Proceedings of the 2012 Coal Operators' Conference

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COAL OPERATORS’ CONFERENCE AWARD

Robert Kininmonth

“In recognition of his outstanding contribution to Australasian mining”

Bob Kininmonth graduated from the University of Otago (New Zealand) with a bachelor of mining engineering degree. He was employed in England, New Zealand and Australia before joining the then ICI (now Orica), as an explosives field engineer. After a period he left to join the Inspectorate of the New South Wales Department of Mines.

Bob has been an active member of the Melbourne, Sydney and Illawarra Australasian Institute of Mining and Metallurgy, (AusIMM) Branches, including many years on the Illawarra Branch committee and a period as Branch Chair. Bob is the principal editor of the, Monograph 12 (third edition, 2009) of the Australasian Coal Mining Practice, and co-editor of Monograph 21 “History of Coal Mining in Australia”. He is the Chair of the Illawarra Branch Heritage Sub-committee, ensuring the retention and documentation of coal mining history in the region, as well as being the technical adviser for the recently released “Beneath Black Skies” DVD documentary. Bob is also the Chair of the Illawarra Gas and Coal Outburst Committee.

With the Australian Coal Operators’ Conference, Bob has been a member of the organising committee, a member of the editorial board and reviewer of papers since 2001. He was also the principal reviewer of papers for the International Application of Computers in the Minerals Industry Symposium (APCOM) held in Wollongong, 2011.

Bob has received both the AusIMM Branch Service Award and the Institute Service Award for his contributions to safety and coal mining practice in the Illawarra region and throughout Australia.
FOREWORD

Coal 2012 marks the start of amalgamation of both surface and underground operations under a common umbrella of Coal Operators’ Conference. It is an opportune time for the amalgamation because of; the increasing expansion of the coal industry activities in both operations, with significant movements of technological knowhow and expertise across the defining line; the engagement of many consulting services in both operations, with many of their members being regular contributors to the Coal Operator’s Conference; the recent mergers of equipment manufacturers with interests in both surface and underground operations. All these make a sensile strategy for better positioning of the conference in the coal mining scene. This year also marks the inauguration of an annual award to individuals recognised for their outstanding contributions to the industry, the science and technology of mining and the Coal Operators’ Conference.

The Coal Operators’ Conference is growing in acceptance by the industry. The online papers are increasingly accessed from different corners of the world, thanks to the University of Wollongong research online initiative http://ro.uow.edu.au/coal. This pioneering online access initiative has now being followed by several other mining conferences.

The steady increase in the delegate participation at the conference is a welcome recognition of its importance by the industry. This is being acknowledged by the increasing number of paper contributions, sponsorships and exhibitors. A total of 430 papers and well over 1,700 speakers and attendees from the Australian coal mining industry as well as 15 foreign countries have made this conference a recognised event since its beginning in 1998.

The Pike River disaster is now recognised as having a significant impact on issues related to safe mining under hazardous conditions. This disaster, as tragic as it was, has brought out a number of important issues, with consequences for future mining practices with increased emphasis to miner safety and management responsibility. Thus the allocation of a special technical session deserves its place at this conference.

The current practice of making available a copy of the proceedings will continue to be maintained as a norm and will be available mainly to the conference attendees with some limited additions for special circulation such as libraries. For others, the papers will be available online. All papers are peer reviewed for technical and editorial competence, which is then finalised in accordance with the recognised standard. The ultimate aim of the proceedings quality improvement is to secure a respectable ranking recognition and the publication of selected papers in high quality international journals and transactions.

Sincere thanks to all the authors of the papers, the conference sponsors, the exhibitors and above all participants without which this conference would not be realised. We encourage and welcome your paper contributions.

Special thanks to the conference organising team, particularly Bob Kinninmonth, Jan Nemcik, Ting Ren; various paper reviewers; Zhongwei Wang for typesetting of the proceedings; Peter Vrahos of Eventico for the conference general management; The university of Wollongong printery staff Tristan Dus for proceedings cover design, Maria O’Hearn and others for printing the conference proceedings at lightening speed, and UniCentre for catering.

Associate Professor Naj Aziz
Conference chairman and convenor
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LARGE EXCAVATIONS AND THEIR EFFECT ON DISPLACEMENT OF LAND BOUNDARIES

Jan Nemcik, Naj Aziz and Ting Ren

ABSTRACT: A study to estimate land surface movement caused by large surface excavations in sedimentary strata is presented. In stratified or jointed strata the stress relief driven movement adjacent to large excavations can be significantly larger than expected. High lateral stresses measured in Australia and other places around the world indicate that the ratio of horizontal to vertical stress has been particularly high at a shallow depth. The in situ strata is in compression and during excavation, stress is relieved towards the opening causing strata movement. Large excavations such as, open cut mines or highway cuttings, can initiate an extensive horizontal slide of surface layers towards the excavation. These ground movements can be damaging to surface structures such as water storage dams and large buildings. Based on stress measurements at shallow depths in Australian coal mines the study presented here calculates the extent of potential ground movement along the bedding surface adjacent to large excavations and provides a new prediction tool of land movement at the excavation boundary that can benefit the geotechnical practitioners in the mining industry.

INTRODUCTION

Numerous stress measurements undertaken in Australian mines and elsewhere around the world provide evidence of significant lateral stress in rock below the earth’s surface. The lateral stress appears to increase with depth and can be larger than the vertical stress. The maximum horizontal stress is usually oriented in a specific direction depending on the geographic location and ground structure formation and its magnitude can be higher than the vertical stress. The minor horizontal stress is oriented at 90° to the maximum horizontal stress.

On a large scale the faulted and bedded ground behaves as a fractured rock mass. The continuous tectonic movement induces large horizontal stresses within the ground that are partially relieved close to the surface as the unconfined fractured rock displaces in time along the faults and other discontinuities. The amount of stress relief below the surface depends on the minimum compressive stress that provides confinement to the triaxially loaded fractured rock mass. As the depth increases, the confining stress will increase, thereby allowing a progressively larger horizontal stress to transmit through the fractured ground. Taking this into consideration, the maximum lateral stress at a shallow depth should be very low. However, stress measurements indicate that there can be a considerable amount of the lateral stress locked within the strata very close to the surface. In general, if fractures are plentiful and are oriented at a low angle to the tectonic movement, the lateral stresses would be low however, if these fractures are not present or are oriented at a high angle to the direction of ground stress, a significant proportion of the lateral stress can remain locked within the ground. Large excavations can relieve the locked lateral stress within the surrounding strata mobilising large displacements towards the excavation that can cause damage to surrounding structures. This effect can be magnified in sedimentary strata where failure of weak bedding planes can initiate a far reaching lateral slide of strata.

STRESS AT SHALLOW DEPTH

A number of stress measurements around the world were analysed by Mark (2010) indicating that the average lateral stress magnitude at a shallow depth may be of a significant value. Mark’s statistical analysis matched the equation \( S_{h\text{max}} = B_0 + B_1 \text{(Depth)} \) to the data, where \( S_{h\text{max}} \) represents the maximum horizontal stress \( (\sigma_1) \), \( B_0 \) is an excess stress and \( B_1 \) is the gradient of lateral stress increase with depth of cover. The excess stress is described as the lateral stress extrapolated from the data to be the near surface stress (Figure 1). Mark estimated that the maximum lateral stress near the surface is on average about 7 MPa, however the plotted data indicate variation from 0 to 12 MPa. The excess stress described by the linear equation serves as a prediction tool of lateral stress at a depth. If...
considering the measured values near the surface only than the average lateral stress at the surface may not be greater than 5 to 6 MPa.

Variations of rock stiffness within the sedimentary strata beds can influence the magnitude of the measured stress and a normalisation technique is recommended to re-calculate all measured data to an average Young’s Modulus of rock (Nemcik, et al., 2005). Since about 70% of the earth’s crust is sedimentary in composition of various stiffness, normalised stress data would increase the accuracy of the lateral stress. Figure 2 shows the measurements by Strata Control Technology (SCT) representing the pre-mining maximum lateral stress in Australian coal mines (Nemcik, et al., 2006). These measurements were taken exclusively in coal measures. All measurements were normalised to the average Young’s Modulus of 16 GPa of common sandstone. Despite the normalisation, there is a scatter of lateral stress magnitudes at a shallow depth probably due to orientation of structures within the strata. From the data presented in Figures 1 and 2 the maximum lateral stress close to the surface can be assumed to be approximately 5 MPa.

![Figure 1 - The coal stress measurements data base, showing predicted ranges of stress for regions with “normal” and “low” depth gradients (Mark, 2010)](image1)

![Figure 2 - Increase of pre-mining maximum horizontal stress with depth in Australian coal mines (Nemcik, et al., 2006)](image2)
INFLUENCE OF THE LATERAL STRESS RELIEF ON STRATA DISPLACEMENTS IN STRATIFIED ROCK

Large surface excavations relieve the lateral stress towards the free excavated face. In an elastic rock, predictable strata displacements occur. Sedimentary rock formations that cover some 70% of the world’s surface are predominantly bedded with discontinuities that are much weaker than the strength of the surrounding rock. The stress relief towards the excavation generates significant shear stresses along the bedding planes and under the right conditions, failure of the weakest bedding plane occurs and initiates displacements towards the opening (Figure 3). The slip along weak bedding planes initiate displacements that are greater than displacements in massive strata. In many cases floor failure and heave can be experienced within the excavation.

Figure 3 - Cross-section of excavated trench in laterally loaded bedded strata

CALCULATIONS OF LATERAL DISPLACEMENTS IN BEDDED STRATA ADJACENT TO LARGE EXCAVATIONS

Lateral displacement along a failed bedding plane is influenced by stress relief, strength of the bedding plane, stiffness of strata and the depth of cover. The simplified method presented here assumes a shallow vertical trench, a single weak bedding plane adjacent to the excavation and the in situ lateral stress $\sigma_x$. The failure mechanism is detailed in Figure 4a and 4b. Mathematical equations verified by a numerical model were used to calculate the total failure length along a single bedding plane and the lateral displacement towards the opening. Note that in this initial model the strata deformation due to the Shear Modulus of rock is not taken into consideration while for simplicity the cohesion of a weak bedding plane is also assumed to be zero.

Figure 4a - Compressed strata before excavation

Figure 4b - Stress relief and movement towards the excavation
The initial analysis also assumed a single rock bed where the uniformly distributed in situ lateral stress \( \sigma_x \) is relieved into the excavation. The total length of bedding failure and the lateral displacement towards the opening are calculated as follows:

The failure along bedding plane will occur when the lateral shear along the bedding plane exceeds the bedding shear strength. This can be described in terms of forces as:

\[
F_x \geq N \tan \phi + C_b
\]  

where:

- \( F_x \) = Lateral reaction force along the bedding plane induced by the lateral expansion of the rock bed above;
- \( N \) = Normal force on bedding plane;
- \( \phi_b \) = Angle of friction along bedding plane and
- \( C_b \) = Cohesion force along the bedding plane (assumed here to be zero);
- It is assumed that \( C_b = c_b A_b \) when
- \( c_b \) = Cohesion stress along the bedding plane (assumed here to be zero);
- \( A_b \) = Area of the bedding plane under consideration.

The small reaction force \( dF_{\text{(R)}} \) within the distance \( dx \) along the failed bedding plane (Figure 4b) opposes the movement of expanding strata above and can be calculated as:

\[
dF_{\text{(R)}} = \gamma h b \tan \phi \ dx
\]  

Where:

- \( dF_{\text{(R)}} \) = the small reaction force along the bedding within the distance \( dx \);
- \( \gamma \) = Rock density;
- \( \phi \) = Angle of friction along the bedding plane;
- \( h \) = Depth above the bedding plane;
- \( b \) = Out of plane rock bed thickness.

The small force \( dF_{\text{(R)}} \) is opposing the difference in lateral stress \( d\sigma_x \) acting over the cross-sectional area \( A=hb \) and the distance \( dx \). Note that the small force \( dF_{\text{(R)}} \) is proportional to the bedding depth \( (h) \), the angle of friction \( (\phi) \) along the bedding and is opposing the lateral movement due to the stress relief above the failed plane.

The rate of lateral stress increase \( d\sigma_x/dx \) from the excavation edge can be expressed as:

\[
\frac{d\sigma_x}{dx} = \frac{dF_{\text{(R)}}}{bhdx} = \gamma \tan \phi \text{ or } dF_{\text{(R)}} = \gamma bh \tan \phi dx
\]

Therefore from the equation (2) the magnitude of lateral stress in rock above the bedding anywhere along the horizontal distance \( x \) from the excavation edge towards the end of the bedding failure can be expressed as:

\[
\sigma_x = \int_0^x \gamma \tan \phi \ dx = \gamma \tan \phi \ x
\]  

Equation (3) indicates that in this particular case the magnitude of lateral stress \( \sigma_x \) adjacent to the excavation edge increases linearly with the constant frictional resistance along the bedding until it reaches the virgin lateral stress value \( (\sigma_{\text{Virgin}}) \).
The length of the bedding failure ($L$) can be calculated from the total frictional force generated along the failed bedding plane that has to balance the opposing force generated by the lateral stress in the overburden rock equal to $h b \sigma_{\text{Virgin}}$ at a distance $L$ from the excavation face.

For static equilibrium the total frictional and driving forces must be equal. The length of the bedding failure ($L$) can be calculated from:

$$h b \sigma_{\text{Virgin}} = \int_0^L dF_x = \int_0^L \gamma h b \tan \phi \, dx = \gamma h b \tan \phi \, L$$

Where the total driving force

$$F_{\text{envirin}} = \sigma_{\text{Virgin}} \, h b \text{ and } L = \frac{\sigma_{\text{Virgin}}}{\gamma \tan \phi} \quad (4)$$

The derived equation (4) can be used to calculate the approximate length of the single bedding plane failure.

The total lateral displacement at the excavation edge along the failed bedding plane can be calculated by integrating small displacements along the bedding. These can be calculated from the stress relief within the overburden rock. The small displacement ($dD$) along the lateral distance $dx$ can be expressed as:

$$dD = \varepsilon \, dx = \frac{d\sigma_x}{E} \, dx$$

Where:

- $\varepsilon$ = strain relief at the distance $x$;
- $d\sigma_x$ = change in lateral stress in rock bed at distance $x$ from excavation;
- $E$ = Young’s Modulus of the rock bed.

The total lateral displacement ($D_0$) at the excavation can be calculated as follows:

$$D_0 = \frac{1}{E} \left[ \sigma_{\text{Virgin}} - \gamma \tan \phi \, x \right]_0^L = \frac{1}{E} \left[ \gamma \tan \phi \, L - \gamma \tan \phi \, x \right]_0^L$$

$$= \frac{\gamma \tan \phi}{E} \left[ \frac{L^2}{2} - \frac{x^2}{2} \right]_0^L = \frac{L^2 \gamma \tan \phi}{2E} \quad (5)$$

And the total displacement anywhere along the failed bedding plane is:

$$D_s = \frac{1}{E} \sigma_{\text{Virgin}} \tan \phi \, x \, dx = \frac{\gamma \tan \phi}{E} \int_0^L (L - x) \, dx$$

$$= \frac{\gamma \tan \phi}{2E} \left[ L^2 - 2Lx + x^2 \right] \quad (6)$$
NUMERICAL MODEL

The numerical modelling using FLAC, (Itasca 2005) was used to validate the calculations of the bedding failure. The bedding plane was simulated using the FLAC interface available to model movement along the discontinuities. The properties used in the FLAC model are given in Table 1. The calculated and modelled results of the slip length and the lateral strata displacement for various properties along the bedding plane are presented in Figures 6a and 6b. The length of the single bedding plane failure varied depending on the bed friction properties that can be very low for weak clay or mudstone deposits and larger for competent rocks such as sandstone. For bedding friction ranging from 15° to 45° and range of lateral stresses from 1 to 10 MPa the results (Figure 6a and 6b) indicate strata displacements from very small up to 480 mm at the excavation edge with slip length along the bedding ranging from 40 m to 1 500 m.

Table 1 - FLAC - Modelling parameters

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<td>Bulk Modulus (GPa)</td>
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<tr>
<td>Shear Modulus (GPa)</td>
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<tr>
<td>Rock Density (N/m³)</td>
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<td>Rock bed thickness (m)</td>
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<td>Range of virgin lateral stress (MPa)</td>
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<td>Normal stiffness (GPa)</td>
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<td>Shear stiffness (GPa)</td>
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<tr>
<td>Angle of friction (°)</td>
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<td>Cohesion (MPa)</td>
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<td>Tension (MPa)</td>
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<tr>
<td>Dilation (°)</td>
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Figure 6a - Slip length

Figure 6b - Maximum lateral displacements of strata at the bedding plane level

Figure 7 shows the magnitudes of shear slip along the weak bedding plane for various angles of friction and lateral stress of 5 MPa. As expected, lower displacements occur for higher friction along the bedding.
DISCUSSION

Good correlation between the calculated and modelled shear displacements and bed failure length were achieved. Calculations of the lateral slip slightly overestimate the modelled results as the elastic distortion of strata above the bedding was not taken into account. The bedding failure mechanism and the prediction method of surface displacements adjacent to large excavations can be calculated as suggested. The derived calculations of strata displacements are of a simple form. This was done primarily to make the reader familiar with the proposed concepts of strata failure and movement along the bedding planes. Together with the prediction of strata movement adjacent to large excavations, land surveys should be conducted to build a database for strata movement verifications. The calculations of lateral strata movement can be very useful when protecting large surface structures such as concrete water storage dams or bridges that are very sensitive to ground movement.

CONCLUSIONS

The intention of this paper is to provide the reader with a better understanding of the ground behaviour at a shallow depth and to stimulate further research in this poorly understood topic. The origin of lateral stress at a shallow depth, the lateral stress relief adjacent to an excavation, the bedding failure mechanism and lateral displacements along failed bedding planes due to the stress relief have been explained. The described theory indicates that it is possible to use simple calculations to estimate relatively complex strata movements adjacent to large excavations within the bedded ground. The derived equations provide a simple solution to estimate the lateral ground movement without resorting to more complex numerical modelling.

Further research is underway to enable displacement calculations at any point within the strata (surface or underground) incorporating multiple bedding failure, shear distortion and stress distribution as measured underground.

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APPLICATION OF A TRANSVERSELY ISOTROPIC BRITTLE ROCK MASS MODEL IN ROOF SUPPORT DESIGN

David Oliveira

ABSTRACT: Accurate modelling of the potential failure modes in the rock mass is an essential task towards a robust design of roof support systems in coal mines. The use of generalised rock mass properties based on averaged properties (e.g. Hoek-Brown model) has been found to limit the capability to reproduce the actual rock mass behaviour which may include a wide range of interacting and complex failure mechanisms such as shear and tension fracturing of the intact rock and shear and separation of pre-existing discontinuities, including re-activation. Recent studies have also shown that traditional models, such as the Mohr-Coulomb, may not accurately describe the behaviour of the intact rock, particularly for stress induced failures where spalling and slabbing are observed. This is mainly due to the cohesion and friction components of the shear strength of the intact rock not being mobilised at the same rate with strain-softening of the former component playing an essential role in the post peak behaviour. In addition, coal measure rocks are often transversely isotropic, both by way of the preferred orientation of clay particles within the finer grained lithology and by bedding textures and bedding partings, and this is often ignored in computer simulations. A newly developed transversely isotropic brittle rock mass model is applied in the simulation of a hypothetical and simple roadway development. A Cohesion Weakening - Friction Strengthening (CWFS) approach is adopted to describe the intact rock where the mobilisation and strain-softening of the two shear strength components are linked to plastic deformation. Failure and plastic deformations are also allowed to develop along any number of ubiquitous joint sets using a conventional Mohr-Coulomb failure criterion and applying the stress-strain correction accordingly. The impacts of anisotropy and brittle rock on the development of the excavation disturbed zone or height of softening, as often referred to, are investigated and their implication in the roof support design discussed.

INTRODUCTION

According to Seedsman (2011), most coal roof support design is either precedent/practice or empirically based using a rock mass classification scheme. Although a relatively dense bolting pattern is typically installed at the face often supplemented by longer reinforcement elements, such as cable bolts, 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. Seedsman (2011) also states that 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.

Detailed monitoring studies conducted in coal mines in a number of countries have shown that the failure mechanisms can be highly complex, involving fracture of rock, failure of bedding or joints, buckling of parted rock, and slip along weak surfaces (Gale, et al., 1992; Gale and Tarrant, 1997; Mark, et al., 2000). In such a complex scenario, computer simulations seem to provide more realistic results when a detailed geotechnical characterisation of the strata and stress field is available, and the many potential failure mechanisms involved are considered.

For example, Strata Control Technology (SCT) has reported successful modelling with a relatively comprehensive rock model at a wide variety of mine sites (Gale, et al., 1997; Sandford, 1998; Kelly, et al., 1998). Their model is similar to the Strain-softening Ubiquitous Joint Model (SUBI) included in the finite difference codes FLAC and FLAC3D where rock failure is based on Mohr-Coulomb criteria relevant to the confining conditions within the ground. A range of potential failure modes are simulated including:

- Shear fracture of intact rock;
- Tension fracture of the rock;
• Bedding plane shear; and,
• Tension fracture of bedding (bedding separation).

In the SCT model, as well as the SUBI model, the intact rock matrix exhibits strain-softening post-failure behaviour with cohesion, friction angle, dilation angle and tensile strength specified as functions of the "plastic strain". A weakness plane of any orientation is also included in the SCT model, and this weakness plane may exhibit strain-softening behaviour. Due to this limitation, only bedding planes are modelled as it is the dominant weakness plane in coal measure rocks. However, rock masses in coal measures will typically have two other joint sets aligned orthogonally such that a valid model is a rock mass of cubes with dimensions from centimetres to metres (Seedsman, 2011).

In addition, 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, which according to Seedsman (2011) could only be modelled in a continuum assumption by invoking a transversely isotropic elastic model at the cost of only being able to conduct elastic analysis.

A new rock model has been developed at Coffey Geotechnics (Coffey) in order to address the above limitations. In addition, modelling of brittle behaviour of the intact rock by means of a Cohesion Weakening - Friction Strengthening (CWFS) approach and its effect on the development of the roof softening (fractured) zone are also discussed.

EXTENDED SUBIQUITOUS ROCK MODEL

The Extended Subiquitous (ESUBI) rock model was developed using subroutines in FLAC3D code and is in principle similar to the SCT and the Itasca SUBI models, thus, modelling the same range of potential failure modes. However, the model has been extended to include the following features:

• Unlimited number of ubiquitous joint sets given with any dip and dip direction angles;
• Transverse Isotropy with two different elastic moduli given: $E_1$, parallel to the plane of isotropy and $E_3$ perpendicular to this plane which is specified by a dip angle and a dip direction angle;
• Different elastic moduli for loading and unloading-reloading paths which is a behaviour typically observed in the field, particularly due to the highly non-linear effects of existing rock defects. As a result, total of four elastic moduli are given: two for loading and two for unloading-reloading.

The ESUBI rock model also accounts for both intact rock matrix and joint strain-softening/hardening post-failure behaviour where cohesion, friction angle, dilation angle and tensile strength are specified as functions of the appropriate "plastic strain". However, even when using strain-softening/hardening functions the conventional modelling approach is to implicitly assume that the cohesive and the normal stress-dependent frictional strength components are mobilised simultaneously, i.e., they are assumed to be additive before yielding of the rock.

Recent studies have shown that models based on the simultaneous mobilisation of cohesive and frictional strength components have not been successful in predicting the extent and depth of brittle failure. Hajibololmaji et al. (2002) have presented an approach where only the cohesional strength component is mobilised up to a stress level corresponding to the onset of micro-cracking. Beyond this stress level, there is a degradation of cohesion, i.e. softening, and mobilisation of the frictional strength component, which takes place due to the development of micro fractures. This CWFS mechanism is depicted in Figure 1. In this figure $\varepsilon_c^p$ marks the plastic shear strain beyond which there is only a residual cohesion and $\varepsilon_f^p$ marks the plastic shear strain at which the frictional component is fully mobilised.

Hypothetical triaxial tests on jointed rock samples

Hypothetical triaxial tests on "virtual" rock mass samples are simulated in FLAC3D in order to demonstrate the capability of the ESUBI rock model with a CWFS approach as discussed above. Although, the ESUBI model allows for an unlimited number of joint sets, only one joint set (i.e. a bedding plane) was modelled for simplicity and clarity of the results, which could then be compared to simple analytical assessment for validation of the results. The adopted parameters are presented in Table 1 in which Poisson’s ratio equal to 0.25 has been adopted. Upon tensile failure, the residual tensile
strength is instantaneously set to zero. Unloading-reloading moduli have been assumed equal to the loading moduli. Dilation angles have been assumed zero although dilative behaviour would be expected for coal measure rocks and rock defects.

Figure 1 - Mobilisation of the strength components - cohesion and friction - in the CWFS model (after Hajiabdolmajid, et al., 2002)

Table 1 - Rock and joint parameters adopted

<table>
<thead>
<tr>
<th>Rock mass</th>
<th>Intact rock</th>
<th>Bedding plane</th>
</tr>
</thead>
<tbody>
<tr>
<td>(E_1) (GPa)</td>
<td>(E_3) (GPa)</td>
<td>(\sigma_{ci}) (MPa)</td>
</tr>
<tr>
<td>10</td>
<td>2.86</td>
<td>16</td>
</tr>
</tbody>
</table>

where \(E_1\) and \(E_3\) are the horizontal and vertical Young's moduli of the rock mass respectively when the joint dip angle is equal to zero (i.e. it rotates according to the specified joint angle); \(\sigma_{ci}\) and \(\sigma_{ti}\) are the Uniaxial Compressive Strength (UCS) and tensile strength of the rock matrix respectively; \(\psi_{pi}\) is the peak friction angle of the rock matrix mobilised at a plastic shear strain \(\epsilon_{ci}\); \(c_{pi}\) is the peak cohesion of the rock matrix assumed half of the stress at which unstable cracking starts to occur (approximately 80% of the UCS); \(\sigma_{ti}\) is tensile strength of the bedding planes \(\psi_{pj}\) is the peak friction angle of the bedding planes simultaneously mobilised with the peak cohesion, \(c_{pi}\); and \(c_{ri}\) is the residual cohesion of the bedding planes at a plastic shear strain \(\epsilon_{cj}\) which has been assumed 10% of the peak value. Figures 2 and 3 present the results of the hypothetical triaxial tests for confining pressures equal to zero and 1 MPa respectively. The stress-strain curves clearly indicate the effect of varying the joint dip angle and consequently the plane of transverse isotropy on both rock mass sample strength and stiffness. In addition, the onset of unstable cracking or start of mobilisation of the frictional strength component is clearly observed by a kink in the stress-strain curve of those samples where failure occurs entirely within the rock matrix, i.e. samples with joint dip angle of 0°, 30° and 90°.

Figure 2 - Stress-strain curves for virtual samples tested at \(\sigma_3 = 0\) MPa and varying joint angle

Figure 3 - Stress-strain curves for virtual samples tested at \(\sigma_3 = 1\) MPa with varying joint angle

Figure 4a presents the axial stress developed upon axial loading at total axial strain of 0.08% for the rock mass samples under zero confining pressure. The results clearly indicate the effect of the
transverse isotropy with increasing stiffness for increasing joint angle. The dotted lines indicate the values that would be observed if a fully isotropic model had been adopted with either $E_1$ (upper limit) and $E_3$ (lower limit), i.e. the results would be constant and independent of the joint angle. Figure 4b presents values of peak axial stress and typical analytical U-shape strength curve with matching results of the ESUBI model.

![Graph showing stress mobilisation and peak axial stress](image)

**Figure 4 - (a) Stress mobilised at an axial strain of 0.08%; (b) Peak axial stress**

### EFFECT OF CWFS BEHAVIOUR AND TRANSVERSE ISOTROPY ON ROOF/FLOOR SOFTENING

A simple hypothetical 2D model of a 5 m wide by 3 m high roadway excavation is presented in order to illustrate the potential effects of the both transverse isotropy and the non-simultaneous mobilisation of the cohesion and friction strength components, i.e. a CWFS approach, on the development of the excavation disturbed zone, i.e. the softening/fracturing zone in the roof/floor.

The roadway to be excavated was under a high initial horizontal stress of 12 MPa and a vertical stress of 5 MPa. The following simplifying assumptions were made for both clarity of the results and to highlight the main potential differences in the modelling approaches:

- A homogenous rock mass represented by the parameters given in Table 1 was adopted both above and below a 3 m thick coal seam;
- Despite the ESUBI model capability of accounting for several joint sets only bedding planes are modelled;
- The coal seam was modelled assuming the parameters given in Table 2. For the coal, simultaneous mobilisation of friction and cohesion has been assumed;
- Excavation was assumed instantaneous which does not represent the more realistic gradual excavation of roadway development;
- No roof support was installed.

#### Table 2 - Coal parameters adopted

<table>
<thead>
<tr>
<th>Rock mass</th>
<th>Intact rock</th>
<th>Bedding</th>
</tr>
</thead>
<tbody>
<tr>
<td>$E_1$ (MPa)</td>
<td>$E_3$ (MPa)</td>
<td>$\sigma_{ci}$ (MPa)</td>
</tr>
<tr>
<td>5</td>
<td>1.43</td>
<td>10</td>
</tr>
</tbody>
</table>

The following cases were investigated:

- **Case 1** - An isotropic model was adopted for both coal and surrounding rock mass with $E_i$ assigned. Simultaneous mobilisation of cohesion and friction was assumed for both rock types;
Case 2 - A transverse isotropic model was adopted for both coal and surrounding rock mass with both $E_1$ and $E_3$ assigned. Simultaneous mobilisation of cohesion and friction is assumed for both rock types;

Case 3 - An isotropic model was adopted for both coal and surrounding rock mass with $E_1$ assigned. A CWFS approach is adopted to model brittle behaviour;

Case 4 - A transverse isotropic model was adopted for both coal and surrounding rock mass with both $E_1$ and $E_3$ assigned. A CWFS approach was adopted to model brittle behaviour.

**Model results**

Figures 5 to 12 present the results of the four cases investigated. These figures present both the failure modes experienced in the models as well as the pattern of total displacements.

![Image](image.png)

**Figure 5 - Failure modes observed in Case 1 (exaggerated 15x)**

![Image](image.png)

**Figure 6 - Total displacements observed in Case 1 (exaggerated 15x)**
Figure 7 - Failure modes observed in Case 2 (exaggerated 15x)

Figure 8 - Total displacements observed in Case 2 (exaggerated 15x)

Figure 9 - Failure modes observed in Case 3 (exaggerated 15x)
The two cases with isotropic models (i.e. 1 and 3) showed very similar pattern and magnitude of displacements. When compared to the transverse isotropic models it becomes evident that those models will either underestimate or overestimate the displacements depending on the modulus adopted.
In this case, as the higher modulus was adopted, both vertical and horizontal displacements were underestimated. The main difference between Case 1 and Case 3 is the extent and shape of the softened zone which is in agreement with the findings of (Hajiabdolmajid, et al., 2002). A simple brittle model with simultaneous mobilisation of strength parameters tends to underestimate the extent of the fractured zone. The CWFS approach indicates a deeper disturbed zone with a triangular shape that seems more realistic when compared with those often observed in the field.

The two cases (i.e. 2 and 4) with transverse isotropy seem to provide a more realistic displacement pattern compatible with those often observed in the field, i.e. a deeper zone of influence, with a more linear decrease into the roof. Measurements of displacement near the excavation face are often used as an indication of the softened zone. However, only Case 4 with the CWFS approach indicated a fractured zone that seems more realistic than those experienced in the field. Case 2 mainly limited to bedding plane shearing, although with a similar displacement pattern.

CONCLUSIONS

An extended strain softening/hardening ubiquitous joint model has been developed to address some of the limitations of existing rock models, particularly with respect to transverse isotropy and the number of rock joint sets. The hypothetical triaxial test results presented above indicate that the model is capable of representing a range of failure modes capturing the effects of transverse isotropy which is often ignored in current roof support design.

When combining the model with a CWFS approach, a better modelling of brittle behaviour, e.g. fracturing/spalling, is observed. As a result, the shape and extent of the failed roof is also better modelled, not only with respect to displacements.

Results of the hypothetical and simple roadway model indicate that only displacement profiles in the roof may not fully represent the extent of the softened zone which may have a significant impact in the design of the roof support. For example, a deeper fractured zones may required longer tensile member (e.g. cable bolts) for suspension of the failed roof mass, which may not be captured when using a conventional brittle model.

