‘Pike River Coal - Hydraulic Mine Design on New Zealand’s West Coast’ (http://ro.uow.edu.au/coal/43) with 2 623 hits, was uploaded on 14 December 2008, as was ‘The Use of Downhole Presometers Implications for Modern Underground Mines’ (http://ro.uow.edu.au/coal/42) with only 210 hits. Once again, length of time available on open access was not the major factor in determining the level of access and popularity of an item.

With most of the papers only being online for 12-18 months it is too early to make statements at this stage in regards to general trends re access and download rates. Table 1 show the first ten most accessed and the last ten least accessed papers out of the total 332 papers, covering the period December 2008 to 31 December 2010. The appendix lists full text downloads between December 1<sup>st</sup>, 2008 and December 31<sup>st</sup>, 2010. It is clear from these and the more fulsome set of associated statistics that:

- There is a small correlation between the length of time available online and the number of times an item is accessed or downloaded.
- Popularity of papers can be measured by the number of times they are accessed and downloaded.
- Reasons for the popularity of any individual paper are unclear. It may be related to content (e.g. title, abstract, full text), relevancy to particular trends in the sector, identification as readings in academic teaching and research programs, or interest in the work of particular authors.
- The papers from any one conference are no more popular or heavily accessed than those from any other conference, though there is a slight preference for the most popular papers to be associated with post 2004 conferences, with the top 20 papers so identified. No reason for this has been identified.
- The least accessed papers are from 2010, though this is a short term factor related to limited online availability.

### Access by subject topic

The categorisation of the papers online is in accordance with the topics that were listed in the call for papers at the home page of the conference website (http://www.coalconference.net.au/). In general, most of the published papers are related to underground operations; however there are also a few

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<tr>
<th>No.</th>
<th>Title</th>
<th>URL</th>
<th>Conference Year</th>
<th>Total Hits</th>
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</thead>
<tbody>
<tr>
<td>6</td>
<td>The nature of underground heating as indicated by numerical modelling</td>
<td><a href="http://ro.uow.edu.au/coal/239">http://ro.uow.edu.au/coal/239</a></td>
<td>1998</td>
<td>79</td>
</tr>
<tr>
<td>7</td>
<td>Gas Emission Modelling of Gate Road Development</td>
<td><a href="http://ro.uow.edu.au/coal/216">http://ro.uow.edu.au/coal/216</a></td>
<td>2009</td>
<td>79</td>
</tr>
<tr>
<td>8</td>
<td>Improving colliery performance through one big team, or many teams</td>
<td><a href="http://ro.uow.edu.au/coal/284">http://ro.uow.edu.au/coal/284</a></td>
<td>2009</td>
<td>79</td>
</tr>
</tbody>
</table>
papers on surface mining as the conference is fundamentally intended for underground mining operation. Topics covered in the various aspects of mining include:

1) heading development;
2) mining methods; longwall, bord and pillar, top coal caving and hydraulic mining: mine geology;
3) equipment and machinery performances;
4) ground control and strata management;
5) Instrumentation;
6) rock bolting;
7) mine subsidence;
8) mine ventilation, gas drainage and outburst control;
9) mine fires and control, mine dust and control;
10) risk management;
11) mine management and mine contract.

Table 2 and Figure 1 show the percentages of access for papers download by subject category between 2008 and 2010, based on data extracted from Research Online.

<table>
<thead>
<tr>
<th>Subject topics</th>
<th>Access (%)</th>
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<tr>
<td>2. Mining methods and geology</td>
<td>25.0</td>
</tr>
<tr>
<td>3. Equipment and machinery performances</td>
<td>4.6</td>
</tr>
<tr>
<td>4. Ground control and strata management</td>
<td>10.6</td>
</tr>
<tr>
<td>5. Instrumentation</td>
<td>3.4</td>
</tr>
<tr>
<td>6. Rock bolting</td>
<td>5.74</td>
</tr>
<tr>
<td>7. Mine subsidence</td>
<td>4.65</td>
</tr>
<tr>
<td>8. Mine ventilation gas drainage and outburst control</td>
<td>19.53</td>
</tr>
<tr>
<td>9. Mine fires and control, mine dust and control</td>
<td>8.2</td>
</tr>
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<td>10. Risk management</td>
<td>7.7</td>
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<tr>
<td>11. Mine management and mine contract</td>
<td>10.6</td>
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</table>

Figure 1 - Number of paper downloads per subject topic between December 2008 and August 2010

It is clear that the highest proportion of hits were in the mining methods category at 25%, followed by the mine gases and outburst control category at 19.5%. The lowest percentage was on equipment and machinery, which was expected as many papers on mining methods had sections on equipment and machinery and other topics like ground control. Thus the percentage distribution by subject cannot, to all intents and purposes, be accurately categorised.