It is important to note that the above investigation is a hypothetical case study with significant simplifications that do not fully represent roadway development in coal mines. A detailed geotechnical characterisation of the strata and stress field is required before applying in real cases. However, the hypothetical example illustrates the main differences that may be expected for the different modelling approaches and the potential shortcomings of current design approaches.

REFERENCES


THE STRENGTH OF THE PILLAR-FLOOR SYSTEM

Ross Seedsman

ABSTRACT: The strength of the roof/pillar/floor system is controlled by the component with the lowest strength. In some coal seams the floor can be the weakest component and in these situations bearing capacity concepts drawn from foundation engineering can be applied. The low strength floors tend to be clay-rich and can be analysed as behaving in an undrained state (effective friction angle equals zero). A simple thin-layer bearing capacity equation has been found to correctly identify problematic low strength floors. The input variables are the unconfined compressive strength of the layer, its thickness, and the width of the pillar. All reported pillar collapses should be checked against this simple relationship and removed from the pillar collapse database if floor failure is indicated.

INTRODUCTION

A characteristic of many coal seams is the presence of low strength floors. The international coal industry makes reference to seat earths, underclays and fire clays. In Australia, the tuffs in the floor of the Wallarah and Great Northern Seam have sometimes, but not always, been reported to be very weak and have been implicated in unanticipated pillar behaviour. There can be weak claystones in the floor of the Bulli Seam. In the Bowen Basin, some of the early longwalls encountered major difficulties on the longwall face related to low strength floors. Both the South African and Australian pillar strength databases categorically state that there are no weak roof or floor failures in their databases.

There is no definition of what a low strength floor actually is: Is it less than a certain unconfined compressive strength or simply less than the strength of the coal? This lack of a definition, or even an accepted assessment process, can lead to poor mine design. A recent publication on geotechnical engineering in underground coal (Galvin, 2008) leaves this important question unanswered and dismisses earlier attempts to provide a simple assessment tool. This paper reviews the work on low strength floors and provides case study evidence that simple bearing-capacity methods do provide a useful tool: a tool that is in fact more robust than the empirical method for pillar strength itself.

ANALYTICAL AND EMPIRICAL MODELS IN GEOTECHNICAL ENGINEERING

Traditionally, pillar design utilises empirical methods based on a statistical analysis of databases of pillar collapse. The databases contain only pillar geometry and depth - there are no geotechnical parameters such as Unconfined Compressive Strength (UCS) or triaxial strength. The empirical approach does not invoke a failure mechanism and hence does not require the application of laws of physics. The engineering uncertainty in the subsequent design relates directly to the confidence in the data – in the South African database the coal seam and colliery are identified, in Australia not even the seams were identified and now cannot be as the database information has been destroyed (Galvin, 2010). The recommended factors of safety are based on an interpretation of a presumed normally distributed database of pillar collapse.

In analytical engineering approaches, there needs to be a behaviour model that can be interrogated using physical laws. Limit Equilibrium methods seek to calculate driving and resisting stresses at the point of failure. In some case, the arithmetic is very complex and numerical methods are used to estimate stresses - the behaviour model is still an input in terms of the selection of strength properties. Factors of safety are then based on engineering judgement recognising a number of uncertainties.

In an analytical method, uncertainty can be considered in three ways:

Model uncertainty

- Do we really know how the rock behaves?
- How well do the models represent actual behaviour?

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• Are complex models necessarily better?
• Are we applying the right model to the situation?

Parameter uncertainty

• Accepting the model is appropriate, how good are the inputs?
• Should we consider shear, compressive, tensile, or brittle strengths?
• What is the deformation modulus?
• What are the joint strength and stiffness properties?
• What are the joint orientations and spacings?

Human

• Bias, denial, ignorance, jealousy;
• Failure to conduct an appropriate site investigation;
• Arithmetic errors.

It is apparent that the appropriate values for factors of safety change (reduce?) as a project advances and knowledge improves.

It is worthwhile reviewing the history of research into floor behaviour and bearing failures in this framework.

FLOOR STUDIES

Seedsman (1988) studied the low strength floors in the Newcastle coalfield, with particular reference to the Awaba Tuff in the floor of the Great Northern Seam. At the time there had been several unanticipated subsidence incidents in which low strength claystones were implicated (e.g. Awaba, Chain Valley Bay, Gwandalan Point), and the early development of Cooranbong Colliery had hit major obstacles in attempting to enter the underlying Fassifern Seam.

The starting point for the study on the low strength floors was research from the Illinois Basin, which invoked foundation engineering (Stephen and Rockaway, 1981). To explain the failures, either very large factors of safety were required or the floor strengths were massively reduced with no precedent. There were two options - either dismiss the bearing capacity model and seek another failure mechanism or review the parameters. Bearing capacity theory is well established in civil engineering and importantly is scale independent (the magnitudes of the loads and the widths of the pillars are not material to the application of this elastic stress model). Bearing capacity is referenced in standard mining text books, typically Brady and Brown (1985). It was concluded the application of a bearing capacity model was appropriate.

The research then focussed on parameter uncertainty. The first thing to do was to determine whether low strength tuff behaves as massively overconsolidated clay. This means that when loaded quickly, the load is carried by the pore water (Figure 1). The key implication is that for rapid loading the effective friction angle is zero - over time the pore pressures dissipate, the clay consolidates (gains strength) and the friction angle tends to the drained values (about 25°). From a practical viewpoint, the immediate strength is the lowest.

With acceptance of undrained behaviour, it was possible to return to the bearing capacity equations with a friction angle of zero - this was nothing special as a major part of soil mechanics practice is based on the same assumption. Many of bearing factors vanish when the friction angle equals zero.

But there was still a problem with the required factors of safety to explain failure when reference was made to the available core logging. It was known the tuff was layered, although the scale of the layering was not specifically quantified in much of the old logging of the tuffs. A major advance was possible by referencing any of several thin layer equations - the equation of Mandel and Salencon (1969)
was used: Bearing capacity = UCS/2*(4.14 + W/2/h), where W is pillar width and h is layer thickness with an unconfined compressive strength (UCS).

Figure 2 provides some examples of what this equation implies. The horizontal axis is UCS, and the diagonal lines give the bearing capacity for different layer thickness for different pillar widths: for example a 1 m thick layer with a UCS of 1 MPa will have a bearing capacity of 7.1 MPa, which is greater than the pillar stress for shallow first workings but less than for deeper pillars in an extraction panel.

![Figure 1](image1.png)

**Figure 1 - Undrained triaxial test with pore pressure measurements of a sample of Awaba Tuff**

![Figure 2](image2.png)

**Figure 2 - Bearing capacity under 25 m wide pillars and a range of potential loadings**
This thin layer bearing equation also informs the site investigation what may be required. The focus needs to be on thin layers of very low strength. Figure 2 also includes some field strength categories: S4 - readily crumbled by hand, and S5 - trim with knife, thumbnail scratches core. In this context, examples of good logging have been found from 1947 but none subsequently until the last decade once the insight from the research was clear.

There is a strong bias in the Australian coal mining sector against any simple method that requires site investigations – strangely this does not seem to apply to the use of complex numerical codes where the site investigation demands are extreme and many of the input parameters cannot be determined anyhow. Whilst site investigations were not adopted by older mine management processes, it was not because the ground is highly variable and hence cannot be adequately characterised. In fact the opposite applies: diagenetic (rock forming) processes produce more consistent rock masses than weathering processes which produce soils. The failure to commit to site investigations was the result of ignorance, perception of cost, inconvenience, and unfortunately denial.

The reluctance of the industry to accept this bearing capacity approach has been disappointing to say the least. Floor failures have been dismissed because smaller pillars may have been formed – if this logic was applied to pillar collapse we would have no empirical design at all. The different scale of mine pillars compared to civil engineering footings has been invoked even though the relationships and equations of elasticity in general are independent of scale. The most bizarre outcome, and based on denial, was the application of the bearing capacity model using high presumed strengths that not surprisingly showed there was no bearing failure: this was then used to argue there was in fact no hazard. No attempt was made to actually measure the floor strength or even to back-analyse pillar failures and creeps in the adjacent panels. The new panel subsequently collapsed on very low strength material.

SUCCESSFUL APPLICATION

Undrained failure of the floor under a coal pillar can have serious consequences (Figure 3). Localised floor heave can seriously impact roadway serviceability both in terms of loss of roof control and poor trafficability - if not loss of access. For thicker/weaker layers of claystone, lateral extrusion of the clay may cause the collapse of the pillar with consequent loss of access and possibly unacceptable subsidence outcomes. This range of adverse outcomes demands a specific assessment starting early in the project stages – waiting for a set of mine-specific case studies can be too late.

![Figure 3 - Models for roadway distress associated with a bearing failure in the floor](image)

Awaba

The last of the longwalls at Newstan Colliery extracted the West Borehole Seam under previous Awaba workings, close to the creep documented in Galvin (2008). Site investigations into the Great Northern
Seam, and a review of very old core logging, revealed the presence of low strength Awaba Tuff layers. In 1961 Cliff McElroy logged: SILTSTONE (?)TUFF very friable and easily powdered between fingers, thickness of 3’1” (0.94 m).

Using accepted field strength estimates, this would have a UCS of 150-700 kPa (S4). For pillars on 20 m centres with roads of 6 m width, this would imply a bearing capacity of between 0.9 and 4.1 MPa. The pillar stress would be 1.5 MPa at 30 m depth. With this knowledge, it is not surprising the pillar system failed. At Awaba, the collapses were delayed until the panel span allowed the failure of the massive overlying Teralba Conglomerate - once the critical panel span was exceeded, the collapse was almost instantaneous.

From 2007 to closure in late 2011, Awaba successfully extracted wide panels (less than 100 m) without generating a pillar creep. A key part of this success was the use of the thin layer equation applied to the results of coring underneath standing pillars. The rigorous assessment process gave confidence to the workforce, the mine owner, and government regulators that another creep would not be induced.

South Bulli

Roadways involved in the creep in W and T Mains in the 1980s were recovered about 10 years ago. The creep began during roadway development and before the longwalls were retreated. Removal of the floor material did not initiate new movements. The edge of the creep was clearly identifiable within about 5 m of roadway length. The inbye and outbye roadways either side had the same pillar and abutment dimensions. Site investigations revealed a 380 mm thick tuff unit with a strength of about 400 kPa. The bearing strength on development was 7.7 MPa compared to a vertical stress of 16.5 MPa.

More recently Wongawilli extraction was conducted along strike of the creep zone. When floor heave developed under a fender, there was also a tendency for the intersection roofs to unravel. Poor roof conditions had been observed during the recovery of the Mains but in that case it had been ascribed to the use of very early roof bolts. The roof destabilisation is probably related to the abutment relaxation mechanisms proposed by Diederichs and Kaiser (1999) such that yield of one side of a roadway can lead to de-stressing of the roof and the possible onset of tensile stresses. It has also been speculated that the same mechanism applies to pre-driven roadways on low strength floors.

THE INTEGRITY OF PILLAR COLLAPSE DATABASE

Of particular concern is the claim in the pillar collapse databases that there are no instances of floor failure. Without a specific assessment, how is the claim made? In the South African data, knowledge of the seam and the colliery allows local users in that country to assess the validity of the claim (van der Merwe, 2006). This is not the case for the Australian data base, which was always confidential and has now been destroyed (Galvin, 2010).

The SC3 data point

Colwell (2010) proposed the SC3 case in the Australian pillar collapse database (Salamon, et al, 1996) was drawn from the Great Northern Seam at Wyee Colliery. The floor of the Great Northern Seam at Awaba Colliery has been discussed above. Galvin (2008) also includes a separate discussion on the low strength floor Great Northern Seam and its association with seven unexpected subsidence events.

Old core records including those near the possible site of SC3 have been examined. The logging is not ideal from a geotechnical perspective but units with S4 and S5 strength can certainly be confidently identified. Based on that experience and as a default position, it is anticipated that a floor layer of 1 m thickness and 500 kPa strength exists and evidence from site investigations to demonstrate otherwise is sought.

There are additional problems with SC3 as over the recent years the reported depth, the goaf width, and the time to failure have all changed, and there is now no possible verification. If the stated SC3 dimensions are 170 m deep, a 20 m pillar, and 5.5 m and 70 m voids are accepted, then the extraction ratio of 83% will results and hence a pillar stress of 23.9 MPa can be calculated. Invoking 70 m voids gives more credence to the proposition that SC3 is from the Great Northern Seam with its massive Teralba Conglomerate roof. By contrast, the default bearing capacity would be 3.5 MPa. Floor failure
is indicated, and at such low bearing strength extrusion of the floor and destruction of the pillar would be expected, which is consistent with the observations in Colwell (2010).

According to the rules of the database, there should be no failure of the roof or the floor. Removing SC3 will cause a major problem with the statistical analysis because this point is an outlier. The pillar strength design equation basically passes through this point. If it is removed, the pillar collapse databases for both Australia and South Africa are truncated at a width/height (W/H) ratio of 4.8 (Figure 4). The recent South African database also reveals a disturbing trend for collapsed pillars to have factors of safety well in excess of 1.2 while still constrained by the W/H ratio. Statistically there is zero probability of failure for pillars with width/height greater than 4.8. Pillar failure at greater aspect ratios could be due to unique combination of conditions, but it is wrong to extrapolate statistical trends and hence probabilities beyond a data base.

The question to be asked is why is the database truncated at a width to height ratio of 4.8? This should be the subject of more research. The author’s view is based on the kinematics of pillar collapse (Figure 5). A highly structured coal can have a low UCS and hence low cohesion, but the friction angle must always be finite. It is necessary to separate the concept of pillar collapse from pillar crushing and compression. If collapse requires shear through the body of the pillar (Figure 5) than for a 15° friction angle, kinematic failure cannot develop for W/H greater than 3.73. A ratio of 4.8 implies a friction angle of 12°.

**Figure 4 - Summary of South African and Australian pillar collapse databases**

**Figure 5 - Kinematically acceptable mechanism for pillar collapse**

**Implications to pillar design**

Colwell et al. (1999) uses the Mark Bieniawski pillar strength equation in an empirical method for determining the requirements for tailgate roof support. Seedsman (2001) has argued that the success of the method is related to onset of yield in the pillar leading to de-stressing of the roadways. The Mark Bieniawski equation may be a relationship for the onset of yield and not ultimate strength. The University of NSW method (Galvin, et al., 1999) should not be used for chain pillars.
The University of NSW method is relied on by subsidence regulars searching for long term stable pillars. The method is conservative for pillars with W/H greater than 4.8. For subsidence, the design issue becomes one of considering both collapse and allowable deformations. The empirical data indicates collapse will not happen for W/H ratios greater than 4.8. There will be additional deformations. Figure 6 compares two approaches to pillar design for subsidence control: in both cases the pillars would have long term stability, but the design for 300 m depth of cover based on a W/H ratio of 5: would have a 12.5 m pillar with 52% reserve recovery versus a 22.1 m pillar with 36% recovery. There would be major improvements in mining rates if a place change system could be adopted.

![Figure 6 - Pillar width, extraction ratio, and surface subsidence (posted numbers in mm) for a bord and pillar operation in a 2.5 m thick seam for two different definitions of long term stability](image)

**CONCLUSIONS**

The strength of the pillar system will be the strength of the weakest unit. The floor should always be characterised and the potential instability checked with the simple thin layer equation. Expert/more detail advice should be sought if the factors of safety are less than about 2.0 for greenfield sites or less than 1.5 where there is some precedent practice.

This is a remarkably simple test for weak floor (and by implication weak roof) that should be standard in every geotechnical toolbox. Its limitations are overridden by the simplicity of the calculation and the ability to do sensitivity studies. There is no justification for not collecting the data which will need to include a component of core or test pitting so as to check for thin very low strength layers.

Pillar collapse databases need to be exposed to this simple objective test. Currently, there is no basis for UNSW pillar design methodology for pillars with width to height ratio greater than 4.8. The method is massively conservative and probably results in unnecessary sterilisation of coal. Only the power relationship should be used, and not extrapolated beyond a W/H of 4.8. More research is required on pillar performance and especially the definitions of collapse, failure, and deformation.

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ANALYSING THE EFFECTIVENESS OF THE 1750 TONNE SHIELDS AT MORANBAH NORTH MINE

Kelly Martin¹, Mehmet S Kizil¹ and Ismet Canbulat²

ABSTRACT: Moranbah North Mine has a challenging geotechnical environment that historically resulted in cavity formation on the longwall face with its an associated reduction in productivity. Due to the complex geology at the mine, the increased depths of cover in future panels and the aging of the previous 980 t shields, longwall face stability became a concern. In order to ensure effective strata control in future panels, a new set of 1750 t powered supports were installed into the longwall 108 panel in 2009. These shields were, and still remain, the highest rated capacity shields in the world. This paper presents the results of an investigation into the effectiveness and necessity of the new powered supports. The investigation was undertaken in the form of a comparative analysis to determine the relative effectiveness of the two sets of different capacity shields by analysing the performance of the shields in panels that were directly adjacent and, subsequently, subject to similar conditions. Anglo American has plans to commence two additional longwall mining operations in the same region and the outcome of the investigation will allow the suitability of the larger capacity shields to be determined for the future operations.

INTRODUCTION

Strata control is a fundamental issue in underground coal mining as strata that is destabilised or has insufficient support can lead to events such as roof falls and formation of cavities. The primary area of concern is at the longwall face due to the face being subject to a dynamic, complex and constantly changing stress field. Advancement in mining technology has allowed for increases in mining depths, face heights and face lengths which, in turn, have resulted in the progressive increase of the maximum available powered supports.

In 2009, Moranbah North Mine (MNM) installed a new face of 1750 t longwall shields into the start of the 108 panel in order to combat various strata issues encountered at the mine including weak roof, overlying massive strata and increasing depths of cover. The 1750 t shields replaced 980 tonne shields due to concerns about their aging and insufficient support capacity.

An investigation was undertaken to determine the effectiveness and necessity for the new 1750 t shields. A comparison was made between LW108 and LW107 so that the relative effectiveness of the 1750 t shields could be determined. The two panels are directly adjacent to each other and were subject to similar conditions, hence making them ideal for a comparative assessment. The factors investigated included:

- Shield leg pressures;
- Cavity occurrences;
- Lost time due to strata control issues;
- Lost time due to shield issues;
- Time spent at or above yield pressure; and
- Shield performance in geological hazard zones.

POWERED SUPPORTS AND FACTORS AFFECTING FACE STABILITY

Longwall supports are designed to confine and control the fractured roof as the shearer cuts the coal. The supports must laterally confine high angled fractures to prevent dropout in front of the canopy. In order to improve the integrity of the immediate roof and to create a goaf break off line, the supports must...
also vertically confine the fractured strata to allow the transfer of the vertical stress into the roof. The creation of the goaf break off line behind the supports is important to prevent the caving mechanism progressing over the canopy towards the face. The effectiveness of the shields in controlling the caving environment about the face, therefore, depends on the ability of the shields to transmit stresses into the roof strata (Gale, 2009). There are numerous factors which can affect face stability with the factors that were considered in this investigation including:

- Set and yield pressures;
- Periodic weighting;
- Production delays; and
- Depth of cover.

Set and yield pressures

Hydraulic fluid is pumped into the leg chamber as the shield is set against the roof and the setting valve is closed and the fluid gets locked in. The pressure of the fluid at which the valve is closed is the working pressure of the pump and is referred to as the setting pressure (Deb, et al., 2000). Once the support has been set, roof convergence will occur over time and the fluid in the leg will be compressed, the fluid pressure will increase and the leg cylinder diameter will expand until the yield pressure is reached. Once the yield pressure is reached, the leg will yield as designed.

The yield pressure is the maximum pressure allowed in the bottom stage of the leg cylinder before the hydraulic cylinder and the pistons become compromised (Mitchell, 2009). During yielding, hydraulic fluid is released from the leg cylinder and the leg subsequently lowers a small amount during the yield event until the resetting pressure of the valve is reached. The resetting pressure will determine the amount of fluid lost and the amount of closure. Yield valves typically have a resetting pressure of 90% or higher, which equates to a 10% decrease in pressure before the valves closes (Barczak and Tadolini, 2007). The leg pressure will then gradually increase again due to continued roof convergence until the yield pressure is again reached and the cycle is repeated (Mitchell, 2009).

Roof deterioration between the canopy tip and the longwall face generally occurs after more than three yield events, with the severity of the deterioration increasing with the number of yields (Trueman, et al., 2010). A shield with sufficient support capacity should not be in yield more than 5-10% of the time.

Periodic weighting

Periodic weighting of the powered supports occurs when the longwall face is subjected to recurring cycles of overhang and breakage of strong strata in the immediate and main roof. The strong strata tends to cantilever over the goaf, resulting in periodic weighting of the supports. The frequency and intensity of periodic weighting is a function of the roof strength and thickness, the characteristics of the goaf, the distance of the strong stratum from the seam and the frequency of jointing. The periodic weighting peak occurs immediately before the caving of the strong stratum when the length of the cantilevered strata into the goaf is at a maximum (Agapito, et al., 1998).

Support overloading as a direct result of periodic weighting can result in yielding of the supports. Supports are most likely to yield as a result of period weighting at the peaks of the periodic weighting cycle with yield events also being probable during periodic weighting intervals when cycle times are relatively long. Multiple yield events in a single load cycle have been shown to cause roof control problems and are typically indicative of supports that are being periodically overloaded (Trueman, et al., 2010).

Production delays

The results of an analysis of real-time shield pressures conducted by Deb et al. (2000) showed that leg pressure increases rapidly within the first few hours after longwall downtime and then increases more gradually. When the cutting operation first experiences a delay, elastic or elasto-plastic deformation may occur in the roof which will result in roof-to-floor convergence and will cause a rapid increase in loading on the shield. Once the roof has settled onto the support canopy, the loading on the shield may gradually increase due to the shields response to creep deformation of the roof. This pressure
increase can be large enough to cause shield yielding within hours of the longwall ceasing operation (Deb, et al., 2000). Roof cavities as well as any strata stability issues may, therefore, be a direct result of production delays and not due to insufficient support capacity.

**Depth of cover**

Medhurst (2005) stated that although increasing cover depth generally results in a higher shield loading, modern capacity shields are sufficient to adequately control the roof in deep longwalls with everything else being equal. Hill (2006) also claims that the rate of powered support convergence tends to increase with increasing depth which suggests that at increased depths the supports approach yield more rapidly and ground conditions will tend to deteriorate with all other factors being equal. Hill (2006) suggests that the main cause of this effect would be the higher vertical abutment loading.

A geotechnical assessment conducted at MNM (Medhurst, 2006) determined that a combination of overlying massive strata, weak immediate roof and the presence of a rider seam in some areas of the mine resulted in the longwall face being continually operated at its limit. The assessment determined that under such working conditions, there existed little room for error and, as such, the impact of increasing depth of cover in future panels needed to be taken into account as the conditions were predicted to become more arduous at greater depths. This was a vital issue with respect to this investigation as one of the principle reasons for the shield upgrade at MNM was the increasing depths of cover due to the concern that the older shields had insufficient support capacity to cope with the additional stresses at the increased depths.

**MORANBAH NORTH MINE GEOLOGICAL AND SHIELD INFORMATION**

The geological conditions and rock mass characteristics at a mine site have a significant effect on the overall stability of the mine and the required support capacities. In order to fully assess the effectiveness of the new 1750 t shields, the relevant geological conditions needed to be assessed. The factors that were considered include:

- Lithology;
- Coal seams and ply splits;
- Depth of cover; and
- Faults.

**Depth of cover**

At MNM the seam dips to the east at 3 to 5° and the depth of cover increases to the east with LW108 being situated at approximately 300 m depth. Future panels will increase to depths exceeding 400 m. This increasing depth of cover was one of the principle reasons for the purchase and installation of the new 1750 t shields with depths of more than 220 m showing an increase in cyclic loading and face cavities. The mine layout with depth contours can be seen in Figure 1.

![Figure 1 - Moranbah North Mine layout and depth contours](image-url)
Sandstone channels

The presence of strong strata in the immediate or main roof such as sandstone channels can lead to periodic weighting events. The geology at MNM consists of several such sandstone channels which have led to numerous weighting events resulting in the yielding of the shields. The experienced gained at MNM indicated that in general, only the sandstone channels of more than 5 m in thickness and more than 30 MPa in strength will result in support issues (Laws, 2011). There are three main sandstone channels within the overburden at MNM that are of concern, namely, MP/MR20, MP/MR42 and MP/MR41. The sandstone channels generally also cause normal faults to develop directly below the channels.

Faults

Faults which exist in the immediate roof and overlying strata can impact significantly on ground stability as they can act as planes of weakness leading to excessive fracturing and roof falls. A large strike slip fault is present at MNM that runs approximately parallel to the working face at the southern end of the panels and has caused numerous support issues (Laws, 2011).

Seam Ply 1-2 split

There is a section of the mine where the seam Ply one splits away from Ply two by a distance ranging from 0.2 to 1 m. The area between the main seam and the top ply consists mainly of a weak siltstone material and, subsequently, due to the increasing area of this material as a result of the ply split, the strength of the immediate roof is significantly decreased. Ply one is completely removed from the GM seam thickness when the Ply one to two parting thickens to over 0.2 m (Laws, 2011). This decrease in roof strength results in a reduced ability to accommodate any horizon control issues.

Goonyella Middle Rider Seam split

The Goonyella Middle Rider (GMR) Seam splits off the top of the Goonyella Middle (GM) Seam towards the south and east sides of the lease area. When the split is between 1-2 m thick, major roof control issues have historically been experienced. When the GMR seam is coalesced with the main roof, the material in the interburden has an approximate strength of 5 MPa and consists of a mudstone/siltstone laminate interspersed with bedding plane shears. As the GMR split increases, the thickness of this weak zone also increases and results in the supports being unable to provide adequate confinement. As the GMR splits away to a distance of more than 2 m, however, the material strength increases and the GMR split becomes less of a geotechnical hazard (Laws, 2011).

1750 t shields

A feasibility study conducted in 2006 (Medhurst, 2006) concluded that due to the combination of overlying massive strata, weak immediate roof, the presence of the GMR rider seam and increasing depths of cover, the shields being utilised at that time (with a capacity of 980 t) would provide inadequate ground support for future panels as they were already constantly operated at their limit. The study furthermore concluded that in order to provide adequate roof support and stability for deeper panels, 1750 t capacity shields with a width of 2 m would be required. The project was approved and 151 1750 t capacity shields were installed into LW108 in 2009.

Longwall visual analysis software

Longwall Visual Analysis (LVA) software collects and stores real time data from the longwall shield legs. The leg pressures of every shield installed on a longwall face are recorded every minute as mining progresses. The software has a number of functions that are useful in determining the effectiveness of the shields with the Time Weighted Average Pressure (TWAP) function being used for this investigation. The TWAP function calculates the average leg pressures for each leg of each shield for every load cycle. The average pressure is calculated from when the shields are initially set to when the shields release at the end of the load cycle before advancement. Each leg pressure value is recorded against the corresponding chainage value.
DATA ANALYSIS

Longwall sections used in analysis

In order for the comparative assessment to be as accurate as possible, only the sections of the longwalls which were directly parallel to each other were used. As LW108 was significantly longer, this required starting the analysis of LW108 at the install point of LW107. Analysing sections outside of this region would have resulted in inaccurate results. This process was necessary for a number of reasons including:

- Commissioning of the new longwall gear resulted in a very slow start with some shifts only accomplishing a couple of shears;
- There were problems initially with loading the coal onto the AFC which led to the inclination of the AFC and significant horizon control issues; and
- In and around the LW107 install road, LW108 encountered a double stress notch which caused additional delays due to the requirement of additional support.

For the sections being analysed, LVA data was unavailable or corrupt for the start of LW107 and the end of LW108. As such, when this data was required for the analysis, particularly for the analysis of shield performance in geological hazard zones and lost time, deputy delay reports were used to find the required information.

Longwall visual analysis data

As LVA outputs values are recorded on an hourly basis, for each individual shield, multiple leg pressure values were recorded against the same chainage value. In order to condense the data into a more manageable data set whilst still accurately representing the pressure values for each shield, an Excel macro was used to sort the data. The macro sorted the data by taking an average of all the pressure values for a single shield with the same chainage values to give a single pressure value.

After the data had been sorted, the data was then condensed into three columns representing easting, northing and pressure values. In order for accurate coordinates to be assigned that would represent the real-time location of the chainage values according to the mine plan, the shield canopy widths and the angle of the longwall panels had to be taken into account. For the canopy widths, the averaged pressure values were taken to be located at the centre of each shield so, subsequently, the centre to centre distance of each shield had to be added onto each x-coordinate.

In order to allow the data to be aligned accurately with the longwall panels on the mine plan, the coordinates needed to be altered to allow the bottom left hand corner of each LVA data set to be aligned with the right hand corner of each longwall panel at the point in each panel which corresponds to the start of the panel section being investigated. These points can be seen in Figure 2. Using these identified points, the macro adjusted the assigned coordinate values by adding the distance along the length or width of the longwall panels according to the points of rotation. Since the longwall panels are positioned at an angle of approximately 10° west of north, 100° was used as the point of rotation.

![Figure 2 - Data analysis start points and points used for LVA data rotation](image-url)
Leg pressure contours

Once the manipulation of the LVA data was complete, the data was imported into Surfer and contour maps were created for each longwall panel. The red areas represent low pressure areas and are indicative of the presence of cavities. The contour maps for LW107 and LW108 can be seen in Figure 3. The contour maps were then exported into AutoCAD and overlayed onto the mine plan.

![Figure 3 - Leg pressure contours with low pressure regions indicated in dark or red areas](image)

Geological hazard map

Once the leg pressure contours had been overlayed onto the mine plan, mine geological data was used to create a Geological Hazard Map (GHM) so that shield performance in high geological hazard zones could be evaluated and compared. The GHM can be seen in Figure 4. The geological features that were included in the GHM include the following:

- The GM1 - GM2 Ply split zone of over 0.2 m parting thickness;
- The GMR split zone between 1 m and 2 m;
- Major faults; and
- Potential weighting zones.

![Figure 4 - Geological hazard map with pressure contours](image)
Panel delays due to cavities and shield issues

In order to determine the effectiveness of the shields, the amount of time lost due to cavities and shield issues were also analysed to give a more quantitative result. Pivot tables were created in Microsoft Excel using the deputy delay reports for each panel to identify the time lost. The results of the analysis were cross-checked with the mine geotechnical engineers to ensure results were as accurate as possible and were reflective of actual circumstances.

Eliminating data inaccuracies

Prior to the analysis of the LVA data, all data which could result in inaccurate results had to be eliminated. The raw LVA data had to be filtered to ensure there were no zero values present in the data set. This process was necessary as the presence of zeros in the LVA data caused inaccuracies in the manipulated data results as the zero values significantly lowered the averaged pressure values and caused the generated contour maps to indicate low pressure regions where, in reality, none may have existed. The presence of zeros in the data set was attributed to a number of factors; shield mechanical issues, shield electrical issues and issues with LVA.

The rows in the raw data which contained missing values also had to be deleted before the data manipulation commenced as the data manipulation process is such that any cells with no values are automatically assigned zero values. This similarly caused problems in the generated contour maps.

RESULTS AND DISCUSSION

Shield effectiveness determined from leg pressure contours

The generated leg pressure contours show low pressure regions of pressures less than 250 b in dark/red areas. These low pressure regions indicate the presence of cavities and, as such, highlight the shields general performance throughout the panel. From Figure 3 it can be seen that LW107 had significantly more low pressure regions than LW108. By cross-checking the dates corresponding to the locations of the low pressure regions with deputy delay reports, the number of low pressure regions on the contour maps which resulted in lost time due to cavities was able to be determined.

It was found that for LW108 there were only six cavity occurrences which resulted in lost time with four of these cavities occurring in the tailgate. For LW107, however, it was found that there were 38 cavities resulting in lost time with four of these cavities occurring in the tailgate.

The remaining low pressure regions which were not found to have resulted in lost time due to cavity control can be attributed to shield mechanical issues or shield electrical issues. It can be seen from the determined values that LW108 performed significantly better in terms of preventing cavity occurrences than LW107. A summary of the results can be seen in Table 1.