Some subject topics had greater coverage with greater numbers of papers in the conference proceedings in comparison to others with a lower percentage of papers. Also, the relatively higher proportion of the mining methods section was attributed to the fact that this section contained papers on geology, strata control and sometimes on equipment utilisation, which could also be included in the equipment and machinery section and ground control. Similarly the section on mine ventilation, mine gas and outburst control, with the second high access rate at 19.25%, was due to the high numbers of papers on these topics.
Access by country

Access to the UCOC website from both national and international browsers can be ascertained from statistical information generated internally within Research Online and also externally by Google Analytics. From these two sources - which provide variations in their statistical returns due to differing counting methodologies - it is clear that between December 2008 and December 2010 the UCOC website was accessed by researchers in more than 130 countries. Table 3, using Research Online data, lists initiation of full text downloads by location over that period, based on analysis of the two digit top level domain (e.g. au, nz). It should be noted that the two digit top level domain does not apply to the United States, except on rare occasions when the domain .us is used. Therefore, in Table 3 a figure has been allocated to those visitations from sites with no two digit top level domain, assuming that the vast majority are from the US, though this is not always the case.

**Table 3 - Frequency of access to Underground Coal Operators’ Conference papers by country December 2008 to October 2010 (Source: Research Online)**

<table>
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<tr>
<th>Country</th>
<th>Frequency</th>
<th>Country</th>
<th>Frequency</th>
<th>Country</th>
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<td>Nigeria</td>
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<td>United States</td>
<td>30966</td>
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It is clear that the number of visits from any country was dependent on the status of the mining industry in that particular country. In general the access population was higher from recognised coal mining countries such as the United States, India, Germany, Poland, Russia and Turkey, as well as Australia, which commands the highest rate of access. Countries with lower access rates included Armenia, Cuba, and Uruguay where there is little interest in coal mining. Figure 2 uses Google Analytics data for the UCOC website covering the period January 2009 to June 2010. It identifies a total of 4,329 visits to the UCOC website from 56 different countries, with the top 10 from those with significant coal mining industries.

![Figure 2 - Visits to the Underground Coal Operators' Conference website by country January 2009 to June 2010 (Source: Google Analytics)](image)

As expected, Australia was the highest accessing country with more than 61.24%, followed by the United States and China at 7.11% and 5.82% respectively. At the bottom end of the scale, and not shown in the figure, are countries such as Armenia and the Virgin Islands, with a single visit. These figures, whilst varying somewhat from those generated by Research Online, nevertheless reveal similar access trends. Google identifies 4,329 visits to the UCOC website homepage, whilst Research Online lists 88,000+ hits to the site as a whole, which includes all those pages below the level of the homepage and thereby accounts for the discrepancy.

**SUMMARY**

The online access to the Coal Operators’ Conference website is a useful exercise for the benefit of the mining education, the mining industry and research organisations. It provided an opportunity for access from remote locations on a variety of coal mining issues as the conference is a multi-topic event covering a variety of topics of vital importance to the industry operation.

Placement of the Coal Operators’ Conference papers online access has enhanced the reputation of the University of Wollongong as the centre of excellence in teaching and research in mining engineering with keen interest in promoting mining technology nationally and internationally. Equally benefited are both The Illawarra Branch of the Aus IMM and Mine Managers Association of Australia.
Appendix - Full-text downloads of Underground Coal Operators’ Conference papers between 1-12-2008 and 31-12-2010