<table>
<thead>
<tr>
<th>Low Pressure Regions</th>
<th>LW107</th>
<th>LW108</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total</td>
<td>87</td>
<td>27</td>
</tr>
<tr>
<td>With lost time</td>
<td>38</td>
<td>6</td>
</tr>
<tr>
<td>With lost time in the tailgate</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td>With lost time in the face</td>
<td>34</td>
<td>2</td>
</tr>
</tbody>
</table>

Shield performance in geological hazard zones

By analysing the leg pressure contours in correlation with the constructed GHM, the performance of both sets of shields in high hazard zones was compared and analysed. As can be seen from Figure 4, the end of LW108 and start of LW107 were missing from the analysis due to the unavailability of the data. As such, for the purpose of assessing shield performance in high hazard zones, deputy delay reports were used to identify any cavity occurrences in the regions with missing data. The actual number of cavity occurrences in these circumstances could not be explicitly stated due to the deputy delay reports not stating explicitly whether the delays due to cavities recorded on consecutive days were due to a continuation of a single cavity or due to multiple separate cavities.
Figure 4 shows that the LW108 shields performed significantly better in terms of strata control in all high hazard regions with the ply split zone appearing to pose the most difficulty for the shields. A summary of the results can be seen in Table 2.

### Table 2 - Shield performance in geological hazard zones

<table>
<thead>
<tr>
<th>Hazard Zone</th>
<th>Number of Low Pressure Regions</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>LW107</td>
</tr>
<tr>
<td>Fault zone</td>
<td>multiple</td>
</tr>
<tr>
<td>Ply split zone</td>
<td>multiple</td>
</tr>
<tr>
<td>GMR split zone</td>
<td>9</td>
</tr>
<tr>
<td>GMR split and weighting zone</td>
<td>6</td>
</tr>
<tr>
<td>Weighting zone</td>
<td>41</td>
</tr>
</tbody>
</table>

**Time lost due to cavities**

Using deputy delay reports, the amount of time lost due to cavity control issues in each panel was determined and subsequently compared. It was found that LW107 had more hours lost due to cavities than LW108 with LW107 losing 170 h while LW108 lost 141 h. When taking into account the significant difference in cavity occurrences in each panel, the difference in lost time is not as large as would be expected.

Further filtering of the deputy delay reports in correlation with mine site geotechnical advice, however, allowed the total cavity occurrences for each panel to be separated into tailgate cavities and longwall face cavities. This was a necessary process as the effects of the double stress notch encountered in LW108 extended to approximately 100 m or one cut-through past the install face of LW107. The effects of the stress notch resulted in major tailgate support issues in LW108 and required additional tailgate support to be installed which caused significant delays. Large unmapped faults which ran perpendicular to the longwall face and parallel to the tailgate were also encountered in LW108 which led to stoppages due to tailgate support issues. Additional delays were also encountered in LW108 due to gas levels, not experienced in LW107, preventing immediate entry to the tailgate for the installation of secondary support for the longwall retreat.

Taking into account the intense tailgate conditions unique to LW108, and in order to provide an accurate comparative assessment, the amount of lost time due to cavities in the tailgate for both panels should not be included in the final assessment of the shield effectiveness. The final assessment will, therefore, focus only on strata control issues relating directly to the longwall face. A summary of the results can be seen in Figure 5.

![Figure 5 - Time lost due to strata control and shield issues](image)

**Time lost due to shield issues**

Again using the deputy delay reports, the amount of time lost due to shield issues in each panel was able to be determined and subsequently compared. The shield issues incorporate all mechanical, electrical and operational issues. The lost time due to shield issues in both panels can be seen in Figure 5.

As can be seen from Figure 5, LW107 had 603 h lost due to shield issues which is almost double the 314 h lost in LW108. While the aging duty of the LW107 shields likely impacted on the lost time in LW107, this effect potentially would have been equalised by the lost time due to the commissioning of
the new shields in LW108 and the various teething issues involved. The large amount of time lost due to shield issues in LW107 suggests that the shields were being overloaded due to operating outside of their capacity.

**Shield effectiveness determined from LVA data**

If a shield is constantly operated in yield it is reasonable to accept that the shield support capacity is inadequate for the working conditions. As such, in order to determine the effectiveness of the shields, and to determine if shields with a capacity as large as 1750 tonnes were necessary, the amount of time the shields were operating in yield should be assessed. In this case, insufficient data was available to calculate total time spent in yield percentage values and, subsequently, the percentage of time that the shields spent operating at or above the yield pressure was calculated instead.

The LVA data was also used to determine the percentage of time that the shields were operating under 250 b. As already stated, areas where the leg pressure values are less than 250 b is indicative of the presence of cavities. As such, by determining the percentage of time the shields spent operating under this pressure, the percentage of time that cavities were encountered for the panel in question is effectively being determined. A summary of the results can be seen in Figure 6.

![Figure 6 - Shield effectiveness determined from LVA data](image)

The results of the analysis show that the shields in LW108 spent an insignificant amount of time operating at or above the yield pressure value while the LW107 shields had a significantly higher result. Similarly, the amount of time that the shields in LW107 spent operating under 250 b was significantly higher than that experienced by the LW108 shields. Further analysis of shield pressures at different stages in the panel showed the following:

- The LW107 shields directly adjacent to cavity zones consistently went into yield; and
- Even around large cavity zones, the LW108 shields did not go into yield.

These trends can be seen in Figure 7 where the leg pressures have been plotted against the shield numbers.

![Figure 7 - LW107 (left) and LW108 (right) shield performances around cavity zones](image)

Figure 7 shows that LW107 had constant fluctuations in leg pressure values with several shields operating at or above the yield pressure especially around cavity zones, while the LW108 leg pressures...
remained relatively constant across the panel, excluding the cavity zones. It also shows that even when increased loading occurred on shields directly adjacent to cavity zones, the leg pressures in LW108 remained well below the yield pressure value. These results indicate that the shields in LW108 were more effective in terms of strata control than LW107 and, similarly, had shield capacities that were more suited to the conditions.

**Shield suitability to mining conditions**

In order to assess the suitability of the shields for the mining conditions, the total amount of time that the shields spent in yield can be utilised. It should be reiterated that the percentage values previously calculated give the percentage of time the shields operated at or above the yield pressure and not the actual total time the shields spent operating in yield. The actual time the shields spent in yield should include not only the time spent above the yield pressure, but also the time spent in yield as the leg pressure was released until the reset pressure was reached.

Due to data being unavailable with the depth of detail required for the calculation, the actual total time the shields spent in yield could not be determined. However, as this value would be significantly higher than that calculated for the time the shields spent above the yield pressure, the previously calculated percentage values can still be used to determine if the shields were suitable for the mining conditions.

Based on an acceptable yield percentage value of 5%, it can be said that the 980 t shields in LW107 were not suited for the mining conditions. As the percentage of time that the shields spent operating at or above the yield pressure was calculated to be 6.5%, the actual total percentage of time spent in yield would have been significantly higher, hence indicating that the shields were working outside of their capacity. This finding is supported by the large amount of lost production time in LW107, calculated as 148 h, due to face cavities.

Similarly, as the percentage value for the time the shields spent operating at or above the yield pressure for LW108 was calculated to be 0.6%, the actual total percentage of time spent in yield would have been significantly higher. Given the low calculated percentage value, however, it is feasible to say that the shields in LW108 were more than adequate for the mining conditions. This result is in accordance with the low number of lost production hours, calculated as 20 h, due to face cavities in the panel. It is therefore feasible to assume that in future panels of increased depths of over 500 m, the 1750 t shields will have an appropriate capacity, provided that the geotechnical environment will be similar to that experienced in LW 108.

**CONCLUSIONS**

Moranbah North Mine has a challenging geotechnical environment that historically resulted in cavity formation on the longwall face and unplanned downtimes with a subsequent reduction in productivity. Due to the equally complex geology of the future panels at increased depths of cover, the previous roof support issues gave rise to concerns about roof stability in future panels. It was determined that 1 750 t shields would be required in future panels instead of the previous 980 t shields to combat the various strata issues. As shields of this capacity did not already exist, the shields had to be custom built and still currently remain the highest capacity shields in the world.

Overall, taking into account the results of all aspects of the analysis, it is feasible to say that the 1 750 t shields were more than effective in terms of strata control and the support capacity will be well suited to the mining conditions that will be experienced at greater depths.

**ACKNOWLEDGEMENTS**

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BACK ANALYSIS OF ROOF CLASSIFICATION AND ROOF SUPPORT SYSTEMS AT KESTREL NORTH

Sabine Stam¹, Glen Guy² and Nick Gordon³

ABSTRACT: Kestrel Mine is a Rio Tinto owned underground longwall operation that mines the German Creek coal seam in Queensland’s Bowen Basin. Kestrel Mine can be separated into two parts North and South. Kestrel North has been in operation since 1990 (called Gordonstone at the time) while Kestrel South is a comparatively new mine having begun in-seam development in the first quarter of 2011. Over the recent history of Kestrel North several methodologies have been employed to characterise the roof and floor conditions, with a view to optimising the roof support design system and process.

The objective of this study was to review various roof classification systems against actual conditions encountered during extraction of coal from the 300 series longwall panels at Kestrel North. The aim was to determine what systems would work well in the deeper Kestrel South environment. The study also reviewed different UCS sonic relationships in use and derived a new correlation for the entire Kestrel area.

The back-analysis for the primary support was conducted by comparing the actual conditions and installed bolting patterns versus the rock mass conditions predicted using a variety of different roof classification systems. For secondary roof behaviour, extensometer data was used to review roof performance. The systems reviewed were the Roof Strength Index, sonic derived UCS and Roof Mass Rating.

The study confirmed that UCS is a good first predictor for the primary roof conditions, whereas the Roof Strength Index showed the best correlation with the secondary roof conditions. It is inferred that formation of a beam in the primary support horizon is more closely related to rock strength, compared to the secondary support horizon where the influence of the stress regime appears more critical.

INTRODUCTION

Currently Rio Tinto is constructing the Kestrel Mine Extension (KME or “Kestrel South”), located to the south and deeper than the current Kestrel operations. There is an opportunity to back analyse the various roof classification systems at Kestrel North in order to provide Kestrel South with the best system to predict upcoming roof conditions. This will allow optimisation of the roof support systems and improve the geotechnical input into the Life of Mine (LOM) model. The layout of Kestrel North and South areas is shown in Figure 1.

In addition a full review of the UCS vs. Sonic relationship has been carried out to ensure the most appropriate equation is used in the future.

BACKGROUND

Various roof classification systems are being or have been used at Kestrel.

There are many classification systems available in the underground coal mining industry, but since Kestrel is in an environment where, with the exception of bedding laminations there are not many significant rock mass discontinuities. This presents a comparatively geotechnically benign environment for which many of these systems have been determined not to be practical for Kestrel. This does not imply that those systems are incorrect or do not work.

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² RTCA Principal Geotechnical Engineer
³ Gordon Geotechniques Pty Ltd.
It is noted that rock mass classification systems are designed to assign a value to a rock mass competence. This is a relatively generic number and these systems do not always take external factors like mining direction and anisotropy into account. They should be used in conjunction with local experience and knowledge. They are not the sole input for engineering design.

ROCK MASS CLASSIFICATION SYSTEMS

There are three main systems that are currently used in various applications at Kestrel.

Roof mass rating

Strata Control Technology (SCT) has developed roof and floor class maps or Roof Mass Rating (SCT_RMR) that are being used by mine planning staff (SCT, 2007). It originates from work by Coffey Partners (1995) in the Gordonstone period and used by the site geotechnical engineer at the time Lawrence (1997) and continued at Kestrel by Gordon (2000) as shown in Figure 2. The roof and floor class maps produced by SCT are a refinement of this original work. The primary and secondary roof and floor ratings developed were based not only on strength, but also on the number of bedding laminations and the presence of weaker layers. It was identified that as well as strength, these weaker layers and bedding laminations play a role in roof behaviour. It should be noted that this rating system was first utilised in the 100 and 200 Series area of the mine where the range in depth of cover was relatively consistent, mostly between 210 m and 260 m.

Figure 2 - Comparison of roof ratings (Gordon, 2000)

Roof strength index

The Roof Strength Index (RSI) is a numerical value developed by Gordon and Tembo (2005) using sonic derived UCS values and depth of cover. This system has been used in a number of areas of the Kestrel North mine to explain roof conditions.
Sonic velocity derived UCS

The development hazard plans at Kestrel have historically used sonic derived UCS values per 100 m (per ‘cut through’) in 2 m roof intervals to predict roof conditions. Primary and Secondary support plans refer to this information as part of the design process. This system has been in place for all the 300 Series development roadways and operational staff are trained in using these hazard plans.

Other classification systems

Coal mine roof rating (CMRR)

CMRR (Mark and Molinda, 2003) assumes that the structural competence of coal mine roof is determined primarily by discontinuities that weaken the rock structure. CMRR is specifically designed for bedded coal measure rocks, concentrating on the bolted interval (and its ability to form a stable mine roof). This rating system is applicable to all coal mine roof types. Inputs into the CMRR calculations comprise the UCS of the intact rock, the spacing and persistence of discontinuities, the cohesion and roughness of discontinuities and the presence of groundwater and the moisture sensitivity of the rock.

Work undertaken at Kestrel by Colwell as part of the ACARP Rib Support project (Colwell, 2004) describes the CMRR at Kestrel as weak with a CMRR value varying between 30 and 45. Geotechnical staff at Kestrel at the time found that the CMRR derived did not give a better prediction of roof conditions compared to the sonic derived UCS values. It is not implied the CMRR does not work at Kestrel as a predictive tool but data required to determine the CMRR was not routinely collected. The system has not been used operationally at Kestrel North due to the reliance on the comparatively simpler sonic to UCS data.

Geophysical strata rating (GSR)

The GSR has been developed through various ACARP projects by Hatherly et al. (2004; 2009). It is based on geophysical logging where the p-wave velocity is the main input, but includes inputs from porosity, clay content and depth.

GSR has not been used operationally at Kestrel as a roof condition prediction tool, but data from Kestrel has been a significant input in the development of the system. As part of a test case four holes were used to create a GSR on the Kestrel lease. Given the sparse coverage it is not sufficient to compare this method with the other available methods.

SONIC TO UCS CONVERSION

In the Australian mining industry the exponential relationship between geophysically derived sonic velocity and inferred UCS values as proposed by McNally (1987) is widely accepted (Equation 1). It is also recognised that there is a specific relationship for every site.

\[ UCS = 1000 \times e^{-0.035 \times \text{sonic velocity (ms}^{-1})} \]  

At Kestrel (and Kestrel South) three separate equations have been used on different occasions. This is predominantly due to data ownership and the different consultant involved in various projects. A summary of these correlations is presented in Table 1

In comparing the three equations in Figure 3 to the original McNally equation it can be seen that:

- in the lower strength range (0-15 MPa) the formulae correlate quite well;
- in the stronger ranges the Seedsman correlation is more conservative;
- The Geotek equation is the least conservative but probably reflects the variety of material tested;
- All three equations show a significant difference from the McNally formula;
- It is interesting that the Kestrel data sets are different, except for three holes that have been used by both SCT and SGPL.
Table 1 - Summary of Kestrel specific UCS sonic correlations

<table>
<thead>
<tr>
<th>Author</th>
<th>Formula</th>
<th>Data Points</th>
<th>$R^2$</th>
<th>Comments</th>
</tr>
</thead>
<tbody>
<tr>
<td>SGPL</td>
<td>$UCS = 0.3196 * e^{0.0012 \cdot \text{sonic velocity} \left(\frac{m}{s}\right)}$</td>
<td>131 52 32</td>
<td>0.57</td>
<td>32 unknown or Kestrel West data points rejected for 2010 set</td>
</tr>
<tr>
<td>SCT</td>
<td>$UCS = 0.2769 * e^{0.0013 \cdot \text{sonic velocity} \left(\frac{m}{s}\right)}$</td>
<td>12 117 0</td>
<td>0.69</td>
<td>2 outliers rejected by both SCT and 2010 set</td>
</tr>
<tr>
<td>Geotek</td>
<td>$UCS = 0.104 * e^{0.0016 \cdot \text{sonic velocity} \left(\frac{m}{s}\right)}$</td>
<td>131 0 0</td>
<td>0.84</td>
<td>All holes located in Kestrel South drift area</td>
</tr>
</tbody>
</table>

Figure 3 - Comparison of various Kestrel Sonic vs UCS correlations

After reviewing the full dataset available, it was decided to remove any Kestrel West holes, as well as the holes that were drilled only for the Kestrel South drifts (Geotek, 2007). The Kestrel West holes were removed as they are relatively far removed from any current mining areas. The drift holes were removed as they spatially skew the dataset and samples taken from these holes were not necessarily targeted at the German Creek seam, but at shallower areas.

After undertaking this and combining datasets over Kestrel North and South a new site specific equation was developed (Equation 2).

$$UCS = 0.2512 * e^{0.0013 \cdot \text{sonic velocity} \left(\frac{m}{s}\right)}$$ (Equation 2)

Figure 4 - Comparison of various older equations with new 2011 equation

BACK ANALYSIS

The original back analysis of the three predictive systems was undertaken with the UCS equation that was used by the consultant responsible for the system at the time. With the new equation having been developed, the back analysis was then redone, to ensure its validity.
Primary support

The primary roof at Kestrel is classified as the 0-2 m of immediate roof above the German Creek Seam. Support is installed directly off the continuous miner and consists of either 6 or 8 x 2.1 m bolts per metre. When comparing the contour plots for the RSI, SCT_RMR and sonic derived UCS in Figures 5-7, it is clear that the SCT_RMR and UCS plots show a very similar overall pattern with competent and less competent areas highlighted. There are exceptions, for example RSI does not predict bad conditions in the western panels but it does show deterioration in the outbye areas of panels 303 to 308.

Figure 5 - Inferred UCS 0-2 m roof   Figure 6 - RSI 0-2 m roof   Figure 7 - SCT-RMR 0-2 m roof

When comparing the methods, the following support predictions have been used:

Table 2 - Support predictive systems suggested bolting density

<table>
<thead>
<tr>
<th>No of Bolts/m</th>
<th>Support Predictive System</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>RMR</td>
</tr>
<tr>
<td>4</td>
<td>N/A</td>
</tr>
<tr>
<td>6</td>
<td>1</td>
</tr>
<tr>
<td>8</td>
<td>4</td>
</tr>
</tbody>
</table>

For comparison of the systems it was assumed that the bolting pattern that was installed is regarded as the “pattern required for adequate stability”. When a system would have predicted (based on Table 2) more bolts than actually installed it is called “oversupported”. When the same pattern is used as predicted it is called “equal” and when a lighter pattern than installed was predicted it is called “undersupported”.

For the oversupported cases it is almost impossible to tell if in the cases where 8 bolts per metre were installed, that in fact six bolts may have been sufficient, apart from roof stability observations. The corollary is true that where six bolts per metre have been predicted and either six or eight bolts were installed. In these areas of oversupport Kestrel crews typically notify the geotechnical engineer that the roof conditions are good, with low levels of roof movement.

On this basis it was expected that the UCS rating system would more accurately predict the required primary support pattern than the other two systems, as this is what the original support plan would have been based on. Adjustments to the support patterns would only be made in significantly worse conditions or if conditions were so good that after an extended period of installing eight bolts per metre the pattern would have been dropped back to six bolts.

Figure 8 presents the comparison of the three systems prediction vs. actual.
Development rates have been reviewed to see if a correlation between the various systems and these rates exists. No correlation could be developed for the data analysed. This is in part due to the fact that development rates are influenced by many other factors including the experience of personnel, maintenance, the introduction of a monorail system, motivation, belt and machinery delays and of course strata control. To filter all these factors from the data would be a major project in itself.

It should be noted that with all the geological and geotechnical data interpretation, the accuracy is dependent on the spatial presence of the data. Borehole spacing at Kestrel varies from 400 m in general to less than 100 m in faulted areas. Lithology changes can occur fairly rapidly at Kestrel. It is noted that systems based on borehole data will not always be able to accurately predict these changes.

The overall outcome of the comparison is that in general the RSI would tend to oversupport in the deeper areas of the 300 series. The RMR system on the other hand would typically undersupport in the development of 303 to the 307 panel when this study was undertaken.

After development of the 2011 UCS correlation (Equation 2), the analysis was revised to compare the outcomes with the results using the equations developed previously.

As can be seen in Figure 9, the results do change slightly, but the overall result does not differ from the original data set. The most interesting finding is that all three systems do not improve significantly better with the new dataset. This is thought to be due to the fairly broad cut-off range for bolting pattern prediction.
SECONDARY SUPPORT

Methodology and assumptions

The secondary support methodology was not as straight forward as the primary support. No method is currently employed to give a direct prediction of what support to use.

As a rule Kestrel installs secondary support in all maingate areas. It is accepted that there will be areas in the mine where the secondary support density may not have needed to be as intense. From a risk perspective the current system maintains stable maingate conditions for longwall retreat. Only occasionally is the timing of tendon installation shortened due to adverse conditions on development.

Comparison of the various systems was therefore not a straightforward exercise especially as different support tendons have been installed.

Secondary support patterns at Kestrel have been designed by using a dead weight calculation. In this calculation the strength of the anchorage horizon is used to determine the anchorage length required to mobilise the full capacity of the tendon (Figure 10). The capacity of the support pattern is also calculated to ensure the dead weight can be supported.

![Figure 10 - Anchorage length outside roof softening zone versus UCS (SGPL, 2006)](image)

Depth of cover is not regarded in this method rather the strength the roof strata in the anchorage horizon. As weaker ground has been encountered in the 300 Series area the cable support has gradually increased from 6 m HITENS installed in 28 mm holes to 8 m Megastrands in 42-45 mm holes to ensure sufficient anchorage is available.

In order to compare the available predictive methods an attempt was made to compare production rates with predicted conditions, but the production rates are influenced by many factors non-strata related and therefore this exercise was deemed unsuccessful.

At Kestrel, roof movement is monitored with the use of “Clockits”. Clockits are installed at every intersection and every roadway with “unusual dimensions” such as drillers’ niches and driveheads. The anchors are installed at 2 m and 6 m to differentiate between primary and upper roof movement. To be able to reconcile the secondary support, the upper movement from the Clockits has been assessed.

Significant movement (>30 mm) in the 2-6 m region suggests the secondary support is only just sufficient in that location or additional support is required. It should be noted that the accuracy of Clockits is dependent on the quality of installation. The frequency of reading them will also influence the quality of the data. At Kestrel the reading on development happens on a very regular basis (since Maingate 303), but tailgate and maingate readings are not as regular once longwall production has commenced.

The maximum value of upper movement of all Clockits around a cut through was plotted in Figure 11. Data from longwall installation roadways and takeoff areas were not taken into account for this analysis. The support in these areas is designed with a different timeframe, roadway span and loading in mind.
Results

As discussed above, the prediction of secondary support patterns at Kestrel are not as clear and objective as the patterns for the primary roof horizon.

Maingate roadways

The secondary support comparison is based on the contour plots for the various systems, as well as the experiences of the author and other previous geotechnical engineers at Kestrel.

A few particular areas of interest are focussed on:

- Significant strata control issues occurred in the tailgate for LW303 around 16 cut through. Both RMR and RSI secondary roof plots indicate these as weaker areas. UCS does not recognise this as an area of particular concern.

- LW304 encountered serious issues regarding the maingate in 4-5ct where spiling and shot firing was required to continue producing. This area is most clearly identified in the RSI plot. In the UCS contour plot, this is an area where roof strength is reducing but it does not identify this exact area as weaker than the surrounding roadways. LW305 had serious issues in the tailgate in the same area.

- Again there were issues in the outbye areas with LW305 and 306. In both the tailgate and Maingate of these panels significant roof convergence was encountered requiring the installation of standing support. These areas are most clearly defined in the RSI plots, but UCS and RMR show some weaker areas as well (Figures 12-14).

All the previously discussed events were at relatively shallow depth of cover. Two significant incidents happened in the deeper areas.

- A roof fall occurred in a partly driven installation road where the roof was highly laminated. This fall was due to a lack of secondary support related to operational constraints. There was no incorrect design or unexpected weak conditions.

- A face fall occurred on LW305 at the start of the panel. This fall was entirely due to leg pressure issues on the longwall. This incident had no direct relation to strata conditions but was inferred to be an operational issue.
When comparing the last three plots with the Clockit data presented in Figure 11, the RSI plot has the best match regarding the shape of the contours. The RSI system is also the only method that does not predict the worst conditions in the panels up dip from 307. Clockit data and current underground experience confirms this.

**Installation roadways**

All installation roads are at the deeper parts of the 300 series. A separate comparison has been undertaken on strata conditions for the different installation roads. It is important to keep in mind that drivage strategies and different support patterns for these roads have changed over time. Drivage direction (up dip or down dip) and lifespan of these roadways differ significantly.

Secondary roof conditions for 300 series face road Clockit measurements are compared in Table 3. The most obvious point of comparison is that roof movement in the 308 face road was minimal (<5 mm) with a similar support pattern to 307. When looking at the classification systems the RSI is the only one that predicts 308-310 as the best conditions, where both sonic derived UCS and RMR do not show a significant improvement.

### Table 3 - Face road secondary roof comparison for 300 series

<table>
<thead>
<tr>
<th>Face road (7.5m)</th>
<th>301(1)</th>
<th>302</th>
<th>303</th>
<th>304</th>
<th>305</th>
<th>306</th>
<th>307</th>
<th>308</th>
<th>309</th>
<th>310</th>
</tr>
</thead>
<tbody>
<tr>
<td>RSI</td>
<td>3.5-4</td>
<td>3.5-4</td>
<td>3.5-4</td>
<td>3.5-4</td>
<td>3.5-4</td>
<td>3.5-4</td>
<td>4-4.5</td>
<td>4.5-5</td>
<td>4</td>
<td>3.5-4</td>
</tr>
<tr>
<td>RMR</td>
<td>3</td>
<td>2</td>
<td>2</td>
<td>3</td>
<td>3</td>
<td>3</td>
<td>2</td>
<td>3</td>
<td>4</td>
<td>4</td>
</tr>
<tr>
<td>Total average movement before widening</td>
<td>19mm</td>
<td>37mm</td>
<td>20mm</td>
<td>20mm</td>
<td>&lt;5mm</td>
<td>&lt;10mm</td>
<td>&lt;10mm</td>
<td>&lt;10mm</td>
<td>&lt;10mm</td>
<td>&lt;10mm</td>
</tr>
<tr>
<td>Total average movement after widening</td>
<td>47mm(2)</td>
<td>60mm(2)</td>
<td>19mm(2)</td>
<td>26mm</td>
<td>44mm</td>
<td>62mm</td>
<td>60mm</td>
<td>&lt;5mm</td>
<td>&lt;10mm</td>
<td>&lt;10mm</td>
</tr>
</tbody>
</table>

(1) 301 face road had a “bleeder road”;
(2) Support material and patterns were different than later face roads.

**Conclusion secondary support**

When comparing all the data discussed above, the RSI appears to give the most accurate prediction for secondary roof conditions. This is different from the primary roof outcome, where sonic derived UCS was the best prediction tool for immediate roof conditions.
The variation could lie in the fact that both support systems serve different functions. The primary support is installed initially to create a beam in the immediate, laminated roof. Whereas secondary support is installed to ensure the weight of the area softened by longwall abutment stress is securely anchored to the overlying strata. The depth of cover will obviously be proportional to the magnitude of stress.

Based on the results of the assessment the question is raised can the Roof Strength Index be used in a similar way as the sonic derived UCS for primary roof, predicting longer term roof conditions and consequently specify secondary support patterns. It is noted that this will be investigated by Kestrel South geotechnical staff once coal drivage has commenced and there is confirmation of roof conditions at greater depths of cover.

REFERENCES


METROPOLITAN MINE UNDERGROUND EMLACEMENT OF COAL REJECTS - A CASE STUDY

Greg Tarrant, Tim Gilroy, Gasper Sich and Dane Nielsen

ABSTRACT: Metropolitan Collieries Pty Ltd (MCPL), a wholly owned subsidiary of Peabody Energy Australia Pty Ltd, has developed a method to emplace coal mine rejects underground. The rejects are processed to form high density slurry which is pumped into underground workings. The technology will eliminate the transport of rejects by truck through the township of Helensburgh, thereby significantly reducing the environmental impacts of dust, noise, and visual amenity as well as improving road safety. MCPL conducted a range of investigations from initial laboratory studies, field trials and a pilot underground emplacement facility. Key flow characteristics of the high density slurry include a pumping distance of up to 8 km, non-settling; and a drained strength that eliminates the liquefaction risk.

INTRODUCTION

Metropolitan Mine is an underground longwall coal mine located in Helensburgh, midway between Wollongong and Sydney. Mined coal is washed on site to produce a high quality coking coal. The washing process separates the product coal from the waste rock, typically shales and siltstone that comprises 17% of the total mined material. This generates approximately 300 000 tonnes of rejects per annum.

A major environmental challenge for Metropolitan (and the industry more generally) is the disposal of coal rejects. The township of Helensburgh was founded in 1888 to provide housing and support services for the mine and for many years coal mine rejects were used to support town infrastructure such as construction of sporting fields. Later, the rejects were stored on the mine site, but storage space became exhausted in the mid 1990s. Prior to underground emplacement, all rejects were transported by truck to an emplacement facility at a disused washery at Glenlee, near Campbelltown. The only transport route to Glenlee, or any other external site, is through the main street of Helensburgh. This creates environmental issues of noise and dust, together with other issues such as road safety and visual amenity.

Peabody Energy Australia Pty Ltd took ownership of Metropolitan Mine in 2006 and recognised that future development would be limited unless an alternative to trucking rejects through town could be found. An investigation into the emplacement of coal mine rejects into abandoned underground workings by high density slurry was conducted.

This paper describes the laboratory, surface field investigations, and pilot underground plant leading to successful development of the emplacement method and presents the potential future applications of the technology.

METROPOLITAN FILL SPECIFICATIONS

The specifications for the emplacement of rejects by high density slurry were developed according to site specific aspects as outlined below:

- The Metropolitan Mine layout has a range of potential voids for filling. These include:
  - abandoned mine roadways accessible by mine personnel;
  - abandoned mine roadways inaccessible to mine personnel but accessible by fill depending on flow characteristics;
  - limited access to previously extracted longwall areas; and
  - future longwall areas.
the range of pumping distances required was between 500 and 8 000 m;

- liquefaction of the emplaced fill (for example by earthquake) was considered an unacceptable risk if uncontrolled flow could endanger mine personnel. The risk of bulkhead failure resulting in inrush that may endanger mine personnel was also considered an unacceptable risk;

- the risk of pipe blockage would need to be minimal;

- the risk of groundwater contamination after mine closure should be negligible;

- the environmental site constraints of noise and dust must not be exceeded;

- the Metropolitan surface footprint is extremely small, being constrained by topography which precluded a large fill preparation facility (no ball mills; no grinding facility; no dewatering facility; no batch plant; and no large pumps);

- water usage should be minimised;

- no introduction of additional material such as flyash;

- emplacement could not disrupt mine production; and

- the emplacement method should be cost effective compared with trucking off site ($12.50/t).

These site specific limitations drove the fill specifications, which include:

- a friction loss of less than 4 kPa/m was initially targeted to achieve the pumping distances required;

- low free water release, target moisture content of less than 30%;

- pumping capacity of at least 125 tph;

- continuous process - not a batch process;

- the particle size distribution would need to be achieved without using milling or grinding processes;

- an emplacement strength (UCS) of at least 100 kPa to eliminate the liquefaction risk in accordance with industry best practice (Le Roux, et al., 2004);

- a beach angle\(^1\) of 3 to 4° to facilitate flow into inaccessible areas;

- non-settling to minimise risk of pipe blockage, and

- additives must not be potentially harmful to the environment.

**INDUSTRY EXPERIENCE**

Emplacement of coal mine rejects had previously been conducted at Walsum Colliery in the Ruhr Valley, Germany using a goaf cavity filling technique developed by Deutsche MontanTechnologie (DMT) in conjunction with Ruhrkohle AG (Mez and Sill, 1992; Mez and Schauenberg, 1998). Walsum generated a pumpable paste from the fine tailings fraction of the coal mine rejects mixed together with flyash from external sources. The paste was pumped underground and emplaced behind the longwall supports in trailing pipes as shown in Figure 1. Unfortunately the colliery closed in 2006.