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<td>Crinum Mine, 15 Longwalls 40 Million Tonnes 45 Roof Fails - What did we Learn?</td>
<td><a href="http://ro.uow.edu.au/coal/2">http://ro.uow.edu.au/coal/2</a></td>
<td>2008/1/12</td>
<td>1021</td>
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<td>Advances in Surface Seismic Acquisition, Processing and Interpretation</td>
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<td>Gate Road Development in High Gas Content Coal Seams at Karaganda Basin Coal Mines, Kazakhstan</td>
<td><a href="http://ro.uow.edu.au/coal/65">http://ro.uow.edu.au/coal/65</a></td>
<td>02/15/2009</td>
<td>893</td>
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<td>Geological and Geotechnical Influences on the Caveability and Drawability of Top Coal in Longwalls</td>
<td><a href="http://ro.uow.edu.au/coal/4">http://ro.uow.edu.au/coal/4</a></td>
<td>2008/1/12</td>
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<td>CMRR - Practical Limitations and Solutions</td>
<td><a href="http://ro.uow.edu.au/coal/7">http://ro.uow.edu.au/coal/7</a></td>
<td>2008/1/12</td>
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<td>Polymer-Based Alternative to Steel Mesh for Coal Mine Strata Reinforcement</td>
<td><a href="http://ro.uow.edu.au/coal/9">http://ro.uow.edu.au/coal/9</a></td>
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<td>Technology Knowledge Base for Coal Mining: Websites at the University of Wollongong</td>
<td><a href="http://ro.uow.edu.au/coal/30">http://ro.uow.edu.au/coal/30</a></td>
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<td>A Case Study on Longwall Mining under the Tidal Waters of Lake Macquarie</td>
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<td>Exploration for Results: Moura Coal Mine</td>
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<td>Gas Content Estimation Using Initial Desorption Rate</td>
<td><a href="http://ro.uow.edu.au/coal/100">http://ro.uow.edu.au/coal/100</a></td>
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<td>Coming of Age for Low-Density Explosives</td>
<td><a href="http://ro.uow.edu.au/coal/126">http://ro.uow.edu.au/coal/126</a></td>
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<td>'Keep the Cream'-- Reconciling Coal Recovery at BMA Goonyella Riverside</td>
<td><a href="http://ro.uow.edu.au/coal/121">http://ro.uow.edu.au/coal/121</a></td>
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<td>Longwall Roof Control by Calculation of the Shield Support Requirements</td>
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<td>Longwall Website for Australian Mining Conditions</td>
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<td>Innovative CFD Modelling Ling to Improve Dust Control in Longwalls</td>
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<td>Geotechnical Evaluation of Roof Conditions at Crinum Mine Based on Geophysical Log Interpretation</td>
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<td>Introduction of continuous haulage (4FCT) at the Clarence Colliery</td>
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<td>The Limitations of the Observational Method for Roof Support and Subsidence Management in High Production Longwalls</td>
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<td>Spontaneous Combustion in Open Cut Coal Mines -- Recent Australian Research</td>
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<td>Sorption Characteristic of Coal, Particle Size, Gas Type and Time</td>
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<td>Determining the Controls for Strata Gas and Oil Distribution within Sandstone Reservoirs Overlying the Bulli Seam</td>
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<td>Roof Support in Gateroads in Multiple Seam Longwalls - Lessons From Ultraclose Mining at Gibson's Colliery</td>
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<td>Managing Roof Control Problems on a Longwall Face</td>
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<td>New Developments in Australian Coal Production</td>
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<td>CFD Modelling of Goaf Gas Migration to Improve the Control of Spontaneous Combustion in Longwalls</td>
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<td>Geotechnical Engineering at German Creek - A Historical and Sometimes Hysterical Review</td>
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<td>Low Temperature Oxidation of a High Volatile Bituminous Turkish Coal Effects of Temperature and Particle Size</td>
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<td>A safe way to reduce roof support costs and improve safety and productivity</td>
<td><a href="http://ro.uow.edu.au/coal/290">http://ro.uow.edu.au/coal/290</a></td>
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<td>Shear testing of 28mm hollow strand &quot;TG&quot; cable bolt</td>
<td><a href="http://ro.uow.edu.au/coal/303">http://ro.uow.edu.au/coal/303</a></td>
<td>03/16/2010</td>
<td>173</td>
</tr>
<tr>
<td>Non destructive integrity testing of rock reinforcement elements in Australian mines</td>
<td><a href="http://ro.uow.edu.au/coal/304">http://ro.uow.edu.au/coal/304</a></td>
<td>03/16/2010</td>
<td>168</td>
</tr>
<tr>
<td>The Use of Downhole Presometers Implications for Modern Underground Mines</td>
<td><a href="http://ro.uow.edu.au/coal/42">http://ro.uow.edu.au/coal/42</a></td>
<td>12/14/2008</td>
<td>165</td>
</tr>
<tr>
<td>3D geotechnical models for coal and clastic rocks based on the GSR</td>
<td><a href="http://ro.uow.edu.au/coal/301">http://ro.uow.edu.au/coal/301</a></td>
<td>03/16/2010</td>
<td>163</td>
</tr>
</tbody>
</table>

Continue......
<table>
<thead>
<tr>
<th>Title</th>
<th>URL</th>
<th>First published</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Applications of RFID and mobile technology in tracking of equipment for maintenance in the mining industry</td>
<td><a href="http://ro.uow.edu.au/coal/326">Link</a></td>
<td>03/16/2010</td>
<td>156</td>
</tr>
<tr>
<td>AMCMRR - an analytical model for coal mine roof reinforcement</td>
<td><a href="http://ro.uow.edu.au/coal/302">Link</a></td>
<td>03/16/2010</td>
<td>156</td>
</tr>
<tr>
<td>The Use of Sonic Velocity Logs to Define Potential Goaf Delamination Horizons</td>
<td><a href="http://ro.uow.edu.au/coal/158">Link</a></td>
<td>02/16/2009</td>
<td>156</td>
</tr>
<tr>
<td>The Effect of Resin Thickness on Bolt-Grout-Concrete Interaction in Shear</td>
<td><a href="http://ro.uow.edu.au/coal/64">Link</a></td>
<td>02/15/2009</td>
<td>153</td>
</tr>
<tr>
<td>Observations on the Variation in Acoustic Emissions with Changes in Rock Cutting Conditions</td>
<td><a href="http://ro.uow.edu.au/coal/38">Link</a></td>
<td>2008/11/12</td>
<td>152</td>
</tr>
<tr>
<td>Using Helium as a Tracer Gas to Measure Vertical Overburden Conductivity Above Extraction Panels</td>
<td><a href="http://ro.uow.edu.au/coal/95">Link</a></td>
<td>02/15/2009</td>
<td>152</td>
</tr>
<tr>
<td>Inseam drilling for gas exploration - recent advances</td>
<td><a href="http://ro.uow.edu.au/coal/251">Link</a></td>
<td>2009/8/3</td>
<td>150</td>
</tr>
<tr>
<td>Roadway roof support design in critical areas at Anglo American Metallurgical Coal’s underground operations</td>
<td><a href="http://ro.uow.edu.au/coal/297">Link</a></td>
<td>03/16/2010</td>
<td>149</td>
</tr>
<tr>
<td>New Regulation for Health, Safety and Subsidence</td>
<td><a href="http://ro.uow.edu.au/coal/155">Link</a></td>
<td>02/16/2009</td>
<td>144</td>
</tr>
<tr>
<td>Advanced numerical modelling methods of rock bolt performance in underground mines</td>
<td><a href="http://ro.uow.edu.au/coal/325">Link</a></td>
<td>03/16/2010</td>
<td>141</td>
</tr>
<tr>
<td>An integrated mine development and supply system</td>
<td><a href="http://ro.uow.edu.au/coal/279">Link</a></td>
<td>2009/9/3</td>
<td>137</td>
</tr>
<tr>
<td>Industry Impacts Outburst and Gas Management</td>
<td><a href="http://ro.uow.edu.au/coal/189">Link</a></td>
<td>02/17/2009</td>
<td>137</td>
</tr>
<tr>
<td>A New Approach for Determination of Tunnel Supporting System Using Analytical Hierarchy Process (AHP)</td>
<td><a href="http://ro.uow.edu.au/coal/82">Link</a></td>
<td>02/15/2009</td>
<td>137</td>
</tr>
<tr>
<td>How NSW and Queensland Coalfields Differ - What We Need To Do Better</td>
<td><a href="http://ro.uow.edu.au/coal/214">Link</a></td>
<td>02/19/2009</td>
<td>135</td>
</tr>
<tr>
<td>Strata Management at the Goonyella Exploration Adit Project</td>
<td><a href="http://ro.uow.edu.au/coal/210">Link</a></td>
<td>02/19/2009</td>
<td>133</td>
</tr>
<tr>
<td>Coal mine goaf gas predictor (CMGGP)</td>
<td><a href="http://ro.uow.edu.au/coal/308">Link</a></td>
<td>03/16/2010</td>
<td>133</td>
</tr>
<tr>
<td>Computer Animation of Hot Spot Development in Bulk Coal as an Aid for Training Coal Miners</td>
<td><a href="http://ro.uow.edu.au/coal/154">Link</a></td>
<td>02/16/2009</td>
<td>129</td>
</tr>
<tr>
<td>Modification of four-section cut model for drift blast design in Razi coal mine - North Iran</td>
<td><a href="http://ro.uow.edu.au/coal/320">Link</a></td>
<td>03/16/2010</td>
<td>128</td>
</tr>
</tbody>
</table>