An investigation of other operations involving waste disposal in underground workings was conducted. This included:

- metalliferous tailings into stopes as cemented backfill at Lisheen Mine (Ireland) and St Ives Gold mine (Western Australia);

- waste product into old disused underground workings as low strength backfill such as Boulby Mine (UK);

\(^1\)Beach angle is a term borrowed from tailings dam impoundment. It refers to the slope created from settling of particles after discharge. Typically the ‘beach’ slopes from the discharge point to the middle of the impoundment. In the context of the Metrop backfill project, beach angle is the slope created from the settling of particles following discharge from the pipe - it (along with other factors) governs how far the fill will flow along a roadway from the discharge point before the discharge point becomes choked off.
flyash waste disposal into coal mine longwall goaves at Carbisulcis mine, Sardinia, Italy; and
use of backfill for spontaneous combustion and subsidence control using combinations of sand, coal reject, coal tailings and flyash into Polish coal mines such as Staszic Mine, Myslovice Mine, Pniowek Mine and Borynia Mine.

Figure 1 - Goaf filling method - Walsum Colliery (after Mez and Sill, 1992)

An investigation into long distance pumping with cementitious materials was conducted such as 3 km to 7 km flyash-cement slurries at mines such as Auguste Victoria Mine (Germany), 1 km dredged sludge-cement slurries for Haneda airport (Japan) and general concrete pumping technologies from Putzmeister and Schwing (Germany).

The outcomes of these investigations were:

- the cost of grinding and dewatering to achieve the same particle size distribution and moisture content as a metalliferous backfill plant was a major cost component and required significant real estate on the surface;
- a very fine particle size distribution used in metalliferous mine backfill which made for easy pumping would be a liquefaction risk without cement;
- chemical pumping aides were commonly achieving a 40% to 60% reduction in viscosity in both very fine pastes and coarse concretes;
- long distance pumping of coarse and fine particle size distributions was proven and established in various industries, and
- coal mine goaf and roadway disposal of hydraulic fill into coal mines was proven and established.

METROPOLITAN MINE REJECTS CHARACTERISTICS

The material characteristics of the rejects are summarised in Tables 1 and 2. The particle size distributions (PSD) of the various feeds individually and combined are shown in Figure 2.

Figure 3 indicates the PSD resulting from crushing of a sample in a combined Horizontal Shaft Impactor (HSI) and then Vertical Shaft Impactor (VSI). It was considered desirable to achieve a PSD that would not require a ball mill due to the limited space availability within the mine surface facilities. A ball mill would also require a de-watering process step, again requiring additional space and additional processing complexity. The total rejects raw feed is also shown in Figure 3 for comparison. The VSI/HSI PSD shown in Figure 3 provided an initial target PSD for laboratory and field studies. To put the trial mix into perspective, the HIS/VSI trial PSD plot is similar to a pumped concrete which is a mix that is 10 mm aggregate with sand and cement, which is a mix that is regularly pumped long distances in the construction industry.
Table 1 - Rejects material characteristics

<table>
<thead>
<tr>
<th></th>
<th>Density (kg/m$^3$)</th>
<th>Moisture (% w/w (as))</th>
<th>Hardgrove Grindability Index</th>
<th>Unconfined Compressive Strength (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coarse Reject</td>
<td>2000</td>
<td>6</td>
<td>62</td>
<td>32</td>
</tr>
<tr>
<td>TBS Reject</td>
<td>1700</td>
<td>19</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Flotation Tailings</td>
<td>1700</td>
<td>70</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

TBS stands for Teeter Bed Separator. It's a process during coal washing that uses an upward current of water to separate particles. Those particles that have a free settling rate equal to the upward current are held in a state of 'teeter'. What matters here is that the TBS process generates a high percentage of rejects that are sand (1 mm) sized particles.

Table 2 - Feed throughput and particle top size

<table>
<thead>
<tr>
<th></th>
<th>Nominal (tph)</th>
<th>Maximum (tph)</th>
<th>Top Size (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coarse Fraction</td>
<td>49.5</td>
<td>107.5</td>
<td>50.0</td>
</tr>
<tr>
<td>TBS</td>
<td>12.2</td>
<td>36.7</td>
<td>0.8 wedge wire</td>
</tr>
<tr>
<td>Flotation Tailings</td>
<td>14.5</td>
<td>29.0</td>
<td>0.35 wedge wire</td>
</tr>
</tbody>
</table>

LABORATORY STUDIES

MCPL commissioned two independent laboratories: Tunra Bulk Solids Handling Research Associates at Newcastle University and DMT laboratory in Germany to provide an initial assessment of flow characteristics.

Laboratory samples were prepared by crushing the coarse rejects to a top size of 5 mm and adding the TBS rejects and tailings according to the nominal feed throughput ratios summarised in Table 2.

The DMT laboratory results indicated that a suspension of one part tailings and 6.75 parts coarse rejects (this is the ratio outputted from the washplant) were pumpable and non-settling. The DMT results for the 5 mm top size mixture indicated that instability could cause a blockage, although none were encountered during the trial. The Tunra results indicated that the rejects sized to 5 mm could be pumped, stopped and restarted.

Whilst there were clearly aspects of the suspension that required further investigation, MCPL were sufficiently encouraged by the laboratory results to proceed with surface field trials.

Figure 2 - Particle size distribution - coarse rejects, TBS, tailings and combined
A field trial was conducted at the pit top facilities at Metropolitan Mine. A 500 m long tortuous 100 mm diameter pipe range was constructed as shown in Figure 4. The pipe circuit included two sections of 50 m that could be separated to test settling/restart for various residence times. The plant was composed of: a mixing hopper; pug mill; agitator tank; and pump as shown in Figure 5. The backfill plant had a swing tube pump mounted under an agitated tank which could hold up to 2.3 m$^3$. The pump was a swing tube positive displacement pump with the following features:

- stroke of 1.219 m stroke (48");
- a line bore internal diameter 0.1524 m (6");
- an hydraulic cylinder internal diameter of 0.0672 m (3"); and
- a variable speed hydraulic drive from 0 to 52 strokes per minute.

The field trials were designed to establish the operating envelope in terms of PSD; moisture content; pumping speed; density; friction loss; pipe retention times; and effect of pumping aid. In addition, qualitative assessment of flow characteristics after discharge onto the ground and potential penetration into caved strata were conducted.

The field trials established the following operating flow characteristics:

- Friction loss within a 100mm pipe range: 2 to 6 kPa (nominal 4 kPa);
- Moisture content (% w/w as): 15 to 36% (nominal 30%); and
- Density (kg/m$^3$): 1500 to 1700 (nominal 1600);
- Pipe retention time: up to 9 days.

The use of a pumping aid (EZ Flow from Cellcrete Australia Pty Ltd) in the range 50-150 mL/t was found to reduce friction loss and provide suitable pipe lubrication. Observation of flow after discharge from the pipe indicated that the slurry would flow over the ground with a beach angle of between 3° and 5°.

A trial of emplacement into a dam composed of loose rocks (Figure 6) to simulate caved strata indicated that adequate penetration should be achieved. Measurement of the shear strength of the slurry emplaced within the dam versus time was conducted.
The results shown in Figure 7 indicated that the drained shear strength of the emplaced slurry reached 50 kPa after approximately 75 days and continued to gain shear strength to approximately 95 kPa after 140 days. The results were considered adequate to eliminate the risk of liquefaction.
PILOT UNDERGROUND PLANT

Following the encouraging field trials, the plant used in the field trial was relocated to its final position adjacent to the CHPP as shown in Figure 8. At this stage the totality of rejects could not be emplaced underground until the remaining elements of the Metropolitan expansion project were completed. This would include an upgrade to the power facilities and CHPP upgrade to include a rejects comminution circuit. Prior to completion of the expansion project, the pilot plant included refinements including automation of the feeds; addition of a sizer to reduce the coarse material to -15 mm and modifications to the hydraulics. The feed combination of 50% TBS and flotation tailings and 50% coarse rejects sized to -15 mm represented a combined particle size distribution that would be achievable from the rejects comminution circuit once commissioned.

Figure 8 - Pilot plant layout

The PSD using a mixture of 50% from the sizer and 50% TBS and tailings provided a PSD consistent with that shown in Figure 3, in other words, that could be achieved without a grinding or milling circuit.

Underground emplacement commenced in May 2011 into disused workings. The underground pipe range extended for 890 m. Visual observations of the underground emplacement included:

- flow of over 200 m from the discharge point (unable to access workings beyond this distance);
- minimal segregation at a distance of over 200 m from the discharge point; and
- penetration through a roof fall that otherwise choked the roadway.

The PSD successfully emplaced underground (using 50% sized to -15 mm and 50% TBS and flotation tailings) compared with that obtained from a combined VSI and HSI crusher and compared with the unprocessed rejects is shown in Figure 9. The underground emplacement results confirmed that a high density slurry coarser than that generated by employing a comminution strategy that excluded a ball mill or grinding circuit, could be pumped underground.

At the end of October 2011, MCPL has emplaced over 15 000 tonnes of rejects underground which has taken some 500 trucks off the road. MCPL will proceed to full scale plant commissioning towards emplacing all rejects underground.

POTENTIAL FUTURE APPLICATIONS

As found in Poland, once the technology was installed, the use of low strength backfill found a number of uses including, control of spontaneous combustion, subsidence mitigation, and goaf ventilation reduction.

Another use identified was to use the backfill plant and reticulation system, and to add cement to make construction materials. It is envisaged to replace conventional concrete use underground with a lower strength mass concrete made from reject and delivered efficiently via the backfill underground reticulation pipeline. This would lead to mass concreted roadways and bulkheads at <$100 /cu.m. This would result in operational costs savings including eliminating scrapers, road repairs, reduced
travel times, less wear and tear on underground vehicles, and higher factors of safety on bulkheads. A recent study for another coal mine identified up to $50M in savings if concreted roadways were implemented at the start of a 20-year mine life.

MCPL will investigate the emplacement of rejects into the goaf behind an operating longwall using methods similar to that used at Walsum Colliery (Figure 1) to potentially reduce surface subsidence. The general concept is to construct a cemented rejects pillar approximately mid-panel that will bear a proportion of the overburden load that would otherwise be redistributed to the longwall abutments, thereby reducing subsidence.

![Figure 9 - Particle size distribution - rejects emplaced underground](image)

CONCLUSIONS

The underground emplacement of coal mine rejects has been successfully developed to pilot underground phase and is expected to advance to full-scale emplacement of all rejects underground at Metropolitan Mine.

The application is expected to progressively reduce the number of coal rejects trucks through Helensburgh, eventually eliminating the trucks by the end of 2015. MCPL intends to further develop the technology to potentially reduce subsidence by emplacement behind the operating longwall face.

In addition to the application at Metropolitan, based on overseas experience the technology has a range of other industry uses. These include: as a construction material; mitigation of spontaneous combustion events; goaf ventilation improvement; confinement of pillars for subsidence control; and use in standing support. Whilst the operating cost is three to four times more than typical surface emplacement, these applications may be cash positive since the backfill plant is relatively low capital cost.

The adoption of this technology not only replaces surface disposal, it provides a coal mine with a new hydraulic media and distribution system.

REFERENCES


NUMERICAL MODELLING FOR ESTIMATION OF FIRST WEIGHTING DISTANCE IN LONGWALL COAL MINING - A CASE STUDY

H.Manteghi\(^1\), K.Shahriar\(^2\) and R.Torabi\(^3\)

ABSTRACT: There are many parameters which affect caving processes in longwall coal mines, such as roof and floor strata conditions, thickness of immediate roof, excavation geometry and magnitude and direction of principles stresses. Estimation of first weighting distance, in this context is rather complicated. In this paper, the results of numerical modelling of the first weighting mechanism by using the finite difference code FLAC\(^2D\) at the E1 longwall panel of the Parvade1 underground mine, which is located in the Tabas area at the central part of Iran, are presented. The obtained first caving distance is 11.2 m. In addition, the results of numerical modelling have been compared with some conventional methods such as Peng’s method. The results show good agreement with each other.

INTRODUCTION

Longwall mining is a method with high production capacity and mechanisation potential. Its development backs to 17 century in European collieries. At the beginning of the 19 century this method with development of self-advanced support systems also was applied in United State mining. However, in the recent years, by using the longwall mining, coal production is increasing; the main reasons can be highly mechanized procedure and safety of this method. Two essential parameters for high safety are perfect ground control and prediction of strata movements around the coal seam.

Strata control in longwall mining has been a grey area of research since its introduction in underground coal mining industry worldwide. A reliable prediction of the caving behaviour of strata and its interaction with the roof support helps in selection of sustainable mining parameters and rational capacity of supports. It is prerequisite for developing a reliable support selection tool essential for successful planning of longwall working in a given geo-mining condition (Singh and Singh, 2009).

Theoretical models for prediction of main fall (first weighting) and periodic caving distances are based on plate -beam theory and bending moment approach. A number of empirical models have been developed on the basis of either a certain concept or some field experience to assess the caving behaviour of strata. Some of these approaches suggest roof classifications for qualitative assessment of caving behaviour. Some other models propose quantitative relation to predict the span of main fall. Similar relations have been proposed by various researchers to estimate the span of periodic caving. A few models give both the options of qualitative assessment of roof caving and the quantitative assessment of caving span (Singh and Singh, 2009).

Although application of the numerical modelling technique for strata control in longwall workings is not a new topic of research, a study has not been done to assess the strata–support interaction with progressive face advance in most of the cases. Most of these studies have been done using elastic analysis where simulation of face advance bears no importance. This paper describes a numerical modelling to assess the first caving of strata and the results will be compared with theoretical models and real results. Then, the performance of the numerical model; to assess the acceptability of the model for predicting the caving behaviour of strata in a given strata condition, is compared with field observations at the E1 longwall panel of the Parvade1 underground mine which is located in the Tabas area in the central part of Iran.

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CONVENTIONAL METHOD

Peng’s theoretical method

When a longwall panel of sufficient width and length is excavated, the overburden roof strata are disturbed in order from the immediate roof toward the surface. The overburden roof strata are divided into four zones shown in Figure 1: the caved zone, the fractured zone, the continuous deformation zone and the soil zone. The caved zone is created as the mining face advances and the immediate overburden falls and fills the void created by the extraction of the coal. The caved zone extends upwards, two to ten times the extraction thickness. The caved zone is characterized by irregular rock fragments that may have rotated relative to their initial locations, resulting in relatively high void ratios and permeability. The caved rock (goaf) is re-compacted by the weight of the overburden. The amount of re-compaction depends on the depth of overburden and the strength of the gob material. The fractured zone is located above and around the caved zone and is characterized by near vertical fractures and bedding plane shearing caused by the passage of the longwall face (Peng and Chang, 1984). Bed separation can occur in this zone. The fractured zone can extend 30 to 60 times the extraction thickness. Above the fractured zone is the continuous deformation zone. The rock is essentially un-fractured, but can experience shearing along bedding planes as they are deflected over the edges of the extracted longwall panel. Bedding plane shear will affect the horizontal conductivity of the rock. Field observations have shown that water levels and ground movements occur up to 60 m (200 ft) ahead of an advancing longwall face. These movements can be associated with shear along weak clay filled bedding planes. On the surface, there is a soil zone of varying depth depending on the location. In this zone, cracks open and close as the longwall face comes and goes (Peng and Chang, 1984).

![Figure 1 - Overburden movement resulting from longwall mining](image)

Movement of the zones has different degrees of effect on roof control at the longwall face. The effect decreases as the strata are located farther upward from the roof line. Those strata, the movement of which will affect roof control at the long wall face, can be classified into two types, immediate and main roofs. The immediate roof is that portion of the overburden strata lying immediately above the coal seam top, approximately two to ten times that of the mining height. Above the immediate roof, the strata in the lower portion of the fractured zone are called the main roof (Figure 2) (Peng, 2008).

Formulation of caving height or thickness of the immediate roof

The thickness of the immediate roof is the basis for designing roof control techniques. Normally, caving initiates from the lowest strata in the immediate roof and propagates upward into the fractured zone. The process of caving in each stratum is that the stratum sags downward as soon as it is undermined. When the downward sagging of the stratum exceeds the maximum allowable limit, it breaks and falls. As it falls, its volume increase; therefore, the gap between the top of the rock piles and the sagged but un-caved stratum continues to decrease as the caving propagates upward. When the gap vanishes, the caving will be stopped. Thus the height of the caving must satisfy the following condition (Peng, 2008).

\[
H - d = h_{im}(K-1) \quad (1)
\]

\[
And \ d \leq d_0 \quad (2)
\]
Where $H$ is the mining height, $d$ is the sagging of the lowest un-caved strata, $d_0$ is the maximum allowable sagging (without breaking) of the lowest un-caved strata, $h_{im}$ is the thickness of the immediate roof or caving height, and $K$ is the bulking factor of the immediate roof. Therefore, according to the Equation (1), the caving height can be determined by:

$$h_{im} = \frac{(H-d)}{(K-1)}$$  \hspace{1cm} (3)

It must be noted that if $d=d_0=H$, then $h_{im}=0$, which means no caving: that is, the roof sags gradually until it touches the floor. On the other hand, if $d=d_0=0$:

$$h_{im} = \frac{H}{(K-1)}$$  \hspace{1cm} (4)

**Formulation of first weighting**

There are a lot of complexities in design of roof loading in longwall coal mining when the strata above the coal seam has not moved yet and the first collapse is expected. Therefore, theoretical models must be used for prediction of main fall (first weighting). The distance is assessed based on beam fixed at both ends theory and bending moment approach. A beam fixed at both ends that is affected by uniformly distributed load is shown in Figure 3.

![Figure 3 - A beam fixed at both ends that is affected by a uniformly distributed load](image)

According to the Figure 3, the reaction forces at both ends are shown in Figure 4 (Majumdar, 1986).

![Figure 4 - The reaction forces at beam supports](image)

Therefore, the equilibrium equations are as follows:

$$\sum F_x = 0 \Rightarrow A_x - B_x = 0 \hspace{1cm} A_x = B_x$$  \hspace{1cm} (5)

$$\sum F_y = 0 \Rightarrow A_y + B_y - qL = 0 \hspace{1cm} \text{if } A_y = B_y \Rightarrow A_y = B_y = qL/2$$  \hspace{1cm} (6)

$$\sum M = 0 \Rightarrow -M_A - M_B + qL \cdot L/2 + A_y \cdot 0 - B_y \cdot L = 0 \Rightarrow M_A = -M_B$$  \hspace{1cm} (7)
Where $A_x$ and $B_x$ are reaction forces in the x direction, $A_y$ and $B_y$ are reaction forces in the y direction, $L$ is the length of beam, $q$ is the force at unit of beam length and $M_A$ and $M_B$ are bending moments at beam supports. According to Equation (7) the bending moments have been calculated by another method that is called super position method as following:

$$M_B = \frac{qL^2}{12}$$  \hspace{1cm} (8)

The maximum of bending moment occurs at both ends of supports (Equation (9)) and also the maximum of tensile stress that is sustainable for a beam fixed at both ends as shown in Equation (10).

$$M_{\text{max}} = \frac{qL^2}{12}$$  \hspace{1cm} (9)

$$\sigma_{t_{\text{max}}} = \frac{M_{\text{max}} C}{I}$$  \hspace{1cm} (10)

Where $\sigma_{t_{\text{max}}}$ is the maximum of tensile stress, $C$ is the neutral axis distance of beam from neutral surface and $I$ is the moment of inertia. On the other hand, $C$ and $I$ have been calculated by Equation (11) and Equation (12) for a rectangular beam.

$$C = \frac{h}{2}$$  \hspace{1cm} (11)

$$I = \frac{bh^3}{12}$$  \hspace{1cm} (12)

Where $h$ is the beam diameter and $b$ is the beam width. Therefore, by means of Equations (9) to (12) and to know that is $q = \gamma b h$, where $\gamma$ is the beam density, the maximum of allowable length beam is estimated as follows:

$$L = \sqrt{\frac{2\sigma_t h}{\gamma}}$$  \hspace{1cm} (13)

Where $L$ is the maximum of the allowable length beam or distance of the first weighting, $\sigma_t$ is the tensile strength of the immediate roof, $\gamma$ is the density of the immediate roof and $h$ is the thickness of the immediate roof.

**ROCK PROPERTIES**

**Assessment of material properties and rock mass strength**

Proper assessment of rock mass strength and modulus values is very important for a meaningful numerical modelling study of caving behaviour and support requirements. Therefore, physical and mechanical properties of each geological unit must be properly determined. In general, intact rock properties are determined by means of laboratory testing. However, there is an important difference between rock material and rock mass characteristics. It is compulsory to determine representative physical and mechanical properties of the rock mass instead of intact rock material. Data regarding the physical and mechanical properties of surrounding rock are given in Table 1 (Tabas Coal Project, 2003).

<table>
<thead>
<tr>
<th>Formation</th>
<th>Definition code</th>
<th>Density (MN/m3)</th>
<th>Uniaxial compressive strength (MPa)</th>
<th>Tensile strength (MPa)</th>
<th>Internal Friction angle ((\phi))</th>
<th>Cohesion C (MPa)</th>
<th>Modulus of elasticity E (MPa)</th>
<th>Poisson’s ratio v</th>
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</thead>
<tbody>
<tr>
<td>Siltstone</td>
<td>1</td>
<td>0.027</td>
<td>37.38</td>
<td>2.5</td>
<td>24.12</td>
<td>1.3</td>
<td>2838</td>
<td>0.26</td>
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<td>Sandy siltstone</td>
<td>2</td>
<td>0.025</td>
<td>73</td>
<td>2.6</td>
<td>-</td>
<td>-</td>
<td>2987</td>
<td>0.25</td>
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<tr>
<td>Silty mudstone</td>
<td>3</td>
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<td>18.8</td>
<td>0.3</td>
<td>-</td>
<td>-</td>
<td>2256</td>
<td>0.28</td>
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<tr>
<td>Coal</td>
<td>4</td>
<td>0.016</td>
<td>6</td>
<td>-</td>
<td>15-25</td>
<td>0.016</td>
<td>316</td>
<td>0.25</td>
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<tr>
<td>Mudstone</td>
<td>5</td>
<td>0.026</td>
<td>24.82</td>
<td>-</td>
<td>18.62</td>
<td>0.94</td>
<td>2838</td>
<td>0.31</td>
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<tr>
<td>Sandstone</td>
<td>6</td>
<td>0.027</td>
<td>72.79</td>
<td>6.3</td>
<td>31.75</td>
<td>8.69</td>
<td>5281</td>
<td>0.25</td>
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</tbody>
</table>

**Table 1 - Physical and mechanical properties of coal and surrounding rocks**
Laboratory tests were carried out on core samples from exploration drilling. Rock blocks are taken directly from the mine. The data presented in Table 1 are representative only of rock material. Determination of rock mass strength characteristics is rather difficult. Therefore, it is a common practice to derive rock mass strength from rock material properties by using various failure criteria. In this study, rock material properties were converted into rock mass data by using empirical relations widely used in the literature, e.g. Hoek and Brown failure criterion, Bieniawski’s RMR classification system, and Geological Strength Index (GSI). Physical and mechanical properties of the rock mass used for numerical modelling by FLAC2D are presented in Table 2 (Tabas Coal Project, 2003).

<table>
<thead>
<tr>
<th>Rock Definition</th>
<th>Siltstone</th>
<th>Sandy Siltstone</th>
<th>Silty Mudstone</th>
<th>Coal</th>
<th>Mudstone</th>
<th>Sandstone</th>
</tr>
</thead>
<tbody>
<tr>
<td>Definition code</td>
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<td>2</td>
<td>3</td>
<td>4</td>
<td>5</td>
<td>6</td>
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<tr>
<td>Density (MN/m³)</td>
<td>0.0272</td>
<td>0.0271</td>
<td>0.0268</td>
<td>0.016</td>
<td>0.0263</td>
<td>0.027</td>
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<tr>
<td>Internal Friction angle (φ)</td>
<td>27.42</td>
<td>31.75</td>
<td>22.17</td>
<td>15.76</td>
<td>20.13</td>
<td>43.52</td>
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<td>Cohesion c (MPa)</td>
<td>0.357</td>
<td>0.443</td>
<td>0.257</td>
<td>0.084</td>
<td>0.231</td>
<td>0.767</td>
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<td>Modulus of elasticity E (MPa)</td>
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<td>2818</td>
<td>1778</td>
<td>749</td>
<td>1995</td>
<td>3548</td>
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<td>Tensile strength (MPa)</td>
<td>0.012</td>
<td>0.007</td>
<td>0.005</td>
<td>0.002</td>
<td>0.013</td>
<td>0.017</td>
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<td>Poisson’s ratio ν</td>
<td>0.25</td>
<td>0.25</td>
<td>0.28</td>
<td>0.25</td>
<td>0.31</td>
<td>0.25</td>
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<td>Bulk modulus * (K) (MPa)</td>
<td>1492</td>
<td>1876</td>
<td>1347</td>
<td>499</td>
<td>1750</td>
<td>2365</td>
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<tr>
<td>Shear modulus * (G) (MPa)</td>
<td>895</td>
<td>1127</td>
<td>695</td>
<td>298</td>
<td>761</td>
<td>1419</td>
</tr>
<tr>
<td>Uniaxial compressive strength (MPa)</td>
<td>0.273</td>
<td>0.287</td>
<td>0.114</td>
<td>0.015</td>
<td>0.165</td>
<td>1.01</td>
</tr>
</tbody>
</table>

* K = E/3(1-2ν)  
* G = E/2(1+ν)

**NUMERICAL MODELLING**

Planning and modelling

The E1 longwall panel of Parvade 1 coal mine with two entry system is modelled using FLAC2D software. The E1 panel the first panel extracted was 170 m in width and 980 m in length using the retreating method. According to Figure 5 the width and length of model are 25 and 50 m, respectively. In addition, there are 18 layers with six types of rock from floor to roof (according to the Table 1 and 2). The Mohr-Coulomb model is selected for this study. The thickness of the coal seam is 2 m. The immediate roof consists of mudstone with 5.5 m in height (Tabas Coal Project, 2003). Initial and boundary conditions are shown in Figure 5. The constant load of 4 GPa applied instead of strata load to ground surface.

![Figure 5 - FLAC2D model geometry and boundary condition](image)

In the retreating longwall working, following the excavating of main and tail entry tunnels, the setup room tunnel will excavated between them approximately 7 to 8 m in length. When the coal extraction is beginning, the distance between end of the power support canopy and wall is almost 1.5 to 2 meters. Therefore, the first step will be modelling of this section (Figure 6a). A general layout for the longwall panel is described in Figure 6b.
Model formulation

Formulation of the two dimensional Mohr-Coulomb plasticity model consists of construction of model geometry, definition of the constitutive relation and material properties for rock mass and parting planes, in situ stress initialization and assignment of boundary conditions of the model. In addition to this, some monitoring parameters like history of unbalanced force for a given point have also been introduced to check the convergence requirement of the virgin model after its solution.

The model geometry of a longwall panel consists of 5.5 m floor rock overlain by a coal seam. The coal seam is overlain by roof consisting of two layers namely immediate roof and main roof. These layers comprise the caving zone of the overlying strata. The element size along X direction in the mining zone is 0.25 m. In Y direction, the size of elements in each layer has been kept equal. A schematic layout of one of the models showing the grid density in its different zones is shown in Figure 7.

The floor boundary is fixed in both the X and Y directions. The sides are fixed in X direction only (roller boundary) till the virgin model gets converged after initialization of in situ stress. The side boundaries are later on locked after initializing the X and Y displacements and the velocities to zero, simulating the clamped side boundary in the virgin model. These boundary conditions allow vertical and shear displacements in the model without affecting its external geometry. The X and Y displacement and velocity values in the virgin model are initialised to zero after achieving the equilibrium condition to simulate the virgin ground condition.
RESULTS AND DISCUSSION

For estimation of the vertical load variations on the coal production area with increase in the area width, before the first weighting occurs, pressure arch, beam theory and Terzaghi methods were employed. In Figure 8, the modelled vertical load on the working area based on these methods is illustrated.

According to the curves, pressure arch and beam theory methods show good agreement with each other. Also, in the Terzaghi method, vertical load on the tunnel is more than from other methods. By using the beam fixed at both ends theory and Peng’s equation (Equation (13)), the first weighting distance obtained 10.9 m and by means of site observations, this value has been reported as 12 m. By using numerical modelling, the progressive face advance is simulated in stages of 0.75 m till the main fall caving (first weighting) are observed within the two dimensional modelling limitations (Figure 9).

(a) 5.25 m Face advance at 5 steps
(b) 8.25 m Face advance at 9 steps

(c) 11.25 m Face advance at 13 steps, First weighting and caving of mudstone immediate roof

Figure 9 - Main caving (first weighting) of roof at Parvade E1 panel (X and Y scale ×10 m)
(a) 5.25 m Face advance at 5 steps  (b) 8.25 m Face advance at 9 steps  (c) 11.25 m Face advance at 13 steps, First weighting and caving of mudstone immediate roof

In this figure (Figure 9.c) red and brown colours indicate the highest and lowest rate of displacement, respectively. As expected, the displacement progress increases with face width advancing and expands toward its centre. In addition, in Figure 9, the state of displacement distribution, displacement vectors and state of plasticity in longwall model, are presented.

CONCLUSIONS

In this study, 2D modelling of the first weighting distance with the longwall caving method applied at the E1 panel of the Parvade1 Underground Mine was carried out by FLAC2D. For realistic modelling, material properties were derived for the rock mass from laboratory data by using Mohr-Coulomb failure criterion, the RMR and GSI systems together with empirical equations. Results of this modelling study revealed that the first weighting distance was found to be 11.2 m. Also the first weighting distance obtained by using analytical method was 10.9 m. According to the site observations, on the other hand, the first weighting distance has been reported as 12 m. Comparison of the results obtained from numerical approach, analytical calculations and observed data show good agreement. From the point of view of forces affecting the production area, the vertical load on the area based on pressure arch,
beam theory and Terzaghi methods were compared with each other. All these three methods show a consistent increase in the pressure with increasing coal face advance before the first weighting occurs. Pressure arch and beam theory methods show better agreement in comparison with the Terzaghi method.

REFERENCES

EARLY WARNING OF LONGWALL ROOF CAVITIES USING LVA SOFTWARE

David Hoyer

ABSTRACT: It is shown that by monitoring longwall leg pressures in real time, warning can be given for significant weighting events and the formation of roof instabilities, such as roof cavities, several hours in advance.

Longwall Visual Analysis (LVA) is a software package that continuously monitors shield pressures and shearer position in longwall mines. LVA has been running on 22 Australian longwalls for up to five years, and as a result a very substantial database of shield pressure trends in a wide range of longwall situations has been collected. This database has been analysed to develop indicators that will give operators and geotechnical engineers advance warnings of developing conditions such as weighting events and difficult roof conditions.

LVA displays live charts of data, such as shield leg pressures, loading rates, and yield frequencies. Data analysis techniques were applied to historical data records from multiple longwall sites in order to develop a “Cavity Risk Index” (CRI). The CRI indicates the relative risk of roof cavities developing, and it is based on pressure trends that indicate significant yielding and loading rates spanning a region of relatively low support.

Case studies from two different longwalls are given, showing how the CRI can give real-time advance warning of the formation of roof cavities.

INTRODUCTION

Longwall Visual Analysis (LVA) is a software package that continuously monitors, analyses, and displays shield pressures and shearer position in longwall mines. The first version of LVA was installed on an operating longwall in 2006, and the latest version is currently running on 22 Australian longwalls. As a result, a substantial database of shield pressures in a wide range of longwall situations has been collected. These data have been used to develop and test indicators that will give operators and geotechnical engineers advance warnings of developing conditions, such as significant weighting and the formation of roof cavities.

Other researchers have used roof geology in conjunction with leg pressure data to predict areas at risk of experiencing difficult roof conditions (Trueman, 2011; Wiklund, 2011). This paper uses leg pressure data only, which is easier to implement in practice.

LVA MONITORING OF LEG PRESSURES

The LVA software typically connects to the longwall face data via an OPC server (OPC is an industry standard for communication between machines). LVA reads leg pressures and shearer position every 20 to 30 seconds and creates an independent database of these values in a compressed format that allows fast access over networks. This provides sufficient resolution for LVA’s data analysis methods to identify individual set-to-release cycles on each shield, from which individual shears across the face can be identified. Various statistics can then be calculated for each leg during each set-release cycle or during each shear. These include time-weighted average pressure (TWAP), set pressures, loading rates, and leaking and calibration issues. Individual users (“clients”) can connect to the database via the network to view and analyse the data.

Figure 1 shows a typical LVA screenshot of pressures across the face and back in time over a period of 18 h. The black line over the leg pressure data is the shearer path.