Continue….
<table>
<thead>
<tr>
<th>Title</th>
<th>URL</th>
<th>First published</th>
<th>Total</th>
</tr>
</thead>
</table>

Continue......
<table>
<thead>
<tr>
<th>Title</th>
<th>URL</th>
<th>First published</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>The Role of ACARP in supporting Australian Coal Research</td>
<td><a href="http://ro.uow.edu.au/coal/294">http://ro.uow.edu.au/coal/294</a></td>
<td>03/16/2010</td>
<td>105</td>
</tr>
<tr>
<td>Case study of ethane emissions at Mandalong Mine</td>
<td><a href="http://ro.uow.edu.au/coal/299">http://ro.uow.edu.au/coal/299</a></td>
<td>03/16/2010</td>
<td>104</td>
</tr>
<tr>
<td>Improving emergency management in underground coal mines</td>
<td><a href="http://ro.uow.edu.au/coal/305">http://ro.uow.edu.au/coal/305</a></td>
<td>03/16/2010</td>
<td>104</td>
</tr>
<tr>
<td>An Update of Roof Bolt Research at the University of Wollongong</td>
<td><a href="http://ro.uow.edu.au/coal/147">http://ro.uow.edu.au/coal/147</a></td>
<td>02/16/2009</td>
<td>95</td>
</tr>
<tr>
<td>Managing the geotechnical aspects of longwall face recovery</td>
<td><a href="http://ro.uow.edu.au/coal/321">http://ro.uow.edu.au/coal/321</a></td>
<td>03/16/2010</td>
<td>95</td>
</tr>
<tr>
<td>A New Technique to Determine the Load Transfer Capacity of Resin Anchored Bolts</td>
<td><a href="http://ro.uow.edu.au/coal/208">http://ro.uow.edu.au/coal/208</a></td>
<td>02/19/2009</td>
<td>93</td>
</tr>
<tr>
<td>Case Studies in the Application of Influence Functions to Visualising Surface Subsidence</td>
<td><a href="http://ro.uow.edu.au/coal/164">http://ro.uow.edu.au/coal/164</a></td>
<td>02/16/2009</td>
<td>89</td>
</tr>
<tr>
<td>Benchmarking moist coal adiabatic oven testing</td>
<td><a href="http://ro.uow.edu.au/coal/324">http://ro.uow.edu.au/coal/324</a></td>
<td>03/16/2010</td>
<td>82</td>
</tr>
<tr>
<td>Loading Mechanics of the 'Can' and Implications for Improved Strength and Stiffness Properties</td>
<td><a href="http://ro.uow.edu.au/coal/74">http://ro.uow.edu.au/coal/74</a></td>
<td>02/15/2009</td>
<td>81</td>
</tr>
<tr>
<td>The Challenge to Improve the Prediction of Subsidence Impacts</td>
<td><a href="http://ro.uow.edu.au/coal/163">http://ro.uow.edu.au/coal/163</a></td>
<td>02/16/2009</td>
<td>77</td>
</tr>
</tbody>
</table>