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The 3D image in Figure 1 is very efficient for real-time monitoring of a longwall face by operators, but other visualisation techniques can be more effective for reviewing larger amounts of data, such as a whole panel or major portion of a panel. LVA can identify individual shears across the face by analysing patterns of supports setting to the roof, which allows data to be displayed on a shear-by-shear basis. Figure 2 is a “Load Cycle Map” in which each horizontal row of pixels represents the average pressure of each leg across the face during a specific shear.

Figure 2 clearly shows areas of weighting events (horizontal bands of “hot” or red colours), and areas of difficult roof conditions and low-pressure roof cavities (areas of “cool” or blue/green colours). This paper describes a method for determining indicators to show operators in real time when there is a higher than normal risk of these roof cavities forming. This would allow the operators to take preventive action to reduce downtime and safety hazards; these actions could include double-chocking (advancing the shields more frequently and in smaller steps) and not stopping for maintenance.

ALERTS FOR “WEIGHTING DEVELOPING”

The LVA software has an Alert system that logs messages and sends emails when certain events are triggered. One such Alert trigger is called “Weighting Developing.” This is currently being used successfully by several mines to provide early warning of the start of unusually high weighting on a portion of the face. According to Moodie and Anderson, “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 and yield counts in a cycle”.

Figure 3 shows how loading rates and yield counts can be calculated by LVA from the leg pressure trending data for each support. The “Weighting Developing” alert system has the following algorithm. LVA calculates the loading rate during the initial 5 to 10-minute period after the start of each set-to-roof cycle for each shield. During the cycle, LVA also counts yield events by identifying the characteristic rising/falling pattern of leg pressures when yielding. Every few minutes LVA determines which shields have the following properties, where numbers in square brackets are configurable by the user:

- The loading rate is at least two bar/min during the period five to ten minutes after the shield is set to the roof;
- The number of yields is at least three during the set-to-roof cycle;
- The time-weighted average pressure (TWAP) is at least 380 bar;
- A “Weighting Developing” alert is issued when at least ten shields simultaneously satisfy the three conditions above.
An Alert triggered by this algorithm indicates that a significant number of shields across the face are experiencing both high loading rates and substantial yielding. The Alert event is displayed on screen as a text message and optionally emailed to a list of recipients so that appropriate action can be taken.

Figure 2 - LVA load cycle map showing time-weighted average pressure (TWAP) on individual legs across the longwall face for 989 individual shears over about 12 months. Weighting events show as horizontal bands of “hot” or red colours. Areas of difficult roof conditions and roof cavities show as “cool” or blue/green colours.

Figure 3 - Leg pressure trend showing start and end of a set-to-roof cycle, and illustrating the calculation of loading rate and number of yields in a cycle. This cycle had five yields on the maingate leg (red) and three on the tailgate leg (blue).
ANALYSIS OF PRESSURES, YIELDS AND LOADING RATES

Figure 1 showed how instantaneous leg pressures can be displayed across the face and back in time using 3D graphics. Note, however, that these data are often quite noisy and the signal noise can mask longer-term trends such as periodic weighting and cavity formation. LVA provides a signal filter that smooths the data. For example, Figure 4 shows similar data to Figure 1, though over 36 h instead of 18, with smoothing applied. Both the weighting and cavity events are much clearer when smoothed.

The same smoothing and 3D presentation methods can be applied to yield count and loading rates, as shown in Figures 5 and 6. These are the data formats that will be used in the next section to develop a cavity risk alert system. In particular, compare Figures 4 and 5. Note how in Figure 5 showing Yields in a Cycle, there are two "mountain peak" areas that form on each side of where the cavity formed in Figure 4. Note also that these peaks can be identified several hours before the cavity forms.

Figure 4 - Similar data to Figure 1, but with the shearer path removed and with a smoothing filter applied. Note the appearance of a substantial roof cavity near the middle of the face (blue area at front of image)

Figure 5 - Smoothed image of yield count for each shield across the face and back in time. Note the formation of a significant "bridge" to each side where the cavity formed (cf. Figure 4). This bridge was prominent for several hours prior to the cavity forming
WARNINGS OF POTENTIAL ROOF CAVEITY EVENTS

The “Weighting Developing” Alert system described above has proved useful at several longwall sites. The primary concern when responding to weighting alerts is to try to reduce the occurrence of actual roof cavity events, namely complete loss of support pressure over a portion of the face and a section of the roof collapsing. This situation often results in loss of production and increased safety hazards. However, if an operator gets a Weighting Developing alert, they still do not know if the situation may develop into a cavity, which is a much worse condition. It was speculated that it should be possible to develop a “Cavity Risk” indicator by modifying the weighting alert algorithm. Additionally it was considered that a real-time display system with continuous visual feedback would be more useful to operators than a text-based Alert system when heading into potentially difficult roof conditions.

Cavity risk index algorithm

Several different algorithm types were trialled on historical roof cavity events from different longwalls across Australia. Roof cavity events can be clearly identified from the Load Cycle Maps of TWAP, like the example shown in Figure 2. The LVA databases from each longwall site were used to replay conditions leading to cavity events, in order to test the effectiveness of various algorithms. It was found that cavity events are frequently preceded by the type of Yield bridging shown in Figure 5, and this was used as the basis of the Cavity Risk algorithm in conjunction with loading rates. Note that the bridge is neither a physical bridge nor a pressure bridge, it is a conceptual “yield and loading rate” bridge, which begins forming several hours before the pressure bridge of a roof cavity.

The algorithm finally selected to quantify cavity risk is as follows:

- Each individual shield is considered a “cavity risk trigger” if
  - its loading rate is at least two bar/min five minutes after setting the shield to the roof;
  - the number of yields is at least three during the set-to-roof cycle.
- The pattern of cavity risk triggers across the longwall is analysed to determine whether a bridge exists somewhere on the longwall. A bridge in this context is a region of non-triggered shields (low yielding and loading rate) straddled on each side by a region of triggered shields (high yielding and loading rate). For example, a bridge may be said to exist when a region at least ten shields wide has no triggers, and the region is straddled by regions on each side that have at least four out of 12 shields in a trigger state. The numbers given here (10, 4 and 12) have proven satisfactory in the cases studied so far. Further work is required to see whether these may need to be tuned specifically for different sites.
- A “Cavity Risk Index” (CRI) is calculated from the size of the bridge, from 0% when no bridge exists to 100% when four or more out of 12 shields on each side of the bridge are triggered.
- The calculated CRI can actually decrease before a cavity event as the roof begins to weaken. For this reason, the displayed CRI is taken to be the peak CRI value over the previous 12 h when implemented in a real-time warning system.
Filtering is applied to the CRI trending, to reduce the effect of anomalies such as isolated spikes.

**CAVITY RISK - CASE STUDY A**

This case study was taken from an Australian longwall mine with 170 shields of capacity 875 t, with a yield pressure of 430 bar (the same data shown above in Figures 3 to 6). The immediate roof is sandstone conglomerate, with thick massive sandstone conglomerate channels from 60 m above the seam. The case study shows how the CRI changed from low to moderate about seven hours before a roof cavity formed, and to extreme about three hours before the cavity formed.

Figure 7 shows how the calculated CRI varies between close to 0% and 100% over a period of 36 h. The CRI went into the “high” range, above 80-percent at 3:28 am, and reached 100-percent at 7:26 am.

Figure 8 shows a sequence leading to the roof cavity forming at 10:40 and becoming more pronounced by 13:55. The sequence of seven snapshots, labelled A to G, shows the state of the longwall leg pressures at different times. The x-axis shows shield number, in this case from 1 to 170. The blue dots along the bottom edge mark those shields that satisfy the trigger conditions of high yielding and high loading rate. The CRI is calculated from the pattern of these dots as described earlier. The sequence progresses as follows:

- 28 May at 13:04. Leg pressures across the face are within the low to normal operating range. The thick black line is a smoothed profile drawn through the leg pressures. The blue dots along the bottom section of the graph mark those individual shields that are in a “cavity risk trigger” state, meaning they have experienced high loading rates and multiple yields. The Cavity Risk Index (CRI), shown in the gauges to the right, is calculated from the pattern of individual shields in the cavity risk trigger state - specifically identifying when the longwall face is in a bridging state. At this stage the CRI is low, as the algorithm applied to the pattern of blue dots indicates a weak degree of bridging.

- Four hours later, 17:02. Leg pressures across the face are still within the low to normal operating range. The CRI is low but just touching on yellow (moderate). Note that the black gauge needle represents the peak CRI value over the previous 12 h, not necessarily the value from the exact pattern of dots shown in this image.

- Eight hours later, 29 May 03:28. Leg pressures across the face are still within the normal operating range, but the CRI is high, just touching on red.

- Four hours later, 07:26. Leg pressures across the face are still within the normal operating range, but the CRI has just jumped to extreme.

- 10:19. The CRI has been extreme for three hours, while leg pressures across the face remain within the normal operating range.

- 10:40. After the CRI has been high for seven hours and extreme for three hours, major loss of roof pressure occurs between shields 65 to 95, indicating formation of a roof cavity.

- Three hours later at 13:55 the cavity is more pronounced. Mining stops for two days.
Figure 8 - A sequence of seven snapshots (A to G) of the state of the longwall leg pressures, leading to the cavity formation. The CRI becomes high several hours before the cavity forms

CAVITY RISK - CASE STUDY B

This case study was taken from an Australian longwall mine with 181 shields of capacity 913 t and yield pressure 420 bar. The case study shows how the CRI changed from low to moderate about three hours before a roof cavity formed, and to extreme about two hours before the cavity formed. Figure 9 shows the smoothed leg pressures across the face and back in time, developing to a cavity at the front of the image.

Figure 9 - Case Study B. Smoothed leg pressures across the face and back in time. Note the appearance of a substantial roof cavity near the middle of the face (blue area at front of image)
Figure 10 shows how the calculated CRI varies between 0% and 93% over a period of 18 h. The CRI was in the “high” range of 60 to 80-percent at 2:46 am, and reached the “extreme” range of 80 to 100-percent at 3:30 am. Figure 11 shows a sequence of smoothed face pressure profiles and CRI readouts leading to the roof cavity forming at 5:28.

Figure 10 - Case Study B. The trending graph over 18 h of the Cavity Risk Index, rising from low at 0:48 to high at 2:46 and extreme at 3:30. A roof cavity began forming at 5:28

Figure 11 - Case Study B. 21 Aug at 14:00 to 22 Aug at 5:28. Showing how the CRI varies from low to extreme as leg pressures across the face change, leading to the roof cavity at 5:28

IMPLEMENTATION OF REAL-TIME CAVITY RISK INDICATOR INTO LVA

Figure 12 shows a prototype of how the CRI has been integrated into the LVA software for continuous real-time feedback on the future risk of roof cavities forming.
CONCLUSIONS

The LVA shield monitoring software calculates a Cavity Risk Index (CRI) from patterns of yielding and loading rates across a longwall face. This CRI warns of developing poor roof conditions including roof cavities - often giving several hours more warning than might be gained from looking at leg pressure trends alone. This advance warning allows operators to take preventive action such as double-chocking and not stopping for maintenance, which in turn can reduce downtime and safety hazards.

The CRI has been implemented in the LVA software program to provide real-time warnings of cavity risk to longwall operators. Two case studies were presented in this paper. Approximately ten additional roof cavity events have been identified and studied to date. In all but one the CRI was triggered several hours before the cavity formed. The confidence that a cavity can be predicted with several hours notice is therefore high, though additional work is needed to quantify the incidence of false cavity predictions.

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REFERENCES


THERMAL INFRARED-BASED SEAM TRACKING FOR INTELLIGENT LONGWALL SHEARER HORIZON CONTROL

Jonathon C Ralston  and Andrew D Strange

ABSTRACT: Longwall mining remains one of the most efficient methods for underground coal recovery. A key aspect in achieving safe and productive longwall operations relies on maintaining the shearer in an optimal position for extraction within the coal seam. The typical approach to this resource identification issue is labour intensive so is subject to safety and productivity drawbacks. As a solution, this paper describes the use of thermal infrared-based sensing to provide a means to automatically measure the vertical position of the mining machine with respect to the coal seam. This is achieved by identifying and tracking non-optically visible horizontal line-like bands in the main body of coal, which are known as marker bands. These marker bands are strongly linked to the profile of coal seam structure, a geological characteristic often used by operators as an ad hoc datum for maintaining in-seam alignment of the shearer. Details on the theory behind thermal infrared imaging and practical aspects involved in implementation of the method are given. As there are very few real-time solutions available to locate and track coal seam profiles, this approach overcomes a current limitation in implementing intelligent horizon control systems for advanced shearer operation. Measurements from a shearer-based sensing system are given to demonstrate the approach.

INTRODUCTION

Longwall mining

Longwall coal mining is a full extraction underground mining method that involves the removal of coal in large blocks or panels using a mechanised shearer. The coal panel is typically 200-350 m wide and can be up to five kilometres in length. The mechanical shearer is mounted on a shearer pan and rails which guide the shearer as it moves back and forth across the coal face. With this method of mining, the roof is supported by hydraulic shields that are individually advanced as mining progresses. As the roof support system advances into the coal panel, the mine roof is allowed to collapse into the void behind the shields. For reference, a representation of a small portion of a longwall operation and shearer is shown in Figure 1. The coal seam is indicated by the hatched layer between the underlying and overlying strata.

Figure 1 - Representation of a small portion of a longwall operation. The coal seam is indicated by the hatched layer between the underlying and overlying strata. For clarity, the central section of the roof support system is not shown.

LASC automation

Over the past decade, the CSIRO Mining Technology Research Group has undertaken focussed research and development activities to deliver a means to accurately determine both the location and orientation of the shearer in real-time. This technical innovation developed a new automated face alignment capability and new options for extraction control, allowing the shearer to accurately achieve a given path and geodetic heading (Reid, et al., 2006). The benefits of this outcome have been broadly recognised and embraced by the longwall community and have subsequently been taken through to
commercial maturity through an industry-driven Longwall Automation Steering Committee (LASC, 2011).

The need for coal resource location

However, achieving a highly productive longwall operation relies not only on accurately knowing the location of the equipment but also on precise information regarding the spatial location of the coal resource. This information is critical in order to maintain the shearer in an optimal position for extraction within the coal seam. Operating in seam is also important in terms of minimising damage to equipment, reducing dust production, and to relieve personnel from potential hazards such as hard-rock debris dislodged from out-of-seam mining. In an attempt to provide a solution to this problem, this paper considers the use of machine-based sensing, specifically a Thermal Infrared Camera (TIR), to provide a means to measure the vertical position of the shearer with respect to the coal seam. The processed sensor output can then be applied to any LASC compliant longwall automation system to provide geologically intelligent information for real-time shearer horizon control.

SEAM TRACKING BY MARKER BANDS

The importance of marker bands

The standard approach for identifying coal seam boundaries requires shearer operators to monitor various indicators such as visual cues present in the geology, a change in dust conditions when transitioning coal-rock boundaries, or inspection of the extracted resource. For a variety of reasons the most efficacious methods are based on observing the geological characteristics associated with the overall coal seam.

The material mined during the longwall operation is composed of several layers of coal, clay or ash of varying thicknesses. This horizontal layering characteristic is well known in underground coal operations, where the overall body of coal to be extracted also contains thin horizontal layers or bands which consist of high ash-content material. These thin horizontal layers are often referred to as marker or penny bands, which visibly contrast with the host coal strata because of their light-grey or white colour. The nature and presence of marker bands is geology-dependent, so that they may or may not be observable to the naked eye in a given longwall operation. Figure 2 shows a short section of a longwall face revealing the characteristic horizontal geology of coal and marker bands.

Manual tracking of marker bands

In terms of shearer guidance, the major interest in marker bands arises because the bands often exhibit similar horizontal trends to the host coal seam geology. For operators, marker bands are particularly important as they often provide the only visible indicator to infer the relative vertical location of the shearer with respect to the overall geological coal seam trend. To maintain the shearer in a desired coal seam horizon, an operator will manually note the vertical distance of a particular marker band from the floor (or roof) extraction boundary at a nominal location across the face, and then attempt to hold that distance constant as the shearer progresses across the face. In order to achieve reliable and effective horizon control, this process needs to be maintained in a similar fashion as the shearer retreats into the panel.

Figure 2 - Photo taken of an underground longwall face which shows characteristic darker coal and lighter marker band features. A dominant marker band is seen at the bottom of the image, as well as finer, less immediately obvious bands above. The top of the image shows sections of the roof canopy support.
Limitations to manual band tracking

The manual approach for shearer horizon control has obvious drawbacks, particularly in terms of the amount of human effort necessary to constantly identify and track the geological marker band features in light of other operational duties. Being a manual process, there remains a practical requirement to both record and relay information across shearer crews regarding the state of horizon tracking.

Practical situations also arise where the marker bands are simply not observable due to the dust generated from extraction, equipment temporarily restrict the field of view, and operational duties and hazards effectively prohibiting ready access to observe the face. These practical issues in turn affect the quality and consistency of the overall coal recovery process, and ultimately serve to impact on production performance.

Automatic sensing of marker bands

Many of the challenges associated with manual band tracking, however, could be overcome if a reliable means of sensing could be developed to automatically generate a vertical datum for horizon control. Presently, very few (if any) reliable real-time solutions exist to locate and track seam profiles. To this end, research and development was undertaken in an attempt to provide solutions to this problem.

Marker band tracking using vision-based sensing

Initial research concentrated on replicating human vision capability using a visible-light camera. This produced imagery as seen in Figure 2, where a camera was installed on a supporting chock. A prototype system was developed consisting of three camera units. To minimise cable runs, the camera system also trialled a Broadband over Power Line (BoPL) technology and where Ethernet-based communications were maintained via the 110VAC supply. Figure 3 shows the visible light camera system being evaluated during in-house validation.

![Figure 3](image)

Figure 3 - In-house validation of a three-camera visible light camera system. To the left of the image is the junction box which houses the communications components, and to the right are the cameras and lighting flameproof enclosures.

The visible-light camera method represented an intuitive approach to marker band tracking that yielded some useful outcomes. However, the particular implementation also gave rise to some interesting limitations in practice, particularly in terms of its application to the shearer horizon control problem. Four specific limitations encountered were:

- The requirement for near constant illumination in order to clearly image the longwall face;
- The presence of water and dust, which often completely blocked any visible-light imaging;
- A single, fixed position for sensing restricted observability to a small portion of the face;
- Some minesite geology simply did not give rise to readily observable visible marker bands.
Marker band tracking using thermal infrared sensing

To overcome the limitations associated with a visible-light camera configuration, an alternative approach was considered that utilised TIR sensing. The idea of using TIR imaging is based on the concept that all objects emit thermal infrared energy based on their temperature. This energy can be detected using a device that is sensitive to radiation in the thermal infrared range of the electromagnetic spectrum. TIR energy is not visible to the human eye, but rather operates at a different region of the electromagnetic spectrum as noted in Figure 4.

The specific linkage between TIR sensing and longwall operations becomes apparent by noting that the engagement of the rotating shearer drum with the coal face surface gives rise to regions of thermal contrast due to frictional interaction. These temperature differences are observable in the thermal-infrared region of the electromagnetic spectrum. The intensity of this radiation can be detected by an infrared camera and displayed as a digital image, where the pixel values represent the measured TIR intensity.

![Figure 4 - The full electromagnetic spectrum, showing the wavelengths associated with thermal infrared energy. For comparison, the familiar visible light spectrum is also shown. (Courtesy L. Keiner, Coastal Carolina University, adapted).](image)

Thermal infrared cameras

A typical output of a camera is represented as a grey-scale image of varying pixel intensity values where darker and lighter respectively correspond to cooler and hotter objects. A thermal infrared camera can produce live images, much like a video camera, but based on temperature variation in the field of view. Most thermal cameras operate in the low to mid thermal range of the electromagnetic spectrum, being sensitive to electromagnetic energy with wavelengths in the range of 1 - 15 µm. Figure 5 shows a typical compact thermal infrared camera, which is ideal for housing in small flameproof enclosures with a window transparent to TIR radiation (FLIR 2010).

![Figure 5 - A typical compact TIR imaging camera for OEM use (Omega, now FLIR). The small-form factor means that it can be readily incorporated into existing approved flameproof enclosures.](image)
Advantages of thermal imaging over vision

TIR technical offers several advantages over traditional vision-based imaging for the shearer guidance problem. In particular, the technology has important operational characteristics which can overcome the limitations associated with vision-based sensing:

- TIR cameras can operate in complete darkness without any loss of image quality, i.e., the camera output is independent of the level of ambient (visible) light present;
- It can effectively penetrate through large dust and water particles, allowing imaging in a much broader range of operational conditions than possible with visible light cameras;
- Thermal cameras are similarly sized to regular vision cameras, and so can be conveniently deployed in a mining context where mounting space is limited, e.g., a mobile shearer-based installation;
- Sensing of novel “thermal marker bands”, which are thermally observable features strongly linked to visible marker bands and thus coal seam trends, but which are otherwise invisible to the human eye.

Flameproof enclosure

A practical system for underground use must be installed in a flameproof enclosure that can accommodate the unique characteristics of a TIR camera (Ralston, et al., 2006). The flameproof enclosure must also meet the constraints of IEC Ex d standards, namely impact testing, chemical inertness, and pressure testing. However, unlike regular vision-based imaging, thermal-infrared energy cannot readily penetrate through the standard glass or Perspex window typically used for flameproof enclosures. It is therefore necessary to utilise a material that supports transmission of TIR radiation. Of all the candidate materials possibly for this, a germanium-based window is most favourable, and hence a compliant enclosure utilising this material was developed (see Figure 6).

![Figure 6 - An approved TIR thermal infrared flameproof enclosure and camera conforming to IEC Ex d standards](image)

Thermal infrared camera mounting location

An ideal location for the thermal infrared camera is on the body of the mining machine, oriented such that it has a useful viewable aspect of the drum and surrounding strata while itself being sufficiently protected from the rough operational conditions.

Figure 7 shows output from a thermal infrared camera mounted on the body of the shearer that provides a view of the face and cutter drum activity, conducted as part of the original Landmark Longwall Automation activity (Kelly, et al., 2006). This image shows temperature variations on both the face and roof which correspond to different areas of thermal intensity. Figure 7 also shows additional detail in the thermal domain associated with the interaction of the cutter drum with the roof. As the cutting drum departs from the (relatively soft) coal seam and encounters harder shale or rock material, an increase in frictional forces occurs which leads to an increase in temperature of the cutting picks and the surface where material is being extracted. This increase in temperature is readily observable in the TIR
thermal infra-red domain, appearing as a noticeable increase in image intensity in the vicinity of the cutting drum (see top of Figure 7). This additional information from the seam boundary can be processed in a similar manner to the marker-band detection process to provide a coal-interface detector to localise the upper and (with a second TIR camera) lower boundaries of the coal seam.

Figure 7 - Thermal image acquired from a shearer-mounted TIR thermal infrared camera. The viewing region has the camera directed upwards towards the longwall face and roof, with the cutting drum to the right. Thermal regions are mapped to image intensity: lighter sections correspond to higher temperatures and darker sections to lower temperatures.

IMAGE PROCESSING AND SEAM TRACKING

Thermal image acquisition

Figure 8 shows a typical snapshot of the thermal imaging camera video stream acquired from the camera mounted on the shearer towing arm at a longwall mine. This image shows a horizontal feature, corresponding to a “thermal marker band”, arising from the frictional forces acting on material with harder mechanical properties. It should be noted that this thermal feature is not visible to the human eye. The darker section on the bottom third of the image is coal build up on the shearer body.

Of note, this image was acquired in geological conditions where the shearer produced a large amount of airborne particles and debris; hence the face was almost unobservable for visible-light systems. However using TIR imaging the band feature is still readily apparent, demonstrating a key property of TIR thermal infrared imaging technology: its imaging quality is not as severely degraded by particulate matter as an ordinary video camera.

Figure 8 - Typical thermal infrared image acquired from the camera mounted on the shearer towing arm at Broadmeadow longwall. At the top of the image are the roof supports, mid section shows a TIR thermal infrared feature, and at the base is the shearer body.

Data processing

Preliminary data processing was performed on the TIR images in order to evaluate the potential of a line-based tracking algorithm for horizon datum generation. The results of this processing are shown in Figures 9 and 10. The left figure of Figure 9 shows the original TIR image and the right image shows the superimposition of automatically extracted line segments corresponding to the thermal marker band.
Figure 9 - Original thermal image (left), and tracked thermal marker band (right)

Figure 10 shows the extracted marker band position information in a form that is useful for horizon datum generation. Note that a simple interpolation is performed in sections where the tracked marker-band quality is degraded (see Figure 9). This demonstrates how the TIR image data can be meaningfully used for the development of an online automated horizon sensing capability.

Figure 10 - Automated horizon datum extraction from a single thermal infrared TIR image acquired from the Broadmeadow longwall thermal infrared trial

**LASC compliant system integration**

The information generated from the TIR sensor can be integrated with an existing LASC longwall cut-model database to provide a horizon control system. This involves LASC-compliant communications, intelligent processing, system validation and integration. Underground longwall personnel will control and manage the system through an updated graphic user interface that is incorporated into the LASC-standard Operator Controller Display (OCD). This allows the operator to set the desired horizon extraction according to a pre-determined strategy. It will also provide a graphical display of the desired extraction profile and the option to either select or ignore this input. The block diagram in Figure 11 shows the overall system topology to integrate the TIR based sensing into an existing Longwall configuration.

Figure 11 - Interconnection of TIR based sensor with an existing LASC compliant system
DISCUSSION

The evaluation of the TIR camera has identified it as a very useful tool for developing an automated horizon steering capability. These results serve to consolidate existing thermal infrared data acquired in a previous thermal imaging trial at Beltana. Of special note, different geological conditions in the later study at Broadmeadow led to the generation of new and highly useful techniques for processing thermal features.

Amongst other lessons learnt from the later trial, it was clear that the location of the thermal imaging camera and its mounting angle plays an important role in the performance of band feature detection. The results suggest that the camera should be mounted perpendicular to the face to provide the most robust thermal features for automated marker band detection.

Further investigation needs to be undertaken to determine a means to allow for the longer term implementation of the shearer-based camera deployment. Practical survivability of the enclosure is an ongoing challenge, but several options have been considered and it is believed that a viable solution is possible.

Possibilities also exist to utilise a number of cameras at fixed locations, e.g., mounted on the roof supports rather than a single camera on a mobile shearer. Several useful developments can be made using the TIR imaging system to provide additional monitoring capability. In particular, thermal disparity associated with voids in the face could form the basis of an automated face integrity monitoring system.

SUMMARY

This paper described developments associated with thermal infrared imaging to provide a means to measure the vertical position of the mining machine with respect to the coal seam. Both optical and thermal approaches were presented as a means to identify and track marker band features towards providing an automated sensing strategy for shearer horizon control. A LASC-compliant integration of such a system was also described. Developments in this area are important for realising intelligent shearer horizon control systems that will boost productivity and enhance safety for operators.

REFERENCES

BENCHMARKING CONTINUOUS HAULAGE

Allison Golsby

ABSTRACT: Continuous haulage systems have not always presented a satisfactory operational experience for hard rock or coal miners. Not all mines have used continuous haulage. Some mines presently use one of the continuous haulage systems. Some have tried and abandoned continuous haulage. Yet, continuous haulage offers considerable benefits, which are not always realized. However, there is now a resurgence of interest in these systems as coal mines seek to improve gate road development rates. As most market players are very reluctant to publish information, benchmarking continuous haulage systems can be difficult. This is to the detriment of the industry as a whole. The continuous haulage examples discussed are; bridge, flexible belt, chain, temporary belt support, pipe conveyors and pneumatic systems.

INTRODUCTION

Continuous haulage systems have not always presented a satisfactory operational experience for hard rock or coal miners. Not all mines have used continuous haulage. Some mines presently use one of the continuous haulage systems. Some have tried and abandoned continuous haulage. Yet, continuous haulage offers considerable benefits, which are not always realized. However, there is now a resurgence of interest in these systems as coal mines seek to improve gate road development rates.

Since this paper discusses the benchmarking of continuous haulage, continuous haulage needs to be defined. A definition developed from research is, “Equipment designed and used to obtain continuous throughput of material from the mine face to the main mine load-out conveyor belts; unlike the pulsed, batch load throughput made possible by usage of shuttle cars and battery haulers”. This paper is looking predominantly at underground mining, especially the coal mining applications of continuous haulage. There are various designs of continuous haulage systems on the market. Most of these systems are used in the United States. Before use in Australia, the systems need to be modified to meet the Australian Regulatory requirements.

The continuous haulage methods explored in this article will be bridge, flexible belt, chain, temporary belt support, pipe conveyors and pneumatic systems.

SYSTEMS

Bridge conveyor

Most bridge conveyor systems consist of mobile bridge sections; track or wheel mounted and carry chain or rubber belt conveying decks. Bridge sections are typically short (6 m on conveyor bridges and 16 m on chain type bridge systems) and are self-propelled. Depending upon seam (and hence mining) height, the discharge end of these systems can either run over or beside the main conveyor. This enables the bridge conveyor to discharge on the section conveyor as the bridge conveyor follows the continuous miner through the development sequence. Bridge continuous haulage systems provide a haulage system similar to the flexible conveyor train systems.

Bridge conveyors consist of several linked bridge segments using chain conveyors. At each intersection a crawler unit is required, where one operator for each unit might be required. An eighty metre pillar block would require a bridge conveyor with about eight segments and an overall length of 180 m.

The Flexiveyor system (Figure 1) includes a self-deploying conveyor that straddles the section conveyor and loop take up. The conveyor in the Flexiveyor system might have 16 individual cars to a total of 96 m, resulting in a belt advance occurring every 30 to 90 m.
Figure 1 - Flexiveyor linear mining system (Diversified Mining Services 2010) flexible belt conveyor

Various flexible conveyor trains have been produced including both floor mounted and roof mounted continuous conveyor systems. Both systems offer some degree of operational flexibility. The discharge end of the flexible conveyor runs above the section conveyor. This enables the flexible conveyor to discharge onto the section conveyor as the flexible conveyor follows the continuous miner through the development sequence. The face end of the flexible conveyor is attached to the rear of the continuous miner or is self-propelled and kept at that position. Both roof and floor mounted flexible conveyor systems were trialled in Australian mines during the late 1980's with limited success.

The Joy 4FCT01 (Figure 2) is available in lengths up to 128 m and requires one operator.

Figure 2 - 4FCT01 (Joy, 2008)

Chain conveyor

Chain conveyors (Figure 3) consist of four basic units: a breaker car module, conveyor bridge module, mobile bridge module and rigid haulage system. The system configuration and number of these units depends on individual mine application and production requirements. Systems can be up to 200 m of flexible chain conveyor with a feeder breaker behind a continuous miner. From the chain conveyor the coal is transferred via a belt interface onto the section belt. Chain conveyor systems often have a lower profile and thus are more suitable for lower seam workings.

Figure 3 - Joy continuous chain haulage (Joy, 2010)
Temporary belt support

A temporary belt support system is comprised of a telescopic conveyor utilising a belt bending section and collapsible A-frame belt supports mounted on skids. These systems allow the inserting of new belt structure and idlers without interfering with production. Temporary belt support systems are available that can facilitate belt extensions during belt operation. These systems are used predominantly in low seam mining as shown in Figure 4.

Joy’s system requires a take up unit and has a length of 12 m, where CONSOL’s temporary belt support system is 80 m long and has an optional take up unit. There is the potential for the continuous miner to be connected directly to the section conveyor when driving the belt road.

Figure 4 - Temporary belt support (S and S Sliders, 2011)

Pipe conveyor

Pipe conveyors are self-advancing and retreating via a monorail system and a hydraulic winch system. Maximum effective haulage length is approximately 200 m. Due to the closed conveyor concept spillage is non-existent. The design relies on a stretchable rubber belt driven by multiple friction rollers acting on a vertically vulcanised drive strip. Pipe conveyors include tear drop conveyors (Figure 5 and Figure 6); both systems use a closed loop of conveyor belt.

Figure 5 - Vach 500 loading area (Sandvik, 2010)

Instead of running over rollers as a traditional conveyor system would, tear drop conveyors are suspended from a number of idlers on “j” sections that pull the belt into a tear-drop shape for much of its travel. This brings the benefit of enclosing whatever is being transported, removing the need for external structures to be built around the belt to stop dust escaping (Figure 5).
the belt together helps generate bridging, which stops the material being transported falling back down the belt.

Tear drop conveyors have the ability to handle curves with a radius of about 5 m, much tighter than conventional conveyors. The tear drop conveyor belt only takes the weight of the load but not the tensile load; the “j” sections take the tensile load, and also helping the conveyor to reach an 80-degree elevation.

![Diagram](image.png)

**Figure 6 - Vach 500 discharge area (Sandvik, 2010)**

This continuous haulage system is supported by a monorail system from a track driven hopper car, which will also act as the loading device for the conveyor system. The hopper car can be equipped with a roof bolter and enough storage space for 100 m of monorail and an inboard lump breaker.

**Pneumatic conveyor**

Negative pressure (vacuum) conveying systems are ideal for coal recovery because coal can be loaded and conveyed from several faces to a common storage hopper. Coal is loaded directly into the conveying system at the face by the vacuum action of the system. The vacuum system has proved itself in removing slurry and waste from sumps.

The vacuum coal loading system involves use of air injector pumps to generate the vacuum, a separator/surge hopper to remove the coal from the air stream, plastic PVC pipe for haulage and flexible loading tubes for loading the coal at the face.

The vacuum coal loading and conveying system is technically simple and inherently safe. Advantages include operating flexibility, low cost, quiet operation, assists ventilation and ease of automation. This system does not damage the coal particle; though a breaker may need to be placed before the vacuum loading system to reduce oversize.

**MINE PLANNING**

As sites implement continuous haulage, mine planning needs are to be considered. Besides panel design, the sequencing needs to be analysed and developed to optimise the productivity, recovery and utilisation of the new technology and mining operational requirements. The selection process of the potential continuous haulage systems needs to consider matching mining and outbye equipment production compatibility. To optimise utilisation, the continuous haulage system will need belt moves and installations ‘as and when needed’. Since continuous haulage requires a process driven culture, maintenance and operational skills need to be dispersed over all shifts.

Continuous haulage systems are less flexible than batch haulage systems, with mine planning constraints evident where variable geology might be encountered, especially in bord and pillar mining.
When considering wheel driven continuous haulage systems, wider tread pneumatic wheels reduce damage to soft floors. Covers on detection sensors reduce their downtime due to obstruction from dust and mud. Soft floors are damaged more by batch haulage than by continuous haulage systems. Track mounted continuous haulage systems are particularly effective with soft floors.

Continuous haulage systems often can traverse 90° drivage, though 70° angled cut-throughs are preferred to facilitate material handling. This necessitates the formation of diamond shaped pillars, which may in some circumstances be prone to crushing on pillar ends. This may result in larger intersections than would otherwise be preferred.

Other considerations to improve cycle times include: dry and graded outbye roadways, water inflow management, panel move standards, mapping of tasks and resources using Gantt Charts in precise and clear language, and timely feedback for continuous improvement. Water inflow has to be kept to a minimum, pumps have to be installed close to the water sources to protect the road and mud has to be addressed before its formation. This effort is worth the trouble, making most of the other processes faster and easier. It protects the equipment, and keeps the safety and worker motivation higher.

**DISCUSSIONS**

Some mines have found it necessary to modify or reengineer the continuous haulage system as delivered by the OEM, to enable it to adapt to the specific conditions of the mine. These trial and error modifications have proved quite productive. Some of the other issues encountered with continuous haulage have been spillage and deterioration of minerals.

Spillage can occur as every transfer station is a potential source for spillage. Whilst conveying up to 10,000 t per shift, even as little as 0.1% of spilled material (10 t) necessitates an expensive cleaning exercise. Spillage can make tramming a problem. In confined spaces, manual labour is often the only option to removing spillage. Rubber-belts are prone to retain sticky materials and the application of multiple cleaning stations is in many cases not technically feasible. Because of these factors chain conveyor systems are more commonly used, but these are prone to wear and tear.

Deterioration of minerals creates fine particles causing major loss of revenue, especially to the coal industry. The more transfer waterfalls, the more fines. Chain conveyors cause an additional milling action, especially in the bottom layers of the conveyed heap.

Although the initial capital costs for continuous face haulage in some instances may be higher than batch haulage, increases in shift production and productivity with continuous haulage should offset these costs. The goal is to increase shift production of coal and reduce operating and accident-related costs enough to justify the initial purchase and long-term use of this technology.

In some mining conditions, continuous haulage may not be just an alternative to batch haulage, but the only means by which some coal seams can be extracted. It should be noted that haulage costs usually makeup 15% to 20% of the total operational cost of a section. Running steel on steel and transporting sandstone-laden ores causes high wear, and may reduce time between overhauls.

The preference for longwall production has highlighted the growing disparity with roadway development. It is necessary to adopt continuous haulage systems to improve the pace of roadway development. If longwall is to reach the operational goal of being fully automated, then continuous haulage in conjunction with appropriate support services such as monorails need to be introduced and developed.

The industry requirements for continuous haulage are:

- Life cycle of the continuous haulage system needs to be capable of performing as specified and seen to be reliable;
- High level of automation;
- Minimal manpower;
- Ease of operation;
- High level of reliability;
• Meet the legislative requirements of Australia;
• Design risk assessment, inspections and test plans to be supplied by manufacturer and may be audited by the mine operator;
• Installation of continuous haulage as a change management process including managing hazards, installation procedures supplied, competency training by supplier, supply of drawings, parts manual, supervised installation by supplier using competent personnel, with fit for purpose tools and parts.

To develop a benchmark for continuous haulage, the systems required are:

• Delivery tests need to include dimensions, observation of operation, record all functions, check all fluid pressure levels, check overload levels, test emergency stops and pilots, test thermistors and RTDs, test fluid flows, review compliance with all regulatory requirements, check polarity, test operational parameters, vibration testing, compatibility with other equipment onsite and check signage;
• Effective communication and documentation systems;
• Providing a risk assessment for operational use with compliance to site procedures, such as isolation;
• Design and operational compliance to Australian Standards, mine site standards and legislative requirements;
• Training compliance gap analysis;
• Mine extraction actual compared to planned, for the continuous haulage system panels;
• Mine schedule actual advance compared to plan;
• A significant change in safety and productivity;
• Opportunity for the OEM and customer mine operator to partner in design, development, and implementation of a safe and efficient system of work;
• Recognise and action the requirement to reengineer existing methodologies for advancing and retreating panel services (systems, equipment and procedures);
• Review current production and maintenance process monitoring measurement and analysis for application to the continuous haulage system operation;
• Determine appropriate set of Key Performance Indicators (KPIs) in establishing continuous improvement program.

CONCLUSIONS

The ‘optimal choice’ in any analysis is not always made for monetary reasons. Often decisions are made for safety, operational ease or the optimisation of engineering design.

Continuous haulage takes personnel out of shuttle cars, reducing ergonomic issues. Continuous haulage equipment implies safer operation, with relatively less movement of mine personnel and mobile equipment in the face area.

When compared with batch haulage systems, based on the removal of loading times alone, continuous haulage can achieve increases in the utilisation time. Continuous haulage takes the batch haulage bottle neck out of the coal clearance system.

Bolting constraints need to be addressed to complement improvements in continuous haulage. This may require significant design changes to the “bolting machine that mines coal” or the “miner that bolts”. The first principle of design demands consistent steady state production rather than peak throughput capacity.

Continuous haulage systems are required to complement the current equipment and roadway dimensions used on mine sites. Continuous haulage systems are not yet completely compatible with
present development practices. Currently, a common effective mining cable length is 200 to 300 m. Since cables and hoses on the monorail can be extended only so far, this effective length needs to be considered when selecting the pillar length and should be a multiple of the pillar length. If not the monorail has to be relocated more often than necessary.

To improve continuous haulage, issues to be addressed on site include communication, education and the scheduling of tasks. The benefits of scheduling analysis should be able to show potential options for decreased costs, improved productivity, safety and finally increased return on investment.

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AUTOMATED BOLTING AND MESHING ON A CONTINUOUS MINER FOR ROADWAY DEVELOPMENT

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ABSTRACT: Automated installation of primary roof support material has been identified as potentially being capable of increasing productivity and operator safety in the roadway development process within underground coal mining. A series of reprogrammable electromechanical manipulators have been designed to be integrated onto a continuous miner platform to automatically handle the installation of roof and rib containment consumables therefore removing the operator from the miner platform and dangerous face conditions during operation. The system has been demonstrated above ground in a laboratory environment and has proven to be capable of achieving cycle times that are consistent with that required to support high capacity longwall mines.

INTRODUCTION

Roadway development in underground coal mines is a unique process. The methods used to extract coal and support the exposed strata have slowly evolved over the past century by keeping the fundamental concepts of major machinery and incrementally modifying the designs and processes to make small improvements. As a result, the process has become very restrictive for further innovation; especially in the areas of automatic operation and control. The existing machinery used to drive roadways has been specifically designed to accommodate the harsh and challenging environment that it operates within and any changes to their fundamental design can be counterproductive to increased output. Therefore, this machinery still requires high levels of human intervention during its operation.

Subsequently, today’s roadway development rates are failing to keep pace with modern longwall systems, and the methods currently used are proving to be inadequate if the industry is to progress to higher and more profitable production. Through a series of industry surveys (Gibson, 2005) the bottlenecks which restrain improved production and safety of operators have been identified and the manual strata support activities on a Continuous Miner (CM) have been acknowledged to be a major contributor to these constraints.

Primary support is necessary in most Australian mines for the geotechnical support of strata as roadways are driven. This activity is predominantly carried out manually onboard the continuous miner with the help of semi-automatic machinery. The manipulation and loading of consumables into this machinery is a manual process and existing roadway development systems and practices continue to rely on a high level of human intervention, including:

- Manually transferring roof mesh from a storage rack on the continuous miner discharge boom to the face and rotating the mesh into position over the roof bolting rigs;
- Carrying rib mesh along the side of the continuous miner and positioning it against the rib behind the rib-spall protection shield;
- Manually inserting drill steels into drill chucks and operating the roof and rib bolting rigs via push button, semi-automated drill rigs;
- Manually handling roof and rib bolts and their associated washers and chemical anchors, and installing them in the bolting rigs/drill holes after changing out the drill steel for the bolt installation chuck or tool;
- Operating the roof and rib bolting rigs to mix the chemical anchors and subsequently tension the roof and rib bolts, and
- Repositioning the roof and rib bolts for subsequent bolt installations or withdrawing the rigs to a safe position prior to proceeding with coal cutting.
These activities are typically undertaken within five metres of the immediate face in a confined working environment and often in close proximity to rotating equipment - hence the high number of injuries reported for personnel operating on or around continuous miners (Burgess-Limerick, 2009), and the subsequent introduction of Mine Design Guidelines (Industry and Investment, 2010) addressing risks associated with operation of bolting rigs, operator fatigue, and musculo-skeletal disease.

Historically, the drilling equipment on the CM is regarded as ancillary and has been adapted to fit confined spaces as a secondary activity to the CM’s fundamental function; i.e. to cut coal. Insufficient consideration has been given to operator ergonomics, which has resulted in less than ideal work stations for operators having to use powerful drilling machinery and manually manipulate awkward and cumbersome consumables from onboard storage modules or from the rear of the miner. Although some high output Australian mines have routinely reported roadway development rates as high as 8 to 10 m per operating hour in short bursts, human fatigue generally restricts operators from achieving these high rates over the longer term.

Over the past 2 to 3 years Self Drilling Bolts (SDB) have been trialled at a number of mines (Gibson, 2007; Gray, 2007; Bayerl, et al., 2009) and the benefits of a one-step drilling/bolting process are now becoming more widely accepted despite the substantial cost differential when compared to conventional resin anchored bolts. Further refinement in SDB technology to develop tensionable bolts, cut-able bolts, and to develop bolts less prone to binding/blockages in clay bands are expected to result in their wider adoption. When compared to the conventional drilling and bolting process, the reduction in the number of steps required for the installation of the SDB makes the one-step bolting process conducive to automation.

At least two CM Original Equipment Manufacturers (OEMs) now offer push-button semi-automated bolters for roof and rib bolting, and both are reported to be developing a second generation electro-hydraulic bolter which could, with SDB, be incorporated into an automated bolting system. Joy Mining Machinery has also demonstrated an automated bolting cassette which is utilized to feed drill steels, bolts and chemical anchors from the cassette into a bolting rig. Researchers are not aware of any on-board bolt handling systems being developed other than the prototype system that was included in CSIRO’s Autonomous Continuous Bolting Machine (ACBM) (Kelly, et al., 2001).

In 1986, the US Bureau of Mines (USBM) embarked on a major research effort to develop technology that could substantially reduce worker exposure to face hazards by simply relocating the equipment operators to an area of relative safety (Hill, et al., 1993; Schnakenberg, 1997). This Reduced-Exposure Mining System (REMS) research aimed to develop better mining practices using computer-assisted teleremote operation of continuous mining machines, haulage systems and roof bolting machines. However, for various reasons industry has failed to take up these technologies. In the hard rock and civil sectors, bolting and meshing automation has been used to a limited extent for large aperture roadways or single gateroad delivery.

Automation has been incorporated into the CM cutting cycle and CSIRO are currently developing self steering technology for the CM (Reid, et al., 2010). However, relatively low level automation has been applied to the drilling process and little or no automation has been applied to the manipulation of consumables on the machine. In fact, mechanisation of any kind has had little impact on the manipulation of consumables in the modern mine. Therefore a need exists for the purpose built mechanisation of manipulating these materials in order to introduce consistent repeatability into the process and relieve the operator from the physically demanding activities operators experience from day to day.

This work aims to contribute to the Australian underground coal Industry’s vision to achieve rapid roadway development production rates from a continuous miner of at least 10 m per operating hour and utilisation rates of 20 h per day. A key enabler of this vision includes the automation of the primary roof and rib support activities associated with roadway development. This paper discusses the challenges in automating the primary support process, and how a solution to these challenges has been reached. Each of the manipulators used to relieve the operator from their manual tasks are discussed in detail and the results of their performance in a set of laboratory surface trials analysed. Finally, the full roadway development process is discussed as an integrated system and what benefits this may provide in the future design of high productivity systems.
DESIGN FOR AUTOMATION

For every 1 m advance in a typical roadway development cycle (using a six roof bolt, four rib bolt per metre bolting density), 23 consumable items need to be handled consisting of 10 bolts, 10 washer/plates, 1 steel roof mesh module (5.2 x 1.1 m), and two steel rib mesh sheets (1.2 x 1.6 m). This number increases when installing cuttable consumables which are also considered in the design. To manipulate these items, a total of six subsystems have been developed, which include:

1. Bolt delivery system
2. Washer delivery system
3. Roof bolt manipulator
4. Rib bolt manipulator
5. Roof mesh manipulator
6. Rib mesh manipulator

The automated system discussed in this paper has been designed to be integrated into the fundamental framework of a continuous miner platform as a retrofit arrangement. This approach was taken to reduce the scope of reengineering required in developing a next generation CM platform and therefore reduces the overall risk of an automated solution not achieving its desired objectives in the short term.

By using this approach several engineering technical challenges were identified and considered. These include:

- There is very limited space available for attaching additional infrastructure to the bulk of continuous miners used for Australian conditions, particularly in mines with lower mining heights and/or narrow roadways;
- The variation in continuous miner models, frame sizes, and specialised configured layouts makes it difficult to design generic automation equipment across all platforms;
- The restricted use of materials, and non-approved electrical, servo-control, actuation and computer processing devices;
- The significant increase in rates of supply of consumables required to support higher development rates, potentially making it difficult to use onboard storage or buffered delivery systems with the existing continuous miner designs. This issue is compounded when accommodating any additional infrastructure required for installation of self drilling bolts;
- The variation in material consumables used from mine site to mine site;
- Adverse environment including rock falls, dusty conditions, corrosive water ingress and vibration all pose a technical challenge for the robustness of any proposed manipulation equipment, sensors etc;
- Mines, where strata support activities are routinely modified to adapt to challenging geotechnical conditions, have high variability or high density support in their process standards are anticipated to be difficult to automate;
- Many existing onboard services such as electrical supply and hydraulics would need to be modified, while new services such as pneumatics may be required as a powered source for some automation devices;
- Very limited space for movement and storage of consumables both at the face and in outbye areas;
- Extension of ventilation ducting and monorail segments is currently a manual task within the development process and warrants development of a means to eliminate or automate these tasks, including suspension of ventilation ducting on continuous haulage systems (where employed) or monorails;
- Restricted access to OEM control software and modification of the fundamental design;
- Risk associated with changing the fundamental major design of continuous miners.

DESIGN CONSTRAINTS

The automation system in this paper was designed to suit mine roadway dimensions with a minimum roof height of 2.8 m and minimum width of 5 m; a strata support bolt density of six roof bolts and four rib bolts per metre advance. The system also relies on using either one of two types of SDB. However,
to reduce the risk of any competing SDB not taken to fruition, this project has designed manipulation equipment to be as generic as possible and to accept either the Peter Gray® or Hilti® Style bolts (Figure 1). Although these two bolts are significantly different in diameter, the same manipulation principles can be used. In the latter constraint, both SDB technologies either consume more storage space than conventional bolts; or at least the resin systems associated with their one step process consume considerable space.

![Figure 1 - (a) Peter Gray® (Ground Support Services) SDB, and (b) Hilti® OneStep SDB](image)

As mentioned in the list of technical challenges, several issues are apparent when analysing the optimum method for storing and delivering consumable materials to the continuous miner. For those mines capable of achieving rapid roadway development rates in excess of 7 m per operating hour (MPOH) and approaching the target rate of 10 MPOH, material delivery becomes a significant challenge. It is calculated that for 10 MPOH advance, more than 1.4 t of consumables is required per hour using the Peter Gray® SDB. When using the Hilti® SDB, the weight of material for 10 MPOH exceeds 1.6 t. This equates to 40 t of material required to complete a 100 m pillar cycle.

It is anticipated that the increased volume of materials to be handled requires new methods for loading the continuous miner. It becomes clear from a process point of view that the numbers of bolts and mesh required to service the rapid roadway development process would deplete the physical capacity of existing onboard storage methods at too great a rate for them to remain practical.

When considering the use of the Hilti® SDB, this problem is compounded due to the increased diameter of each bolt reducing the amount able to be stored in onboard pods. For the Peter Gray® type SDB, additional pumping infrastructure and possible resin storage remain issues for the competing availability of onboard space.

**Bolt delivery system**

To assist in the controlled distribution of bolts, an automated bolt delivery system has been designed to selectively supply either roof or rib bolts to the respective bolt manipulators. The delivery system accommodates one metre’s advance and is designed to have a limited surge capacity of an additional two roof bolts for circumstances where additional spot bolting is required.

The bolt delivery system is loaded during the one metre advance cycle with a maximum capacity of 11 bolts each side of the machine, consisting of six roof bolts and five rib bolts. Gravity and a series of controlled access points are programmed to allow the bolts to fall onto a common centreline before being conveyed towards the front of the continuous miner. The sequence of delivering either a rib or roof bolt is controlled by a simple logic control. The system is positioned on either side of the continuous miner (LH and RH sides) and an image of the device is shown in Figure 2.

![Figure 2 - Bolt delivery system](image)
Automated washer delivery

During one metre advance, several styles of washers are generally required. These include either a roof bolt washer, rib bolt washer and/or cut-able polymer washer. The industry currently uses several variations in style of washer and Figure 3 shows some of these. For standardisation, two steel washers and one polymer washer adapted for the Hilti bolt were chosen for the delivery system.

Similar to the bolt delivery system, the washer delivery system selectively dispenses the correct washer on demand. For this to occur, two conveyor lines are incorporated into a single unit which transfers washers from the interface point at the rear of the machine. One conveyor transfers a roof bolt washer, whilst the other a rib bolt washer. At the front of the delivery system (see Figure 4), a flip and grab mechanism selectively retrieves and positions each washer in line to the bolt centreline.

Once dispensed, the respective roof or rib bolt is conveyed from the bolt delivery system through the centre of the washer where it is captured and secured before the bolt is manipulated into the drilling rigs.

Figure 3 - A selection of various washer designs used by the Australian coal industry

Figure 4 - Washer delivery system

Roof bolt manipulator

The manual installation of roof bolts into onboard drilling rigs represents a considerable portion of manual labour used to support the roadway development process. A skilled operator has the ability to efficiently follow a repetitious cycle of drilling and loading bolt and chemical consumables during a drill cycle. A typical drill cycle is comprised of the following steps:

1. Setting the drill rigs into position;
2. Loading the drill coupling with an adaptor dolly;
3. Loading the drill bit and collaring the hole;
4. Initiate auto drill and retract cycle;
5. Remove the drill bit and adaptor dolly;
6. Loading of a washer plate on the timber jack head plate;
7. Install a chemical resin sausage;
By using a self drilling bolt, a number of these steps (2, 3, 5, 7) can be eliminated whilst the remainder of these steps can be automated and therefore relieve the operator of these tasks. The combined reduction in process steps and automation creates an opportunity to reduce drilling cycle times and eliminate operator exposure to high-powered machinery.

Several considerations were identified as being important specifications for the design of a bolt manipulator. As mentioned, the manipulator is required to handle both the Hilti® and Peter Gray® SDB. Secondly, in the event where manual roof support installation is required, the mechanism designed is required to occupy as small a volume as practical and can be removed or pushed aside from the work station to allow conventional access for operators during specific operational times. For example - manual operation may be required in difficult or high density bolting areas, such as intersections. In this example, it would be difficult to utilise automation as a result of high variability with bolt and mesh placement.

The design shown in Figure 5 illustrates a robotic type manipulator capable of accepting either SDB bolts one at a time. The roller feed system on the feed entry of the manipulator allows this to occur and simplifies the linear actuation of the bolts. After a bolt is loaded the mechanism is programmed to automatically place a bolt in either of the two vertical drilling rigs. Once the bolt is engaged in the drill rig mast, a rig mounted clamp supports the bolt whilst collaring and finally in-place drilling occurs.

For the bolt manipulator to function automatically, bolts are conveyed from the rear of the machine along the common centreline. This reduces the amount of handling equipment and Figure 5 illustrates the bolt flow required from the rear of the machine forward. The cylindrical shape of a bolt allows them to be simply conveyed longitudinally using a compact polymer roller mechanism located on the same centreline as the bolt delivery system.

**Figure 5 - Roof bolt manipulator positioned on a JOY 12CM30**

Operation of the roof bolt manipulator has been tested and results indicate that a single roof bolt, once conveyed into the rear of the manipulator, can be loaded into one of the two vertical roof bolting rigs as well as attaching a washer plate within 20 seconds. This includes returning to the home position in preparation for reloading.
Rib bolt manipulator

The rib bolt manipulator is designed to be as compact as possible and is located on the side of the hydraulic rib bolting rig as shown in Figure 6. The hydraulic rig requires at least a homing position for the bolt to be loaded. For this unit, the rib bolter has three positions for one bolt to be installed above horizontal and one below. The third position is the horizontal, and is used for loading the rig.

Similar to the roof bolt manipulator, the rib bolt manipulator uses a set of pneumatically controlled roller mechanisms to longitudinally convey a bolt from the common centreline. The washer is attached during this stage and the bolt is secured during the cycle.

Automated roof mesh manipulator

Manual methods for roof mesh installation are a physically demanding and a relatively constrained activity whereby mesh is typically man handled out-bye from the rear of the machine up over the centre coal conveyor before being rotated normal to the roadway and up onto the drill rig head plates. Alternatively for some frames, onboard storage requires mesh to be retrieved from a stack of mesh stored above the centre conveyor, rotated and placed upon the Temporary Roof Support (TRS). Both of these processes are required to transverse the mesh forward to the drill rig operator's platform before rotating the mesh and placing it in a bolting position.

Using the automated manipulator, mesh is conveyed from an interface point at the rear of the continuous miner where it is conveyed to a forward position by a chain conveyor. The manipulator is divided into two sections to allow backslide operation of the rib bolters for rib bolts greater than 1.2 m length.

A rotational turntable elevates and turns the mesh 90° whilst the front manipulator section transfers the mesh above the drilling rigs. Once in position, the roof bolting drill rigs autonomously carry the mesh to the roof whilst the loaded bolts locate through the apertures of the held mesh. After drilling, the mesh manipulator returns to the home position ready for the next cycle.

Automated rib mesh manipulator

Typically only one piece of mesh is required on each side of the miner for one metre advance. However, for the purpose of a demonstration and testing, the rib mesh manipulator has been designed
to store up to 10 pieces in a storage unit. Each piece of mesh can be automatically dispensed on demand. The storage unit separates a single piece of mesh onto a transfer arm (see Figure 8 (a)) whereby a concealed chain conveyor under the miner’s side access platform, transfers the mesh alongside the machine and in front of a modified rib crash barrier.

The rib crash barrier has pneumatic gripping cylinders that grab the mesh once the transfer arm has conveyed the mesh into position. Once secured, the mesh is linearly extended upwards whilst simultaneously the crash barrier extends towards the rib (see Figure 8 (b)). This allows the mesh to be positioned within the top corner of the rib and roof, and prevents any fouling with existing roof and rib anchored bolts. At this point the rib bolt is then positioned within the rib bolter and the drilling cycle commences.

![Figure 8 - Rib Mesh Manipulator: (a) cassette dispensing unit and transfer arm, and (b) crash barrier in the extended out position](image)

**SYSTEM INTEGRATION AND LABORATORY DEMONSTRATION**

Figure 9 illustrates the integration of the complete system on a laboratory test frame for which the entire system was mounted and tested. The dimensions of the frame represent the major mounting points of a Joy 12CM30/32 CM and allowed for the convenient experimentation in a laboratory setting. It is expected that the modular equipment can be removed and reattached to a real CM platform for future underground trials.

The laboratory demonstration has proven that a solution for automated bolting and meshing is achievable. Some major findings from the demonstration include:

- Automation of bolt and meshing activities is possible on a continuous miner without having to change the fundamental design of the miner platform and its operation;
- The cycle times achieved by the integrated automation are consistent with increased roadway development rates of at least 10 MPOH;
- An appreciation of the scale of mechanisms, actuation and control required to automate the process.

![Figure 9 - Laboratory demonstration and testing unit (a) CAD drawing, (b) Physical unit](image)
CONCLUSIONS

The design for the automation of manual tasks on a conventional CM is not a trivial task and several constraints specific to the industry need to be considered. By using a laboratory above ground demonstration a satisfactory solution for automatic control and manipulation of roof and rib support materials now exists and shows that the prototype machinery and control can potentially remove the operator from the dangerous face conditions in an underground production environment.

This work has identified and quantified the time savings that can be achieved through automated repeatability. When taken to full fruition, cycle times, and therefore overall development rates, are expected to be improved inline with the target of 10 m per operating hour.

Finally, the prototype design and laboratory test facility have significantly reduced the technical risk in proceeding forward and adapting the automation to a continuous miner with the intent to conduct a more substantial trial within an underground production environment.

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ANALYSIS OF THE MACQUARIE MONORAIL SYSTEM IN DEVELOPMENT OF GATE ROADS AT MANDALONG MINE

Braedon Smith\(^1\) and Paul Hagan\(^2\)

**ABSTRACT:** Gate road development lies on the critical path in the longwall production process. To address this issue industry has invested substantial resources in developing technologies such as continuous haulage and bolting cycle automation. There has been limited focus on improvement of ancillary development processes and technologies such as the provision of face and panel services and the panel advance process.

A monorail system has been in use for some years at Mandalong Mine near Lake Macquarie in N.S.W. The Macquarie Monorail system is an integrated gate road development services unit and differs from many existing monorail units employed in underground operations as it is designed to manage all face services while as well as carrying heavy panel plant such as the section load centre and auxiliary fan.

Use of the system at the Mandalong Mine has resulted in increased productivity and safety performance during gate road development. Reductions in service move and operating delay times of 32\% and 25\% were observed respectively along with a 70\% reduction in manual handling injuries.

The system is not without its disadvantages chief being significant manual handling is necessary in the erection of the relatively heavy monorail structure and replacement of the flexible ventilation ducting. Despite these minor limitations of the system, the overall safety and productivity performance of gate road development was found to be superior to conventional development units.

**INTRODUCTION**

As longwall productivity has increased over the years, it has not been matched by gate road development with advance rates often being insufficient thereby inhibiting the potential rates of longwall retreat (Mitchell, 2009). The majority of research undertaken to address this issue has focused on the development of capital-intensive technological solutions such as automated bolting and continuous haulage systems (Gordon, *et al.*, 2008). This approach has largely failed to recognise the potential benefits that might be achieved by eliminating or reducing process delays in roadway development such as provision of services, flit times and panel advances.

In addition, while there have been significant improvements to ergonomics within the mining industry in recent years, there is still considerable scope to reduce and in some cases eliminate manual handling tasks thereby contributing to improvements in operator safety. Manual handling injuries are one of the most common causes of significant injury in underground mining operations in Australia (Burgess-Limerick and Steiner, 2007).

**THE MACQUARIE MONORAIL SYSTEM**

**Overview and description**

The Macquarie Monorail (MM) is a roof mounted, integrated development services system that aids in the management and mobility of face services and heavy equipment. The system was first employed at Centennial Coal's Newstan Colliery operations in 2006. It was subsequently relocated to the Mandalong operation in 2009.

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One of the primary design objectives of monorail systems is management of face services such as cables and hoses. In the case of the MM system these services include electrical power, air, water, return water and ventilation as illustrated in Figure 1.

![Image of cables and hoses](image1.jpg)

**Figure 1 - The conventional services handled by the Macquarie Monorail system**

An added feature of the MM system is its ability to also handle the section load centre as well as auxiliary fan as illustrated in Figure 2 together with other ancillary equipment including the electric pump starter and fire depots. This eliminates the need to disconnect these services and relocate the heavy equipment during the process of each panel advance. Hence reducing, if not eliminating, the risk of injuries associated with manual handling of services as well as the time to complete each panel advance.

![Image of load centre and fan](image2.jpg)

**Figure 2 - Additional outbye services handled by the Macquarie Monorail including the load centre and fan**

The entire system is mobilised by a series of manually operated pneumatic traction drives that allow the system and its integrated services to be advanced and retracted as required.

**System configuration in panel development**

A typical layout in a gate road development employing the MM system is shown in Figure 3.

The MM system is currently configured to enable gate road development of up to three pillar lengths in each development cycle before a move of the panel transformer is required. Typically disconnection, relocation and reconnection of services is not required within each cycle.

On completion of a panel transformer move, the section load centre is reconnected to the transformer through the Macquarie Monorail. The other services including compressed air and water are separately connected to service outlets. This arrangement allows extension of services and the high-tension cable to occur independently of the pillar production cycle.
Once development of three pillars is complete the transformer is relocated and reconnected to the monorail as the next transformer move cycle begins.

**Figure 3 - Panel configuration using a Macquarie Monorail system**

**ANALYSIS OF PRODUCTIVITY PERFORMANCE**

A study was undertaken to compare the performance of the MM system against conventional development without the use of a monorail system in similar conditions. The investigation centred on two adjoining gate roads designated as panels MG11 and MG12 that were developed by conventional methods and the MM system respectively. The study spanned development of 26 pillars in each case which equated to approximately 7 km of development and a similar number of panel advance cycles.

Overall it was found development undertaken with the MM system was completed in 303 days which was 26 fewer days than with the conventional method. This equates to an increase in production rate of approximately 8% or on average an additional 2.16 m advance in every 24 h period.

**Key delay groups**

An analysis of the reported delays in each of the two development roadways was undertaken. As is shown in Figure 4 there was an appreciable reduction in the length of many of the different categories of delays with the MM system chief among these being service moves or panel advances delays and operating delays. However there were slight increases in the delay categories designated as production and plant maintenance.

**Figure 4 - A comparison of key gate road delay groups**

One of the greatest improvements was the reduction in panel advance delays where use of the MM system resulted in a 32% reduction representing 285 h. This result can be directly attributed to the design of the monorail system and management of face services.
Of particular note was completion of a panel advance in less than ten hours which was achieved on five occasions as is indicated in Figure 5. In one particular 12 h shift the development crew not only completed the whole panel advance but went on to cut 10 m of coal. Also worthy of note was the gradual improvement in the whole process of a panel advance as evident by the decline in the moving average of the duration time shown in Figure 5.

The reduction in operating delays by 327 h represents a 25% improvement. This was achieved through the simplification of the flitting process and elimination of the need to install rigid ventilation ducting. A slight reduction in electrical delays was also observed due to the reduction in incidence of damage to electrical cables.