Continue……
<table>
<thead>
<tr>
<th>Title</th>
<th>URL</th>
<th>First published</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>The Influence of Gas Environment on Coal Properties - Experimental Studies on Outburst Control</td>
<td><a href="http://ro.uow.edu.au/coal/143">http://ro.uow.edu.au/coal/143</a></td>
<td>02/16/2009</td>
<td>75</td>
</tr>
<tr>
<td>On Mining-Induced Horizontal Shear Deformations of the Ground Surface</td>
<td><a href="http://ro.uow.edu.au/coal/92">http://ro.uow.edu.au/coal/92</a></td>
<td>02/15/2009</td>
<td>73</td>
</tr>
<tr>
<td>A Study of the Formation of Hydrogen Produced During the Oxidation of Bulk Coal Under Laboratory Conditions</td>
<td><a href="http://ro.uow.edu.au/coal/17">http://ro.uow.edu.au/coal/17</a></td>
<td>2008/2/12</td>
<td>71</td>
</tr>
<tr>
<td>Enterprise bargaining and agreements under the Workplace Relations Act, 1996 and their appreciation to the NSW coal mining industry</td>
<td><a href="http://ro.uow.edu.au/coal/319">http://ro.uow.edu.au/coal/319</a></td>
<td>03/16/2010</td>
<td>70</td>
</tr>
<tr>
<td>Development, application and potential of a real-time return gas monitoring system</td>
<td><a href="http://ro.uow.edu.au/coal/249">http://ro.uow.edu.au/coal/249</a></td>
<td>2009/8/3</td>
<td>64</td>
</tr>
<tr>
<td>The Impacts of Mine Subsidence on Creeks, River Valleys and Gorges Due to Underground Coal Mining Operations</td>
<td><a href="http://ro.uow.edu.au/coal/165">http://ro.uow.edu.au/coal/165</a></td>
<td>02/16/2009</td>
<td>64</td>
</tr>
<tr>
<td>Update on Outbursts and In-Seam Drilling in 2002</td>
<td><a href="http://ro.uow.edu.au/coal/183">http://ro.uow.edu.au/coal/183</a></td>
<td>02/16/2009</td>
<td>54</td>
</tr>
<tr>
<td>Calibrated parameters for the prediction of subsidence at Mandalong Mine</td>
<td><a href="http://ro.uow.edu.au/coal/300">http://ro.uow.edu.au/coal/300</a></td>
<td>03/16/2010</td>
<td>53</td>
</tr>
<tr>
<td>Seismic analysis of horseshoe tunnels under dynamic loads due to earthquakes</td>
<td><a href="http://ro.uow.edu.au/coal/327">http://ro.uow.edu.au/coal/327</a></td>
<td>03/16/2010</td>
<td>52</td>
</tr>
</tbody>
</table>

Continue……
<table>
<thead>
<tr>
<th>Title</th>
<th>URL</th>
<th>First published</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>The Demonstration of Electronic Systems to Assist in the Management of a Significant Incident</td>
<td><a href="http://ro.uow.edu.au/coal/175">http://ro.uow.edu.au/coal/175</a></td>
<td>02/16/2009</td>
<td>50</td>
</tr>
<tr>
<td>Towards an integration information infrastructure in coal mining asset management application</td>
<td><a href="http://ro.uow.edu.au/coal/331">http://ro.uow.edu.au/coal/331</a></td>
<td>03/16/2010</td>
<td>49</td>
</tr>
<tr>
<td>Results of Self-Heating Tests of Australian Coals Conducted in a 16m3 Reactor</td>
<td><a href="http://ro.uow.edu.au/coal/146">http://ro.uow.edu.au/coal/146</a></td>
<td>02/16/2009</td>
<td>47</td>
</tr>
<tr>
<td>Assessment of seismic events in German hard coal mining - occurrence and prediction</td>
<td><a href="http://ro.uow.edu.au/coal/322">http://ro.uow.edu.au/coal/322</a></td>
<td>03/16/2010</td>
<td>44</td>
</tr>
<tr>
<td>The nature of underground heating as indicated by numerical modeling</td>
<td><a href="http://ro.uow.edu.au/coal/239">http://ro.uow.edu.au/coal/239</a></td>
<td>2009/8/3</td>
<td>42</td>
</tr>
<tr>
<td>Developments in self escape and aided rescue arising for the Moura No.2 Wardens Inquiry - A Special Report by the Joint Coal Industry Committee from Queensland and New South Wales</td>
<td><a href="http://ro.uow.edu.au/coal/229">http://ro.uow.edu.au/coal/229</a></td>
<td>2009/8/3</td>
<td>42</td>
</tr>
<tr>
<td>Productivity improvement from economic concept to an engineering tool</td>
<td><a href="http://ro.uow.edu.au/coal/328">http://ro.uow.edu.au/coal/328</a></td>
<td>03/16/2010</td>
<td>41</td>
</tr>
<tr>
<td>Improving colliery performance through one big team, many teams or ...??</td>
<td><a href="http://ro.uow.edu.au/coal/284">http://ro.uow.edu.au/coal/284</a></td>
<td>2009/9/3</td>
<td>36</td>
</tr>
<tr>
<td>Gas Emission Modelling of Gate Road Development</td>
<td><a href="http://ro.uow.edu.au/coal/216">http://ro.uow.edu.au/coal/216</a></td>
<td>02/19/2009</td>
<td>31</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td>85325</td>
</tr>
</tbody>
</table>
INDEX TO AUTHORS