Production delays however increased by 100 h or 22%. This increase was primarily due to replacement and cleaning of the flexible ventilation ducting. The ventilation arrangement in the MM system includes a ‘drop box’ located immediately behind the continuous miner. This ‘drop box’ was limited in its ability to filter dust and moisture from the ventilation air. This caused a build-up of fines in the flexible ducting which reduced the ventilation efficiency and necessitated frequent downtime for cleaning and/or replacement. This meant an increase in risk associated with manual handling with change-out of components in the ventilation system.

As the monorail system added to the inventory of equipment in gate road development, the call on planned maintenance increased by 75 h.

ANALYSIS OF SAFETY PERFORMANCE

A comparison between conventional and MM system development showed there were 37% fewer incidents and injuries in using the monorail system. The reduction in injuries alone was found to be 31%. Significantly no instances of medically treated injuries or suitable duties injuries were recorded and there was a reduction in the severity of most injuries.

Of particular note was the reduction in manual handling injuries by 70% as seen in Figure 6.

Manual handling injuries

Within the category of manual handling injuries a reduction in all causes of injury was found. Perhaps the most significant of which was the complete elimination of injuries associated with handling of the miner cable, as shown in Figure 7. As these injuries can often be severe, typically resulting in a suitable duties or lost time injury incident, this is considered as an excellent result by management. It was also found that injuries related to ventilation tube installation were reduced owing to the flexible ventilation system incorporated in the MM system.

It should also be noted that a ‘light’ monorail unit was utilised within the conventional gate road unit, solely for the management of the ‘elephant’s trunk’ between the continuous miner and the rigid ducting. Some manual handling injuries were attributable to the use of that system, more so than with the MM
system. This is significance as the structure on which the MM is mounted is far more substantial than the 'light' monorail system where each segment of monorail has a mass of 30 kg.

![Figure 6 - A comparison of recorded injuries by type](image1)

![Figure 7 - A comparison of Manual Handling Injury causes](image2)

Despite the actual lower number of monorail structure-related injuries recorded with the MM system there was a greater potential risk to operators in the repetitive lifting of each rail segment due to its weight and shape. As a result, a lighter weight structure should be developed to mitigate this risk to operators.

**PROCESS IMPROVEMENT**

In order to determine the areas where improvements could be made in the performance of the MM system, a systematic analysis of the transformer cycle process was undertaken. It was found that the MM system underperformed during the hole-through process between headings, as shown in Figure 8. It was identified that this poor performance may have been due to the lack of a formalised procedure and sequence for the process. To address this, two sequence plans were developed in consultation with operators and staff these were then trialled to determine the optimum method of holing-through. However, due to panel relocation, the results of these sequences have yet to be finalised.

It was also found that operators frequently encountered difficulty in the installation of curved monorail structure during the hole-through process as it relied on judgement of the operators in the installation of the curved structure to join with already installed structure. If operators installed any one of the curved sections of structure in an incorrect location or orientation then the structure would not join to the existing structure as shown in Figure 9. To address this a lightweight ‘template’ of the curved monorail
structure was eventually developed to aid operators in identifying the correct installation location while also reducing the manual handling risk to the operator.

![Figure 8 - Transformer cycle productivity analysis highlighting poor hole through performance](image)

**Figure 8 - Transformer cycle productivity analysis highlighting poor hole through performance**

![Figure 9 - Examples of correct and incorrect monorail curve structure installation locations](image)

**Figure 9 - Examples of correct and incorrect monorail curve structure installation locations**

**FURTHER OBSERVATIONS**

**Ventilation efficiency**

The in-bye air filtration system unit used in the ventilation system resulted in build-up of fines in the exhaust ducting. While this issue was manageable at the Mandalong Mine, it is likely to be of greater concern in mining operations located in warmer climates where heat management at the development face can be more challenging. In these situations it is possible that poor ventilation performance would not only affect operator safety but also reduce productivity.

Research is currently underway sponsored by both mine management and equipment manufacturer to find an adequate solution to the issue.

**Flexibility**

As identified in the transformer cycle analysis, without proper process management, performance of the monorail system can decline when changing drivage direction. This is an issue that can be mitigated through effective management.

This study was limited to investigation of development of a two heading gate road with a single continuous miner. While a similar system has been developed for multiple heading development panel and a version of the system is currently being investigated for use in multiple miner panels, effective application of the technology in these circumstances is yet to be studied.
Structure weight

The mass of each segment of monorail structure is approximately 30 kg. In addition to the potential for serious crush-pinch injuries, the repetitive nature of structure installation increases the potential long-term health risk to operators. It is recommended that a rigid, lightweight alternative to the existing monorail structure be developed to further improve operator safety.

CONCLUSIONS

Overall gate road development using the Macquarie Monorail system was found to be superior in performance to conventional gate road development in terms of both productivity and safety performance at Mandalong Mine.

Despite this positive result the system is not without its flaws, and there exists potential for further performance improvements with a focus on system design and management. As such, it is recommended that systematic and targeted continual improvement of the system be continued to further facilitate the successful application of this technology.

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DESIGN FOR AUTOMATED SELF ADVANCING MONORAIL

Luke Meers and Stephen van Duin

ABSTRACT: This paper discusses the need for the automated installation of monorail sections which are used for the support of services and ventilation ducting to the continuous miner during roadway development in underground coal mining. The paper analyses the current state of development monorail technology and identifies how self extending monorails would complement new technologies emerging in automated roadway development, resulting in increased advance rates and improved safety. The constraints and challenges in designing for an automated solution are identified and discussed.

INTRODUCTION

Research and development into the automation of the roadway development process in underground coal mining is well under way. The Australian Coal Association Research Program (ACARP) has developed a strategy for a high capacity roadway development system. This includes a key goal known as CM2010, aiming to achieve development rates of 10 m advance per operating hour (MPOH), 20 h per day (Gibson, 2010). Part of this planning strategy has been the identification of key enabling technologies required if the development rate goals can be realised. As a result, the industry has funded research addressing the automated handling of bolts, mesh and washers, technology for self steering miners as well as polymer roof support technologies. Upon implementation, these new technologies will mark significant steps forward in the roadway development process; improving advance rates and eliminating the need for operators at the roadway coalface.

A critical element of roadway development is the efficient utilisation of resources and an effective supply and handling mechanism for face services. The interaction between these activities affects the overall performance of roadway development. Furthermore, high cutting rates as envisioned create smaller windows of opportunity to complete necessary activities such as the installation of support infrastructure and extension of face services. The automation of monorail installation would improve the development process by:

- eliminating the need for personnel to manually advance the monorail, thereby reducing heavy manual handling and exposure to ‘no-go’ areas;
- integrating the automation of the monorail with other automation activities being developed for the continuous miner; and
- increasing the installation rate of the monorail inline with a high capacity continuous miner development system.

An industry analysis has been performed to assess the current state of monorail technology. Monorail technology has been utilised in the mining industry since the late 19th Century (Oguz and Stefanko, 1971). Whilst there is much evidence of monorail technology assisting in production safety and efficiency (Coppins, 2008), what is absent is any literature detailing innovation in the automated installation of monorail, either in the mining or any other industry. Currently, monorails are used in underground coal and ore mining as a means of transport (Guse and Weibezahn, 1997), batch haulage (SMT Scharf EMT brochure, 2011; Anon, 2011) and as a means of managing services to the longwall face (ACARP, 2011).

Development monorail systems are less common. In Australia, there are two main varieties of systems available tubular section monorail; and I-beam monorail. The tube section monorail is lighter (approx 15 kg per 2 m rail), lower in cost and is a simpler system capable of carrying light weight services such as air, water, power cables and ventilation ducting (Appleby, 2011).

The I-beam system can support higher loading than the tubular profiles, allowing it to carry heavier equipment such as pump and fan units (Macquarie Manufacturing, 2011). Critically, it is also designed...
to remain in the belt road after development to support services to the longwall face. Figure 1 shows a typical layout for the I-beam monorail system including multiple drive units.

![Diagram of development monorail schematic](image)

**Figure 1 - Development monorail schematic diagram (Coppins, 2008)**

Roadway development without monorail assistance requires the manual fitting of vent tubes and support infrastructure to the roof support at a rate matching that of development. Installation of a conventional monorail system eliminates the need for the manual handling of vent tubes and face services, reducing injuries (Shales, 2010), but replaces this with the manual handling of the typically 25-30 kg rails in the hazardous zones to maintain support to the coal cutting process. Issues that have been related to the manual handling include:

- Potential for back or trunk strain injuries, cuts, abrasions, contusions and crush injuries sustained while transporting, handling and restocking monorail sections and advancing services, particularly in mines where roadways are developed at heights in excess of 3 m;
- Exposure of personnel to the immediate face which is prone to outbursts, dust, noise and collapsing coal;
- Exposure of personnel to interacting processes such as coal cutting, coal wheeling and services advancement.

Manual rail installation becomes more onerous as mining heights increase, with operators facing increasing ergonomic risk as they lift monorail segments into place at height. The rates of development targeted by the industry, further compounds these issues. Furthermore, the work area currently utilised to install and extend monorail segments is likely to be required for functions of the automated continuous miner, necessitating these regions be designated “no-go” areas during automated operation of the bolt.
and mesh handling systems - hence an alternative approach to monorail advancement is required. A self advancing, automated monorail eliminates the need for manual handling of the rails from the miner, effectively removing the three key risk areas noted above.

A key driver for self installing monorail technology is the need to integrate the supply of services and ventilation to the miner with other automated roadway development equipment. The automated handling of roof support consumables on the miner (Van Duin, et al., 2011) utilises the deck space traditionally used to manually install the monorail. Additionally, the automated tasks involve moving equipment which would likely create “no go” zones, effectively eliminating any space on the miner from which rail could be manually installed.

Figure 2 shows one example of a timeline for the automated tasks used in the handling of the strata support consumables over a one metre cycle.

Figure 2 - Automated continuous miner, Task timeline (Van Duin, 2011)

The timeline illustrates that over the five minute cycle there is no time available where personnel would manually install a monorail beam. Manual installation of rail would require the full isolation of all mesh and bolt handling processes on the miner in order to allow the operators to gain access to the machine. The industry is pushing towards faster roadway advance rates and as such the interruption of the already critical bolting cycle is not a feasible option. However, an automated installation procedure could be designed to integrate with the process cycle of the miner allowing the cycle shown to continue uninterrupted and thus achieve maximum advance rates.

**DESIGN FOR AUTOMATION**

An automated self-installing monorail system requires the development of several key mechanical components. When considering the design of existing monorail sections and ancillary components, the following actions/movements, or equivalent would be required:

- Delivery of the rail to the rear of the continuous miner (one 2 m rail every 12 mins in a 10 MPOH system);
- Transfer of rail from the delivery system onto a rail handling device;
- Manipulation of the rail into the installed position;
- Connection of the rail to the previously installed rail;
- Manipulation of the attachment between the rail and roof support;
- Adjustment and locking of a height adjustable mechanism between the roof and rail,
Design constraints

The design of a self advancing, automated monorail system to integrate with an automated continuous miner inevitably involves the consideration of a large range of design constraints and challenges, these include:

**Installation location:** The strict requirements in most underground coal mines for ventilation require the ducting to be run up as close to the face as possible. Most modern continuous miners include sections of integrated ducting. The monorail, supporting the vent tube must therefore be installed inbye of the connection between vent tube and the onboard miner ducting, effectively alongside the rear of the miner.

**Available space:** The rail handling equipment must fit in the envelope between the miner and the rib whilst allowing for variation in rib conditions and the position of the miner. This allows for equipment no wider than 500 mm. The rail handling equipment must also be located and dimensioned such that there is space for it to move 2 m outbye relative to the miner before the space is available for the subsequent rail piece to be attached.

**Installation time:** The installation process must also be performed at the appropriate time to avoid clashes with the automated miner functions such as, movements of mesh and bolts. This would most likely be during the drilling phase because the miner is stationary and movement of consumables is restricted. This also necessitates data communication and logic control integration between the monorail system and the continuous miner.

**Volumes:** At 10 MPOH, one 2 m rail is required every 12 min. This involves not only installation but also delivery from outbye storage. At 30 kg per rail, a 10 h shift would require 1800 kg of stored rails.

**Roof support:** Current practice for I-beam monorail generally involves hanging from a dedicated roof bolt. If 10 MPOH rates are to be achieved, this is not likely to be possible due to the extra time required for dedicated bolt installation, thus requiring the system to be attached to primary roof support.

**Roof conditions:** The system must be able to compensate for uneven roof contours. Some automated adjustment must be possible to allow the rail to run smoothly despite roof conditions.

**Adverse conditions:** Dust, water ingress and rock falls necessitate that any design must be sufficiently robust to cope with the environment.

**Approved materials:** The system must comply with current standards for materials, control equipment and actuation devices and communications.

**Cut throughs:** In order to achieve a fully automated process, handling and installation of curved rails for cut throughs is required. This may also necessitate the ability to attach intermediate support along the curved rail to reduce offset loadings.

**Compliance:** The system must incorporate enough movement and adjustment to tolerate the dynamic loads which will occur during the tramming of the monorail based equipment. This includes sideways tilt and vertical give.

These constraints highlight the range of technical challenges which must be addressed to achieve a feasible self advancing monorail. There are several key features that are likely to be incorporated into the design.

Outer roof bolt placement will be critical along the monorail centreline. Whilst this is not physically part of the monorail system hardware, the equipment will be required to locate and attach to the roof support. Thus, a control system on the miner which can accurately reproduce the spacing of bolts in the roadway is necessary. Additionally, the control system of the continuous miner and the monorail will require integrated logic control in order to ensure correct positioning and timing is achieved.

Height control could be achieved via adjustment of the mechanism which attaches from the rail to the roof support. Current manual monorail systems rely on a length of chain being hung by the appropriate
link, but a more rigid mechanism would be necessary to enable automated handling. The height adjustment system must also lock in place, but allow enough compliance for rail alignment and dynamic loading as described previously.

The connections between rails would ideally be simplified from current systems. This link would not require actuation but would lock in place as the rail is inserted. This connection must also be capable of supporting the heavy loads of the multiple tonne trolleys that will be mounted on the rails. Vertical and horizontal angular compliance is also required, for slight changes in heading or height of the roadway.

At a development rate of 10 MPOH storage of rails onboard the miner, or even below the vent ducting, is unlikely. As such, the transfer of individual rails every two metres from a dedicated delivery system integrated with the delivery of the consumables to the miner would be most suitable.

This research will endeavour to progress with concept design work which incorporates the constraints and design criteria mentioned here, working towards the development of a complete concept for an automated, self advancing monorail system.

CONCLUSIONS

Monorail technology is common in many forms of mining, but to date there have not been any systems developed in which the installation procedure is automated. As advance rates in roadway development increase, the manual handling of services to the face and the manual handling of monorail beams themselves, becomes increasingly difficult. Automating the installation of monorail sections, creating a self advancing system, alleviates the safety problems of handling rails manually near the coal face and complements other areas of roadway automation currently being developed. However, by highlighting the issues that have led to the need for an automated approach, this paper has also identified the challenges relating to the design of an automated self-advancing monorail system. Now these have been categorised, the process for automated design can begin.

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A PRACTICAL INERTIAL NAVIGATION SOLUTION FOR CONTINUOUS MINER AUTOMATION

David C. Reid, Mark T. Dunn, Peter B. Reid and Jonathon C. Ralston

ABSTRACT: The outcomes achieved at the completion of a major industry-funded project undertaken by the CSIRO Mining Technology Group to advance the automation capability of continuous mining equipment in underground coal mining operations are reported. The details of a practical steering and guidance solution for autonomous Continuous Miner operation employing novel inertial navigation aiding techniques are described. The results of navigation performance evaluation using a scaled skid-steer mobility platform completing three segments of a two-heading roadway development pattern under autonomous control are presented. These results represent a significant milestone in 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 specific research outcomes reported in this paper cover 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). Initial navigation results in this CM2010 CM automation project, including an introduction to the technology and the experimental setup used to evaluate the navigation performance has been previously presented in Reid et al., (2011).

This paper presents updated results of the CM navigation system which has been recently tested on an above ground coal surface over a two-heading roadway development mining pattern which covers a total path length of over 900 m.
INERTIAL NAVIGATION TECHNOLOGY FOR MINING GUIDANCE

As described in Reid et al., (2011) all inertial navigation systems compute 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 has been developed, that builds a theoretical framework for optimally combining the inertial sensor data to compute 3D position and thereby a navigation solution. 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 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.

![Block diagram of Figure 1](image)

**Figure 1** - Block diagram showing the relationship between the IMU sensors and the aiding source used to compute the navigation output

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. As reported in (Reid, et al., 2011) much of the early project work focussed on the development of an 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.

Earlier in this project 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 radar were identified as providing individual and complementary advantages. Subsequent testing has shown that radar
yields significant operation and performance advantages over the other technologies. Radar technology has been developed and further optimised during this project to provide a practical and extremely accurate aiding source that operates at very low velocity and low latency.

**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 captures some of the CM dynamics in terms of motion profile, skid steer manoeuvring, wheel slip and jolting/vibration characteristics. In this figure the INS unit under test is mounted internally and the Doppler radar is mounted on the front far corner angled down towards the ground.

**Figure 2 - Phoenix skid steer vehicle used to evaluate the performance of the inertial navigation systems**

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 a 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 autonomously navigate to a mission plan under closed-loop control.

Previous navigation trials along a 55 m natural bush track have been report in (Reid, et al., 2011). Since then more elaborate and field-realistic experiments have been conducted which map out the path of a CM throughout three sections of a two-heading roadway development mining plan with a total path length of approximately 900 m. This mining pattern is shown in Figure 3.
These experiments have been conducted at the Ebenezer decommissioned surface coal mine nearby the CSIRO research facilities west of Brisbane. A large ex-stockpile area with a remnant coal surface was prepared and mapped-out with the two-heading plan as shown in the aerial view of the test site in Figure 4.

![Figure 3 - Image showing two-heading roadway development mining plan](image)

![Figure 4 - Aerial view of Ebenezer test site with waypoints and mining plan shown](image)

For each of these experiments the Phoenix test vehicle control system was pre-programmed with the coordinates of the 2-heading mining plan and the associated speed/direction profiles. The vehicle was then positioned at the starting point which the navigation system fully aligned and calibrated. The vehicle was then enabled to automatically navigate through the pre-programmed mine plan with regular
brief stops to allow the navigation system to ZUPT. The vehicle velocity profile throughout this 60 minute mission is shown in Figure 5.

RTK differential GPS data were recorded throughout the experiment to provide a baseline reference to determine the performance of the navigation system. The ground survey markings also provided a general visual measure of performance during the experiment. One of these yellow circular ground marks can be seen ahead of the Phoenix in Figure 2.

In these experiments a two-stoke petrol generator was installed on brackets rigidly connected to the inertial navigation mounting plate. With the generator running during the experiments this arrangement provided a more field-realistic condition to test the sensitivity of the navigation system to background vibration, especially during ZUPTing periods.

![Figure 5 - Phoenix velocity vs time plot to indicate the ZUPTs and forward/reverse motion during the two-heading mining plan](image)

**EXPERIMENTAL RESULTS**

Representative results from a recent navigation system performance trial at Ebenezer test site as described in the previous section indicated that the primary measure of system performance is given by comparison between the inertial navigation derived position estimate and the RTK DGPS measurement. The 2D position errors (orthogonal components of along track and cross track) for one recent experiment are shown in Figure 6. This result is indicative of three other similar experiments. As can be seen from this plot the overall error is typically less than 200 mm and importantly the error does not tend to increase with time over the 70 min duration of this experiment. The position at the furthest point of travel has reduced to approximately 200 mm. The beneficial effects of ZUPTing can be seen by the reduction in position error corresponding to times when the Phoenix is stationary. Furthermore it was observed that the path travelled by the Phoenix, which turned on each waypoint twice, is consistently accurate around the independently marked points.

![Figure 6 - 2D position errors (orthogonal components of along track in blue and cross track in red) between the inertial navigation system and on-board RTK GPS equipment](image)
DISCUSSIONS

Extensive practical evaluation of the performance of a VMS-aided inertial system in mine-realistic above-ground experiments using the Phoenix mobility platform travel in a two-heading roadway development mining plan over a total distance of approximately 900 m has yielded encouraging results. The results obtained indicate that custom-designed radar can provide accurate and timely velocity measurement necessary to achieve a practical VMS-aided inertial navigation system for automated control of a Continuous Miner in a roadway development application. Furthermore the robust non-contact nature of the radar technology and the field proven reliability of strapdown inertial navigation technology could provide the complete hardware solution for this application.

Research is continuing on the further development of the radar non-contact speed sensing technology to improve the robustness and measurement reliability by means of multiple sensors. It is expected that this approach will overcome the known limitations of this technology when the target is reflective such as pooled water.

SUMMARY

With the support of the Australian coal industry, CSIRO is currently involved in a large-scale continuous miner automation research and development project. A major outcome of this project to date has been the demonstration of a practical CM guidance system which combines high performance inertial sensors with custom-developed radar. This guidance system has been demonstrated using the Phoenix mobility platform at a decommissioned coal mine and has achieved a position error of generally less than 20 cm over a 70 minutes mission which followed three segments of a two-heading mining plan with a total path length of over 900 m.

Work continues to improve the underlying accuracy and reliability leading to full underground trials on a production mining system.

ACKNOWLEDGEMENTS

The authors wish to thank ACARP and the Roadway Development Task Group (RDTG) for their support and assistance in the development of this mining guidance system. Also Christian Singfield and the rest of the MezurX team for their cooperation in providing access to the Ebenezer test site.

REFERENCES

ADDRESSING RESIN LOSS AND GLOVING ISSUES AT A MINE WITH COAL ROOF

Peter Craig

ABSTRACT: Over the last year, Jennmar has been supplying roof bolts to a new mine with a thick coal roof in which massive resin loss was being measured. After trying various drill hole diameters, bolt profiles and available viscosities of resin, the mine still did not achieve their designed 90% encapsulation which meant an extra two bolts per metre were being installed. Over-coring to determine where the resin was going revealed that near vertical coal cleat was filling with resin but it also revealed that bolts were gloved for over 50% of their length. Extensive in situ testing achieved some interesting new data applicable to that mine site: 1) short encapsulation pull testing of gloved sections of bolt gave similar bond strength to the non-gloved bolts; 2) high viscosity (thick) resin gave improved encapsulation and 3) gloving could only be significantly reduced by a modified bolt end that nearly contacted the side of the bolt hole. Testing methods and results achieved along with comparing them to methods and results from previous literature on roof bolt gloving investigations are reported.

INTRODUCTION

Over the last three years a new longwall mine has been under development in a coal basin that has not been mined by underground methods for over a decade. Routine installation audits of primary roof support within the initial coal seam drivage highlighted significant resin loss (>25%) as a common problem in the thick coal roof. The support rules at the mine stated that the four bolt pattern had to be increased to a six bolt pattern if full encapsulation was not achieved giving the mine a strong incentive to solve the resin loss problem.

Investigations were conducted by Jennmar into the resin loss by over-coring and it was discovered that all bolts were gloved to a significant extent. A project commenced into determining types of improved resin bolting parameters to maximise encapsulation and to reduce gloving.

PREVIOUS RESEARCH INTO RESIN BOLT INSTALLATION AND GLOVING

The following previous research concerning resin loss and gloving were used in developing the methodology for these investigations.

Pettibone (1987)

Resin from three different US manufacturers were installed with the same rock bolt type into concrete blocks. Gloving: “Brand B exhibited glove-fingering in 1 out of 22 tests; Brand C had glove-fingering in 4 out of 24 tests. These incidences of glove-fingering are considered to be minor. On the other hand Brand A had glove-fingering in 22 out of 25 tests”. Hydraulic fracture: “Brand A, blocks split in 17 out of 25 tests. About one-third of the blocks split with brand B (7 out of 22). Brand C had no block cracking problems when manufacturer-recommended procedures were followed”. The work by Pettibone indicated that the resin properties alone can dramatically alter the extent of gloving and the insertion pressure causing hydraulic fracture of the surrounding rock.

Campbell, Mould and MacGregor (2004)

Extensive testing focussed on reducing the extent of gloving and determining the reduction of load transfer of Australian type rock bolts and resins used in New Zealand mines. Testing of modified bolts gave reductions in gloving with chamfered, wiggled and off-set nut giving the best results.

A strain gauge instrumented bolt installed underground showed near to nil load transfer in the top 400 mm length of the bolt. Laboratory 180 mm long encapsulation pull tests conducted at the University of New South Wales indicated gloving did not significantly reduce the load transfer.
Pastars and MacGregor (2005)

More extensive pull testing of simulated gloving was conducted by Strata Control Technology (SCT) operations based on the earlier New Zealand experiences. Laboratory pull tests were completed in concrete cylinders using mix and pour PB1 resin as the correctly mixed non-gloved case. Gloving was simulated by using an empty plastic film made to 27 mm diameter inserted into the drill hole and filled with mix and pour resin before adding the bolt. The result was the simulated gloved bolts only providing 10% of the load transfer of the non-gloved test cases. In situ pull testing was conducted in coal using the same simulation of gloving with mix and pour resin and an empty plastic capsule. The results were similar to the laboratory test results with simulated gloved bolts only providing 10% of the load transfer of the non-gloved cases.

Compton and Oyler (2005)

US resins and bolts were tested at the NIOSH Safety Research Coal Mine (SRCM). Extensive pull testing was completed using the standard 300 mm Short Encapsulation Pull Test Method (SEPT) versus a new technique of over coring a fully encapsulated bolt to leave 300 mm bonded for pull testing. The overcore method achieved results 35% higher than the SEPT method which was explained by the SEPT having resin loss. The SEPT method used in the US calculates the capsule length required to obtain 300 mm encapsulation rather than reaming the hole and adding excess resin into the top target bond length section as commonly used in Australia. All bolts and US resins over-cored showed some extent of gloving, including off set head bolts. Results from a comparison of six lightly gloved bolts versus four severely gloved bolts pull tested indicated there was no reduction in bond strength from gloving. Installation pressures within the borehole were measured with a result around 34.5 MPa (5 000 psi). Resin loss measured within the SRCM mine roof averaged 44%. It should be noted that typical US made resins are high viscosity in comparison to Australian industry standard resins of 2011.

STUDY OF MINE SITE BOLTING PARAMETERS

Lithology

The typical lithology at the mine site is a thick uniform strong coal roof extending up to 5 m above the normal roof line. Above the coal roof is a thick (>20 m) conglomerate which is sometimes encountered during high drivage for overcasts and belt drive head installations. The test work discussed within the paper was all conducted within the uniform coal roof making up the 1.8 m primary bolting horizon

Bolting machines

Primary support was installed off continuous miner mounted bolting rigs, typically within 2 to 3 m from the face. The bolting rig was hydraulically powered with typical capacity of 1 t thrust and 320 Nm torque.

Primary roof support components

The primary roof bolt used was a 1.8 m long JX profile M24 bolt with typical core diameter of 21.7 mm and rib height of 1.5 mm.

The standard drill bit diameter used was a 27 mm with both spade and twin-wing (angel) profile used. Resin capsules varied due to efforts in solving resin loss. All capsules used were in accordance with the Australian industry standard of 23.6 to 24 mm in diameter.

INVESTIGATIONS INTO RESIN LOSS

Initial underground testing involved encapsulation measurements on the standard bolting variables such as drill bit diameters and resin capsule lengths, along with bolt over-cores to determine where the resin was being lost. Hole diameters were measured along the length of the drill holes using a borehole micrometer and average diameter calculated. Theoretical encapsulation was calculated assuming no resin loss into the strata. Bolts were installed into the various drill holes using either 1200 mm or 1000 mm long resin capsules and actual encapsulation was measured.
Within the Main Headings c/t 11 test area it was determined that 27 to 28% resin loss was being experienced with 26.5 and 27 mm diameter drill bits. A limited number of tests indicated that using a longer resin capsule could achieve full encapsulation and that using a 28 mm drill bit may also reduce resin loss (Table 1).

<table>
<thead>
<tr>
<th>Resin Length</th>
<th>27 mm Spade</th>
<th>27 mm Angel</th>
<th>26.5 mm twin-wing</th>
<th>28 mm twin-wing</th>
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</thead>
<tbody>
<tr>
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<td>2017</td>
<td>1918</td>
<td>2205</td>
<td>1509</td>
</tr>
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<td>Actual</td>
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<td>1370</td>
<td>1530</td>
</tr>
<tr>
<td>% resin loss</td>
<td>27%</td>
<td>26%</td>
<td>28%</td>
<td>NIL</td>
</tr>
<tr>
<td>Theoretical</td>
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<td>2238</td>
<td>2572</td>
<td>1761</td>
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<tr>
<td>Actual</td>
<td>1700 + excess</td>
<td>1700 + excess</td>
<td>1700 + excess</td>
<td>1700 + excess</td>
</tr>
</tbody>
</table>

Table 1 – Theoretical V’s actual encapsulation

The mine site bolt, drill bit and resin were used for an observed correct installation followed by over-coring to view the sites of resin loss. The bolt was a 1.8 m long M24 JX profile, the capsule was a 1200 mm long, 2:1 mastic: catalyst ratio water based resin, and the drill bit was a 27 mm spade. The installation was 7 s spin from the base of the capsule to the back of the hole, followed by a further 3 s spin at the back of the hole. As shown in Figure 1, the over-cored bolt revealed resin loss into vertical fractures along the borehole wall. Measurements and observations taken on this first bolt showed: 1) gloving with complete intact capsule film was present along the top 650 mm; 2) encapsulation length was 1100 mm which represented 45% resin loss; and 3) no resin was seen in the coal extending above and beyond the drill hole.

Un-mixed yellow fast set mastic was seen migrating from near vertical fractures in the top one third of the empty hole left after the core was removed. A physical sample of this was recovered and confirmed to be resin mastic.

Figure 1 - 1st site over-core: standard bolt, standard viscosity 2:1 water based catalyst resin

A second over-core of a bolt installed some weeks previously as a normal support bolt, being 1.8 m JX with the same standard viscosity 1200 mm 2:1 mastic: water based catalyst was also extensively gloved (Figure 2).

Figure 2 - Overcored bolt taken from the support pattern

Over-coring was completed using an NQ sized rod, as the 75 mm O.D. of the NQ bit was able to fit through the gripper jaws on standard bolting rigs. When over-coring pre-installed support pattern bolts, difficulty was encountered in aligning the NQ sized rod to travel along the 1.8 m roof bolt without hitting the bolt. The over-core method was changed to install a new bolt with the hydraulic rig fixed in position and over-core the bolt immediately. Over-coring to determine resin loss was also targeted within the working face area to avoid any issues with deteriorated roof strata. Over-coring a bolt can be very slow and take up to one hour per bolt. To enable over-coring to be performed off the continuous miner at the
development face, an optimum custom made coring bit was found through trial and error with the time improving to below 10 minutes per bolt.

The outcome of the first set of testing was that the mine site increased capsule length to 1400 mm to improve encapsulation.

ADDRESSING GLOVING

Resin capsule film configuration

From visual examination of the gloved bolts, it could be seen that the capsule longitudinal plastic weld had ruptured under pressure allowing the film to open up and lay against the borehole wall. It was decided to perform over-cores on bolts installed using a different type of capsule film configuration. The two types of capsule configurations are shown in Figure 3.

Water based catalyst 1/3 of volume

Oil based catalyst 7 % of volume

Mastic 2/3 of volume

Mastic 93% of volume

Figure 3 - Cross-sections of the two main types of resins used in the study

Over-coring of three bolts installed with standard viscosity 1400 mm long J-Lok 93:7 mastic: catalyst oil based resin was completed. The over-cores again showed resin loss into vertical fractures within the coal and again showed extensive gloving extending 300 mm to 800 mm in length from the top of all three bolts as shown in Figure 4. The two capsule film configurations did not appear to vary the degree of gloving and both appeared to have opened up along the plastic weld and lay against the borehole wall in a similar manner.

Figure 4 - Three over-cored bolts, 93:7 ratio mastic: catalyst oil based, standard viscosity

Modified bolt end

A modified bolt end was manufactured similar to those shown in Figure 5 with the intention of the point contact and the flat edges shredding the plastic film. A 1400 mm long high viscosity (thicker) J-Lok
93:7 oil based resin was also used in view of creating more turbulence for film shredding. The high viscosity resin was tested alongside the standard viscosity 1400 mm long 2:1 ratio water based resin. Eight “pinched ear” bolts were installed and encapsulation measured with two bolts removed by over-core.

The five bolts installed with the high viscosity J-Lok all had excess resin expel from the holes during installation, while the three installed with standard viscosity 2:1 water based resins measured 300, 380 and 900 mm free length from the collar. Surprisingly the thicker high viscosity resulted in less resin loss.