Addinell, S. .......................................................... 197
Alem, L ................................................................. 171
Anderson, J .......................................................... 29
Atkins, A. S .......................................................... 105
Aziz, N ................................................................. 141, 231, 269, 307, 390
Baafi, E ............................................................... 148, 154, 385
Bai, Q ................................................................. 22
Balusu, R ............................................................. 249
Beamish, B ......................................................... 375, 380
Beamish, R .......................................................... 380
Black, D .............................................................. 307
Canbulat, I .......................................................... 369, 375
Cao, C ................................................................. 141
Chen, H ............................................................... 361
Chen, W .............................................................. 129
Cheng, Y .............................................................. 277, 326, 335, 343, 361
Cinar, Y .............................................................. 355
Claassens, C ......................................................... 369, 375
Cooper, G ........................................................... 239
Cram, K ............................................................... 231
Dadkhah, M ........................................................ 99
Dunn, M. T .......................................................... 165
Einicke, G ........................................................... 189
Frith, R ................................................................. 73
Gray, I ................................................................. 16, 291, 297
Hadi, M. N. S ....................................................... 121
Haight, T ............................................................. 189
Hainsworth, D ..................................................... 189
Hargrave, C. O ..................................................... 165
Haustein, K .......................................................... 205
He, X ................................................................. 129
Hossaini, S. M. F .................................................. 99
Hoyer, D .............................................................. 40
Huang, W ........................................................... 171
Jalalifar H ........................................................... 115
Jalili, P ................................................................. 355
Jiang, H .............................................................. 326
Jiang, J .............................................................. 326
<table>
<thead>
<tr>
<th>Author</th>
<th>Page Numbers</th>
</tr>
</thead>
<tbody>
<tr>
<td>Johnson, S. F.</td>
<td>159</td>
</tr>
<tr>
<td>Kent, D.</td>
<td>181</td>
</tr>
<tr>
<td>Kirkwood, D.</td>
<td>205</td>
</tr>
<tr>
<td>Kizil, M. S.</td>
<td>48, 91</td>
</tr>
<tr>
<td>Kong, S.</td>
<td>326, 361</td>
</tr>
<tr>
<td>Leal, M.</td>
<td>375</td>
</tr>
<tr>
<td>Liu, H.</td>
<td>361</td>
</tr>
<tr>
<td>Liu, M.</td>
<td>348</td>
</tr>
<tr>
<td>Liu, T.</td>
<td>249</td>
</tr>
<tr>
<td>Lu Y.</td>
<td>22</td>
</tr>
<tr>
<td>McAllister, A.</td>
<td>91</td>
</tr>
<tr>
<td>McTyer, K.</td>
<td>8</td>
</tr>
<tr>
<td>Mitri, H.</td>
<td>129, 136, 348</td>
</tr>
<tr>
<td>Mojeddifar, S.</td>
<td>115</td>
</tr>
<tr>
<td>Moodie, A.</td>
<td>29</td>
</tr>
<tr>
<td>Munday, L.</td>
<td>189, 197</td>
</tr>
<tr>
<td>Mutton, V. S.</td>
<td>257</td>
</tr>
<tr>
<td>Napier, T.</td>
<td>178</td>
</tr>
<tr>
<td>Navin, J.</td>
<td>154</td>
</tr>
<tr>
<td>Nemcik, J.</td>
<td>141, 148, 154, 390</td>
</tr>
<tr>
<td>Nezhadshahmohamad, F.</td>
<td>99</td>
</tr>
<tr>
<td>Nicholls, B.</td>
<td>178</td>
</tr>
<tr>
<td>Nie, B.</td>
<td>129</td>
</tr>
<tr>
<td>Organ, M.</td>
<td>390</td>
</tr>
<tr>
<td>Pascoe, R.</td>
<td>91</td>
</tr>
<tr>
<td>Pathan, A. G.</td>
<td>105</td>
</tr>
<tr>
<td>Plush, B.</td>
<td>231</td>
</tr>
<tr>
<td>Porter, I.</td>
<td>148, 154, 385</td>
</tr>
<tr>
<td>Preusse, A.</td>
<td>83</td>
</tr>
<tr>
<td>Prout, R.</td>
<td>205</td>
</tr>
<tr>
<td>Ralston, J. C.</td>
<td>165</td>
</tr>
<tr>
<td>Reddish, D. J.</td>
<td>105</td>
</tr>
<tr>
<td>Reid, D. C.</td>
<td>165</td>
</tr>
<tr>
<td>Remennikov, A. M.</td>
<td>257</td>
</tr>
<tr>
<td>Ren, T.</td>
<td>231, 239, 269, 315</td>
</tr>
<tr>
<td>Rowland, J. A.</td>
<td>214</td>
</tr>
<tr>
<td>Saghafi, A.</td>
<td>285</td>
</tr>
<tr>
<td>Sahebi, A. A.</td>
<td>115</td>
</tr>
<tr>
<td>Saydam, S.</td>
<td>355</td>
</tr>
<tr>
<td>Seedsman, R.</td>
<td>60</td>
</tr>
<tr>
<td>Singh, R. N.</td>
<td>105</td>
</tr>
<tr>
<td>Author</td>
<td>Page(s)</td>
</tr>
<tr>
<td>-------------------------</td>
<td>-----------</td>
</tr>
<tr>
<td>Sroka, A.</td>
<td>83</td>
</tr>
<tr>
<td>Stace, R.</td>
<td>385</td>
</tr>
<tr>
<td>Sutherland, T.</td>
<td>8</td>
</tr>
<tr>
<td>Tajdus, K.</td>
<td>83</td>
</tr>
<tr>
<td>Tecchia, F.</td>
<td>171</td>
</tr>
<tr>
<td>Thomas, R.</td>
<td>40</td>
</tr>
<tr>
<td>Thompson, J.</td>
<td>197</td>
</tr>
<tr>
<td>Tien, J. C.</td>
<td>225</td>
</tr>
<tr>
<td>Towns, J.</td>
<td>148</td>
</tr>
<tr>
<td>Trueman, R.</td>
<td>40</td>
</tr>
<tr>
<td>Tu, S.</td>
<td>22</td>
</tr>
<tr>
<td>Veera Reddy B</td>
<td>249</td>
</tr>
<tr>
<td>Wang, F.</td>
<td>22</td>
</tr>
<tr>
<td>Wang, H.</td>
<td>335, 343</td>
</tr>
<tr>
<td>Wang, L.</td>
<td>326, 335</td>
</tr>
<tr>
<td>Wang, P.</td>
<td>205</td>
</tr>
<tr>
<td>Wang, Z.</td>
<td>269, 315</td>
</tr>
<tr>
<td>Wei, J.</td>
<td>348</td>
</tr>
<tr>
<td>Wen, Z.</td>
<td>348</td>
</tr>
<tr>
<td>Widzyk-Capehart, E.</td>
<td>197, 205</td>
</tr>
<tr>
<td>Wiklund, B.</td>
<td>48</td>
</tr>
<tr>
<td>Wu, D.</td>
<td>277</td>
</tr>
<tr>
<td>Wu, Q.</td>
<td>22</td>
</tr>
<tr>
<td>Xiao, W.</td>
<td>348</td>
</tr>
<tr>
<td>Yarlagadda, S.</td>
<td>239, 249</td>
</tr>
<tr>
<td>Zhai, Q.</td>
<td>343, 361</td>
</tr>
<tr>
<td>Zhang, L.</td>
<td>269, 315</td>
</tr>
<tr>
<td>Zhang, X.</td>
<td>225, 326</td>
</tr>
<tr>
<td>Zhang, Y.</td>
<td>225</td>
</tr>
</tbody>
</table>