![Figure 5 - Pinched ear bolt modifications](image)

Both 26 mm wide pinched ear bolts over-cored were less gloved than previous standard bolts with the plastic film broken into segments. The higher viscosity resin combined with the pinched ears bolt was even less gloved (Figure 6).

![Figure 6 - Overcores using a single 26 mm wide “pinched ear” bolt](image)

After considering the gloving improvement and ease of installing the 26 mm pinched ear bolts, various widths and number of pinched ears were then tested. It was determined that a single 28 mm wide pinched ear could be installed into a hole drilled with a 27 mm drill bit. It should be noted that previous measurements of hole diameter in coal with a 27 mm bit were 28.1 to 28.5 mm in diameter.

Over-cores of larger 28 mm wide pinched ears and standard bolts were conducted with two different types of know resin viscosities (Figure 7).

Further bolt installations were performed for measuring encapsulation in the same area. Single pinched ear bolts, 1 400 mm long capsules, with standard viscosity 2:1 ratio water based resin loss between 37% and 40%. A standard 1.8 m bolt gave 47% resin loss with the same resin. However, six standard 1.8 m bolts installed with the J-Lok high viscosity (thicker) resin of various capsule lengths gave nil resin loss.

A two hundred bolt trial was attempted of the single “pinched ear” bolts with a high viscosity water based resin. It proved difficult to consistently get the bolts to the back of the hole with the 1 t standard thrust of the bolting rigs with many nut shear-pins breaking out prematurely.

After the failed bulk installation trial, further testing was conducted over four separate days to determine the importance of continuing the project towards eliminating gloving and to further investigate encapsulation gains with high viscosity resin.
Load transfer testing of gloved bolts

A limited number of over-core pull tests were able to be performed using the test methodology described by Compton and Oyler (2005). The method requires a bolting rig to be set in position for up to three hours without moving. A continuous miner with bolting rigs was used at the face during a maintenance shift and one test was done on each side of the miner without moving. The testing method as illustrated in Figure 8 shows the bolt being removed after pull testing to determine the extent of gloving.

The results of the gloved bolt pull tests were compared to some simple Short Encapsulation Pull Tests (SEPT) completed without any reaming but using a short capsule targeting 300 mm encapsulation. The SEPT bolts were not over-cored, due to time constraints, but were assumed to be not gloved. This assumption was based upon the section of all other bolts viewed not being gloved within the bottom 500 mm of initial mixing through the resin capsule and the capsules used for SEPT were less than 200 mm long.

The results for the SEPT testing were 12 to 15 t and assumed to be the no gloving scenario. The over-core pull tests of gloved bolts achieved 12 to 13 t. The testing was quite limited but the results indicate no major difference in bond strength between the two sets of tests.

Removal of the first pull tested bolt by over-core showed damaged to the resin by the core bit hitting the side of the bolt within the top section (Figure 9a) but extensive gloving was present. It was decided to remove the other bolt by continuing to pull it out using the pull test ram. The second bolt was also gloved within the top half of bolt and interestingly indicated failure mode of the resin bond (Figure 9b).
The resin appeared to have been crushed under mostly compression for the top 200 mm with the bottom 100 mm being mostly shear failure along the contact with the borehole wall. This is only an indication as some damage would be expected during the bolt pull out even at less than the 12 t loads in the coal roof.

**Figure 8 - Overcore pull test method**

1. Normal installation 1.8 m bolt (1.7 m hole), Full length two speed resin
2. Overcore leaving 300 mm bonded length
3. Pull test on the 300 mm bond - 1 hour cure
4. Overcore entire bolt

**Figure 9 - Bolts removed after pull tested using the overcore method**

**ADDRESSING ENCAPSULATION**

Considerable effort had been expended into investigating and attempting to solve the problem of gloving. It was accepted by the mine management that the *in situ* pull test results combined with literature reviewed, indicated gloving is not detrimental to bolt/resin and rock bond strength as it was first considered. The mine management’s main concern was the issue of poor encapsulation; Resin loss was being measured at 40 to 45 % with the standard viscosity resin and the change to 1400 mm long capsules did not achieve full encapsulation. Various trials conducted by the mine on lower viscosity resin and smaller rib deformations on roof bolts did not produce improvements to encapsulation. Back analysis of testing conducted with high viscosity J-Lok resin aimed at reducing gloving indicated that the thicker resin gave a much improved encapsulation (Figure 10).
A bulk installation trial was conducted four months later in the colder months of June using 1200 mm long high viscosity water based J-Lok resin in the mains headings. The thicker resin was found to be noticeably more difficult to force the bolts to the back of the hole without the shear-pin breaking in the nut, but generally they were successful. Encapsulation targets were met with over 85% of bolts having resin appear at the collar and the length not encapsulated on the other 15% of bolts ranging from 100 to 350 mm. A further installation trial was conducted in a gateroad development panel some two months later. The same high viscosity water based resin was successful but a coarser limestone high viscosity oil based resin was also trialled and proved successful with an improved ease of bolt insertion without shear-pins breaking out in the nuts.

A full implementation commenced through-out the mine site using the new 1200 mm long J-Lok coarse limestone high viscosity oil based resin. Encapsulation measurements were conducted within the same 100 m of roadway of the new resin versus the previous mine site standard viscosity 1400 mm long resins. The result was that the new J-Lok resin achieved an average improvement of 200 mm encapsulation with a 200 mm shorter capsule. To date, all measurements with the new resin have had the length not encapsulated measured to the collar being less than 300 mm.

CONCLUSIONS

The high standards set by a new mine site in regards to ground support and pursuing full encapsulation of rock bolts has led to a considerable investigation into resin bolting within the very thick relatively uniform coal roof.

The over-coring of rock bolts highlighted a problem with resin loss into fractured rock about the bore hole, but more concerning in the initial stages was the extensive gloving on the bolts. During testing of various modified bolt end profiles it was determined that “pinched ear” ends of 26 to 28 mm widths (patent pending) installed into a hole drilled with a 27 mm bit can significantly reduce gloving, but installation difficulties using standard Australian bolting rigs would need to be overcome. It was found during the same tests that higher viscosity resins definitely reduce resin loss within this mine site roof type in comparison to previous Australian industry standard resin’s viscosity.

Load transfer testing using the overcore pull test method was successfully completed within the mine roof at the development face. The results from the significantly gloved standard bolts was similar to the assumed “non-gloved” SEPT and importantly was typical for strong coal roof at 12 to 13 t per 300 mm encapsulated. The concern for bolt installation changed back from gloving to again focus on improving encapsulation.

A new J-Lok high viscosity coarse limestone oil based catalyst 93:7 resin has been successfully implemented at the mine site. The results to date have given an improvement of 200 mm to encapsulation length with a reduction of capsule length by 200 mm. The length of bolt not encapsulated from the collar is now consistently below 300 mm.
ACKNOWLEDGEMENTS

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REFERENCES


EXPERIMENTAL PROTOCOL FOR STRESS CORROSION CRACKING OF ROCKbolts

Damon Vandermaat¹, Elias Elias², Peter Craig¹, Serkan Saydam¹, Alan Crosky², Paul Hagan¹ and Bruce Hebblewhite¹

ABSTRACT: A new laboratory facility designed and constructed at the University of New South Wales, aims to continue and offer a new approach to researching the phenomenon of the stress corrosion cracking. This new approach includes the use of full sized specimens, a specially designed frame, as well as a new loading regime, known as the Periodically Increasing Stress Test, to closely simulate the loading encountered by bolts in service. Coupled with a detailed water testing program to be undertaken at a number of partner sites, this new approach hopes to further increase understanding of stress corrosion cracking and its causes.

INTRODUCTION

Stress corrosion cracking (SCC) is a failure mode, induced by the combination of an applied stress and an appropriately corrosive environment. SCC will result in the catastrophic, brittle failure of the material, as a result of crack growth, if remediation measures are not taken. SCC is a phenomenon that has caused significant problems for the underground coal industry for the last decade. After first being noted to occur at a BHP mine in the late 1990’s (Gray, 1998), SCC of rock bolts has been an active area of research in Australia. Crosky, et al. (2002) noted that SCC was an issue in at least three Australian coal mines, although this number has since grown. SCC tends to occur in areas that have clay bands in the bolt horizon, thick coal roofs, corrosive ground water and shearing between the bedding planes (Crosky, et al., 2002). They also note the importance of bolt metallurgy, particularly that of steel toughness, and the possible implication of microbiological action has been noted (Crosky, et al., 2002).

To date, no one has been able to effectively re-create underground service conditions to produce SCC failures in the laboratory. Gamboa and Atrens (2003, 2005) produced SCC failure modes in rockbolt steels in the laboratory and concluded that hydrogen embrittlement was the mechanism of failure. However, this work was not carried out in an environment representative of underground conditions. In further work, Villalba and Atrens (2007) concluded that SCC was not linked to any metallurgical factors, which is contrary to the findings of Crosky et al. (2002) and anecdotal field experience (Craig, et al., 2010).

An Australian Research Council (ARC) linkage project introduced by Craig et al., (2010), aims to build largely on the work carried out by these and other researchers. Overall, the project intends to further classify the cause of SCC of rock bolts by undertaking a forensic analysis of the mine environment and bolts that have failed in service. This analysis will be extended to incorporate the design of a purpose built laboratory facility for simulating the mine environment. This facility will be used to induce SCC failure in specimen bolts so that the failure mechanisms can be identified. The project has the following five industry partners that have a history and/or concerns about SCC:

- Springvale Colliery (Centennial Coal) in the Lithgow seam - Western Coalfields;
- Blakefield Colliery (Xstrata Coal) in the Bulga Complex - Hunter Valley;
- Narrabri Colliery (Whitehaven Coal) in Gunnedah Basin;
- Moranbah North Mine (Anglo Coal Australia) in Bowen Basin, and
- Jennmar Australia.

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TESTING PROCEDURES

Environments that cause SCC are usually aqueous and can be either condensed layers of moisture or bulk solutions. As stated in the available literature, SCC is alloy/environment specific, that is, it is frequently the result of a specific chemical species in the environment. However, previous studies conducted into the SCC of rock bolts in underground mines have failed to identify(characterise) the environment that is responsible for SCC. Consequently, this gives rise to one of the main aims of this project.

In order to identify the environment that gives rise to SCC, several miniature three-point coupon testing rigs, illustrated in Figure 1 will be placed in an underground coal mine. These test rigs will be hung from the roof of the mine, situated under a rock bolt with a consistent groundwater flow, and the water will be sampled and examined. The specimens that will be used in the test will be cut longitudinally across the top of the rock bolt of known chemical composition. This is so that each test includes the various factors for full-length rock bolts such as the stress concentration caused by the rig profile, mill scale and decarburized layer. Furthermore, the test specimens, which will be used, will have a thickness of 2 mm, which is excluding the rib thickness. This thickness was selected as fracture mechanics states that the thinner the test specimen the shorter the crack length needs to be before specimen failure occurs. In selecting such a small thickness not a lot of force will be needed to bend the specimen and the amount of deformation need to take the specimen to yield will be minimal.

Once the test specimens fail, the fracture surface will be examined using a Scanning Electron Microscope (SEM) in order to determine that the failure mechanism is that which occurs in service. Upon establishing this link, the environment will be sampled and examined so that it may be identified and compared to the result obtained prior to specimen failure. Consequently, it will be possible to see if there was any change in the water composition.

Water analysis will be carried out for each mine test site and steps will be taken to ensure that the results obtained from such tests are representative of the in situ conditions. It is important to note that the test method complies with both the “NSW Coal Regulations and Consideration to OHS” and the “Standard Methods for the Examination of Water, Waste Water and Ground Water”.

Figure 1 - Miniature three point bend test apparatus. Top shows a schematic of in situ test setup whereas, the bottom shows photo of the testing rig which has been constructed

Water sampling

The objective of sampling is to collect a portion of material small enough in volume to be transported conveniently and yet large enough for analytical purposes while still accurately representing the material being sampled. This objective implies that the relative proportions or concentrations of all pertinent components will be the same in the samples as in the material being sampled, and that the sample will be handled in such a way that no significant changes in composition occur before the tests are made.

The type of sampling that has been chosen to be used during this study is known as the Grab method. Grab samples are single samples collected at a specific spot at a site over a short period of time. The sample bottles, which are being used are predominantly glass, with Teflon® bottles made from
Polytetrafluoroethylene (PTFE) being used for samples collected for Inductively Coupled Plasma (ICP) analysis. This is because, this technique measures ionic and cationic species present in the sample and placing these samples in PTFE bottles prevents silica, sodium and boron being leached from the glass thus affecting the results.

Sample storage

Once samples have been collected, they will be packed in crushed or cubed ice before being transported for testing. This is to keep the samples as cool as possible, without freezing, to minimize the potential for volatilisation or biodegradation between sampling and analysis. However, to ensure that the measurements obtained from the analysis are as close as possible to in situ measurements, samples will be analysed as quickly as possible on arrival at the laboratory.

To determine the extent of change, if any, within a sample between the point of sampling and the point analysis, several tests will be conducted at various places. pH and dissolved oxygen will be tested at the point of sample collection with electronic probes. Once the samples taken back to the surface, they will be tested again using the same methods. The samples will then not be tested further until they arrive at the laboratory. If there is a significant degree of variation between each of the tests, then a new protocol will be implemented.

Water analysis

It must be noted that a proper characterisation of the environment is required as slight changes in the temperature, degree of aeration and/or concentration of ionic species can render an environment that is harmless into one that can cause SCC. Consequently, this allows evaluation of the environments that both cause SCC and don’t cause SCC and assessment of the differences between the two. In doing this, a proper understanding of the environment that causes SCC will be established and appropriate remedial action will be implemented to prevent SCC of rock bolts. Table 1 outlines the different tests that will be used in this research to characterise the underground mine water along with the preservation techniques and container type being used.

The ICP analysis, outlined in Table 2 and Table 3 show different cations and anions which will be measured during this test, respectively.

Table 1 - Water testing regime

<table>
<thead>
<tr>
<th>Determination</th>
<th>Container</th>
<th>Sample Type</th>
<th>Preservations</th>
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<td>Grab</td>
<td>Refrigerate</td>
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<td>Glass</td>
<td>Grab</td>
<td>Refrigerate and ensure there is no head space when the bottle is full</td>
</tr>
<tr>
<td>Biochemical Oxygen Demand (BOD)</td>
<td>Glass</td>
<td>Grab</td>
<td>Refrigerate</td>
</tr>
<tr>
<td>Chemical Oxygen Demand (COD)</td>
<td>Glass</td>
<td>Grab</td>
<td>Analyze as soon as possible or add H₂SO₄ to pH &lt;2; refrigerate</td>
</tr>
<tr>
<td>ICP Anions</td>
<td>PTFE</td>
<td>Grab</td>
<td>Add H₂SO₄ to pH &lt;2; refrigerate</td>
</tr>
<tr>
<td>Cations</td>
<td>PTFE</td>
<td>Grab</td>
<td>Refrigerate, keep in dark</td>
</tr>
</tbody>
</table>
Table 2 - Cations of Interest

<table>
<thead>
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<tr>
<td>Sodium (Na)</td>
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<tr>
<td>Magnesium (Mg)</td>
<td>Calcium (Ca)</td>
</tr>
<tr>
<td>Iron (Fe)</td>
<td>Copper (Cu)</td>
</tr>
<tr>
<td>Zinc (Zn)</td>
<td></td>
</tr>
</tbody>
</table>

Table 3 - Anions of Interest

<table>
<thead>
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<th>Anions</th>
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</thead>
<tbody>
<tr>
<td>Bromide (Br)</td>
<td>Chloride (Cl-)</td>
</tr>
<tr>
<td>Fluorine (F-)</td>
<td>Nitrite (NO₂⁻)</td>
</tr>
<tr>
<td>Nitrate (NO₃⁻)</td>
<td>Sulfate (SO₄²⁻)</td>
</tr>
<tr>
<td>Phosphate (PO₄³⁻)</td>
<td></td>
</tr>
</tbody>
</table>

Preliminary flow rate measurement of the water trickling off the bolts at Springvale Colliery has already been taken to calibrate the flow rates used in the experiments. Flow rate measurements are important as they provide information on the quantity of water passing over the bolts. Three bolts in the F-Heading at the mine were selected, each representing low, medium and heavy flow. The flow rates were used as design values for the reticulation system and these results can be seen in Table 4.

Table 4 - Flow rate measurements taken at Springvale Colliery

<table>
<thead>
<tr>
<th>Description</th>
<th>Flow Rate (mL/min)</th>
<th>Flow Rate (L/Week)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Light Dripping</td>
<td>20</td>
<td>201.6</td>
</tr>
<tr>
<td>Heavy Dripping</td>
<td>60</td>
<td>604.8</td>
</tr>
<tr>
<td>Constant Trickle</td>
<td>100</td>
<td>1008</td>
</tr>
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LABORATORY DESIGN

The core aim behind the design of the laboratory facility was to perform testing on full sized specimens. The work done by Gamboa and Atrens (2003) and Villalbao and Atrens (2007) focused on using small, representative samples for their SCC experiments. While useful in gaining an understanding, an upgrade to full sized specimen testing was needed to examine bolt surface characteristics such as de-carburisation and stress concentrations caused by ribs and surface irregularities.

It was identified from a review of American Society for Testing and Materials (ASTM) testing methods for examining SCC that two relevant tests existed for simulating the in situ stresses placed on a bolt. These were the bent beam test (ASTM G39) and the tensile test (ASTM G49).

The bend test applies a lateral side load to a specimen which generates high tensile forces in the outer radius of the specimen. A number of bend test configurations exists, however the four-point method was chosen because it generates a long section of bolt with a maximum stress (ASTM G39), and allows for easy coupling with a chemical cell (described later).

A static load, two point bent beam test was performed by Satola and Aromaa (2003) on two different types of rock bolt steel - Ø6 mm steel rebar and a Ø5 mm stand of king wire from a 15.2 mm cable bolt. Satola and Aromaa (2003) wanted to assess the corrosion difference between galvinised and un-galvinised steel, SCC was not their primary focus. They stressed the specimens to 85% of the yield strength and left them to sit in a stagnant bath of varying solutions. Their experiments were unable to yield a SCC failure, however transverse cracking was found in some samples tested in low pH water with high Cl⁻ ion concentrations. No follow up examination of cracks were performed.

The Linearly Increasing Stress Test (LIST) carried out by Gamboa and Atrens (2003, 2005) is a mixture of a direction tension tests and a modified version of the Slow Strain Rate (SSR) testing, outlined in ASTM G49 and ASTM G129 respectively. The LIST experiment places a specimen in a rig that applies a constant loading rate of 0.019 MPas⁻¹ until the sample fails, either by SCC or under normal ductile overload. Gamboa and Atrens (2003) tested small, representative samples machined from rock bolt steel and had success in generating stress corrosion cracking and identified hydrogen embrittlement as the SCC mechanism. Close inspection of the fracture surfaces with a SEM indicated that the failure
generated in the laboratory had a similar fractography to the failures experienced in service (Gamboa and Atrens, 2005). However the testing medium, a sulphate solution with a pH close to 2.1, could not be considered representative of normal groundwater conditions in Australia’s major coal basins.

The laboratory facility can be divided into three key facets: the load frame, the chemical cell and environmental room. The load frame and the chemical cell will all be housed in an environmental room in which the temperature and humidity can be controlled. The room is designed to mimic the atmospheric conditions of an underground coal mine. Temperature will be controlled with the use of an air conditioning unit and will maintain air temperature between 16°C and 24°C. The temperature can be set and held with an accuracy of ±1°C. Humidity will be controlled with a steam humidifier mounted in the wall to keep a constant humidity within the chamber.

The load frame

The load frame design is based on several of the ASTM standards for conducting SCC testing. The frame will allow for both bend and tensile SCC testing to be carried out without altering the frame between tensile and bending tests. The frame is designed to load a single bolt at a time. As such, an array of frames will be used to test a number of bolts simultaneously.

Stress is applied to the bolt in the bend test by means of a torque multiplier and a screw. As the screw is turned, it moves a loading jig, which imparts a lateral load to the bolt. This lateral loading can be seen in Figure 2. The ends of the bolt are held in a fixed position in the frame. The displacement of the jig is measured to determine the amount of bend in the bolt, which allows the stress in the outer fibers of the bolt to be inferred analytically by equation 1, found in ASTM G39. However, this equation only applies within the plastic limit. The four point system also allows for a large area of the bolt to be held at a constant stress, allowing for accurate stress measurements in the bolt to be made. These results will be checked by attaching a strain gauge to the bolt and through numerical modelling.

\[
\sigma = \frac{12fy}{(3R^2 - 4A^2)}
\]  

(1)

Where, E is Young’s modulus, t is the steel coupon thickness, y is the amount of deflection, H is the distance between the outer supports and A is the distance between the outer support and the inner support.

The tensile tests are performed with the use of a hydraulic nut. This hydraulic nut is attached to one end of the bolt, while the other end is restrained with the use of a standard nut. The bolts used in the tensile testing are custom made to have threads at each end, which allows for this simple load mechanism. A load cell is incorporated in the system to measure the load on the bolt.

![Figure 2 - Schematic of bend and tensile testing in the frame](image_url)

It was identified from the work done by Gamboa and Atrens (2003, 2005) that a Linearly Increasing Stress Test (LIST) is an acceptable means of accelerating SCC growth. In the LIST carried out by Gamboa and Atrens (2003, 2005), stress in the sample is increased at the very slow rate of 0.019 MPa/s. However, due to the technical constraints associated with scaling this loading method up to a full sized specimen, a modified version of this test has been devised. Figure 3 indicates the design of the new load frame.
The modified loading method will be a Periodically Increasing Stress Test (PIST). This PIST test will work similarly to the LIST, however instead of the test increasing at a constant rate, the stress in the sample will be increased at regular intervals for the duration of the test. These intervals will probably be daily, and the increase in stress to be used will be determined through calibration testing, as outlined in ASTM G129. This loading mechanism is expected to more closely emulate the periodic loading regime experienced by bolts installed in underground coal mines.

The calibration testing described in ASTM G129 is for SSR testing; however it is believed the same principles can be applied for this testing protocol. ASTM G129 requires that the specimens be loaded (in tension) at a chosen rate in an inert environment, and measuring the time taken till failure. The test must then be repeated in the corrosive environment at the same loading rate. These tests must be repeated for a number of loading rates and results plotted on a time vs. loading rate graph to find the optimal loading rate (ASTM G129). The frame has also been designed so that it is easily adaptable to allow for SCC testing of cable bolts in the tensile configuration only.

The chemical cell

A chemical cell is used to contain the corrosive fluid to which the bolt is exposed. To closely replicate service conditions, the bolt is exposed to constantly flowing water. The cell itself is a length of clear vinyl tubing with a diameter large enough to fit a bolt inside and allow for fluid to flow freely around it. This design can be seen in Figure 4. Vinyl tubing was chosen because of its flexibility which will allow the cell to twist and flex while a load is applied to the bolt. The four-point bend test was chosen partly because it allows an easy arrangement to attach the chemical cell. Using a three point system would have posed problems with potentially rupturing the chemical cell as the load was applied. Figure 4 shows the positioning of the chemical cell in the bend test configuration. The same chemical cell will be used for the both the tensile and bend tests, however the cell use for tensile testing will encase more of the bolt.

The use of stagnant water in other test regimes (Satola and Aromaa, 2003; Spearing, et al., 2010) has shown that the water chemistry can change dramatically during testing. A model similar to that used by Villaescusa, Hassell and Thompson (2008), where the water is kept in a continuous cycle will be used for these experiments.
The water used in the experiment is sourced from the mine sites supporting the project. It is known that stress corrosion cracking is occurring in underground mines (Crosky, et al., 2002, 2004). Using water sourced from site gives the greatest chance of capturing the environment that produces SCC increasing the likelihood of generating a SCC failure. Once the load frame has been validated, experiments will be carried out to isolate the water constituents causing SCC.

The water for the experiment is stored in large tanks within the environment chamber and circulated past the bolts, before being collected in a large sump tank. The water quality in the sump is tested frequently to ensure that the water chemistry has not deviated from the planned testing conditions. If the water passes, it is reticulated to the original holding tanks for reuse.

CONCLUSIONS

SCC continues to be an issue in the underground mining industry. Continued research and experimentation is aimed at answering some questions around SCC and deliver solutions for the industry to use in combating SCC.

The laboratory facility designed at the University of New South Wales offers a new approach to SCC experimentation and research. The new PIST loading strategy is designed to closely emulate the loading experienced by bolts during underground use. The up-scaling to full sized specimens has been done to accommodate the influence of decarburisation and stress concentrations from ribs. By using water collected from mine sites as the corrosive medium in the corrosion chamber, there is a greater chance of capturing the environmental factors leading to SCC.

An extensive and detailed water sampling regime will identify all the potential factors that are causing SCC. The water survey is coupled with XRD analysis to determine the mineralogy of any clay bands that are suspected of playing a role in SCC. The knowledge gained from this environmental analysis is pivotal in identifying the SCC mechanism at play and devising solutions to remedy or prevent the problem.

ACKNOWLEDGEMENTS

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FAILURE MODES OF ROCKBOLTING

Chen Cao, Jan Nemcik, Naj Aziz and Ting Ren

ABSTRACT: Rock bolting has advanced rapidly during the past four decades due to a better understanding of load transfer mechanisms and advances made in the bolt system technology. Bolts are used as permanent and temporary support systems in tunnelling and mining operations. A review of reinforcement devices has indicated that three classes have evolved as part of rockbolt and ground anchor while the rock is not generally thought of as being a component of the reinforcement system. A classification of rockbolting reinforcement systems is presented, followed by the fundamental theory of the load transfer mechanism. Finally, various failure modes of rockbolting systems are discussed.

CLASSIFICATION OF REINFORCEMENT

Rock bolting has advanced rapidly during the past four decades due to a better understanding of load transfer mechanisms and advances made in the bolt system technology. Bolts are used as permanent and temporary support systems in tunnelling and mining operations. In surface mining they are used to stabilise slopes and in underground workings they are used for roadway development, shaft sinking, and stoping operations. Rock bolts are installed around openings in mines and tunnels to tie weaker layers to stronger layers above, to prevent sagging and separation and to provide a reinforcement zone in rock mass that makes greater use of a rock's inherent mass strength to enable it to be self-supportive.

A reinforcement scheme is an arrangement of primary, secondary and tertiary reinforcement systems in a variety of dimensional and spatial configurations. Some of these may have been installed as pre- or post-reinforcement, and may be un-tensioned, pre-tensioned or post-tensioned. A review of reinforcement devices has indicated that three classes of device have evolved: rockbolt (generally less than 3 m), cable bolt (generally in the range from 3 to 15 m) and ground anchor (generally longer than 10 m). All of them comprise four principal components as shown in Figure 1, Windsor (1997):

![Four principal components of a reinforcement system, after Windsor (1997)](image)

 Whilst, the rock is not generally thought of as being a component of the reinforcement system, it has a marked influence on the behaviour of the system and must therefore eventually be considered an integral part of the system. For reinforcement with a bolt, the reinforcing element refers to the bolt and the external fixture refers to the face plate and nut. The internal fixture is either a medium, such as cement mortar or resin for grouted bolts, or a mechanical action like friction at the bolt interface for frictionally coupled bolts. The internal fixture provides a coupling condition at the interface.

With reference to the component of internal fixture, the reinforcement system has been catalogued into three fundamental types, Windsor and Thompson (1993):

- Continuous Mechanically Coupled (CMC) systems;
- Continuous Frictionally Coupled (CFC) systems; and
- Discreetly Mechanical or Frictionally Coupled (DMFC) system.

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According to this classification system, cement and resin grouted bolts belong to the CMC system while Split set and Swellex bolts belong to the CFC system. The third group can be anchored by a slit and wedge mechanism or an expansion shell.

Nowadays, grouted rock anchors have been used extensively in a wide range of geotechnical and mining applications as temporary or permanent ground supports. A grouted anchor is defined as a structural support comprising a tendon which is inserted into a drilled hole and then grouted. Grouted steel anchors can be made from solid bar or stranded wire cables. Solid bars can be either smooth or deformed with the latter being further classified as either "threadbar" or "rebar".

LOAD TRANSFER MECHANISM

The load transfer concept is central to the understanding of reinforcement system behaviour, and the mechanical action of the different devices and their effects on excavation stability. This concept can be visualised as being composed of three basic mechanisms, Stille et al (1989):

- Rock movement and load transfer from an unstable zone to the reinforcing element.
- Transfer of load from the unstable region to a stable interior region via element.
- Transfer of the reinforcing element load to the stable rock mass.

A fully grouted bolt is a passive roof support system, which is activated by movement of the surrounding rock. The relationships between them belong to the continuous mechanically coupled bolt system (CMC). The efficiency of load transfer is affected by the type and properties of the grout, profile of the rock bolt, hole and bolt diameter, anchorage length, rock material, confinement pressure, over and under spinning, and installation procedures. It is commonly accepted that the fully grouted bolt provides greater shear surface for transmitting the load from rock to bolt and vice versa. The grout supplies a mechanism for transferring the load between the rock and reinforcing element. This redistribution of forces along the bolt is the result of movement in the rock mass, which transfers the load to the bolt via shear resistance in the grout. This resistance could be the result of adhesion and /or mechanical interlocking. Adhesion is the actual bonding between grout, steel, and rock, and the mechanical interlocking is a keying effect created when grout fills irregularities between the bolt and the rock.

Stress concentration is induced between the roughness of hole wall and the surface profile of the bolt. This localised stress concentration could exceed the strength of the grout and rock resulting in localised crushing that allows additional deformation in the steel.

Singer (1990) demonstrated that there is no adhesion between the grout to bolt and grout to rock interface. Aziz and Webb (2003) reported almost no adhesion between the bolt surface and grout only, Yazici and Kaiser (1992) stressed that the adhesive component was neglected because it cannot be mobilised with frictional strength during the pullout test.

In general, only resinous grouts can meet the high strength required for short anchorages. A grouted bolt can transfer greater loads than expanded shell or wedge type anchorages. This may be essential in weaker rock strata where transfer of high loads over a short length borehole may initiate failure at the rock interface.

FAILURE MODE OF TWO PHASE MATERIAL SYSTEM

By pulling out a steel bar embedded in a concrete column (note, there is no grout material), engineers have been aware that the bond forces radiate out into the surrounding media from the bonding surface of an anchored steel bar. Studies of bonding forces for plain reinforcing bars and rebar show that bond for plain bars is made up of three components, Lutz (1970):

- Chemical adhesion;
- Friction; and
- Mechanical interaction between concrete and steel.
Bond strength mainly depends on chemical adhesion and after slip, on friction. There is also some interlocking due to the roughness of the bar surface. The effect of chemical adhesion is small and friction does not occur until there is slip between bar and concrete.

For rebar, slip can occur in two ways; 1) The ribs can split the concrete by wedging action, and 2) The ribs can crush the concrete. When concrete is crushed to a “compacted powder” it becomes lodged in front of the ribs. In addition, even when slip and separation occur, additional transverse cracks and splitting cracks are very probable. Thus, large axial displacement cannot occur without transverse and longitudinal cracking in the surrounding concrete.

Lutz (1970) also outlined several types of cracks in a concrete cylinder with an axially embedded steel bar to identify the failure modes of reinforcement. The breaking of a concrete beam into small columns is called primary cracking which is the major failure mode. In addition, bond slip and dense minor radial cracks are also presented, as shown in Figure 2.

![Figure 2 - deformed of a concrete cylinder with pulled axially embedded plain reinforcing bar according to Lutz (1970).](image)

Tepfers (1973; 1979) established an analytical model for the tensile stress distribution causing development of the radial splitting cracks. When pulling out, the interface will result in significant stress concentrations. Due to these accumulated stresses, the debonding process will start and extend inside the specimen along the reinforcing bar. There are two types of cracks: cone-shaped cracks and longitudinal splitting cracks, both of which start at the interface as shown in Figure 3. The crack patterns depend on the interface geometry and the properties of the interface and the surrounding concrete; furthermore, these different crack patterns do not form independently from one another, but interact through complicated non-linear mechanisms.

![Figure 3 - Internal cone-shaped cracks and longitudinal splitting cracks, Tepfers (1979)](image)

It is thought that tensile stress is the cause of the splitting cracks. Tepfers (1973; 1979) assumed that the radial components of the bond forces can be regarded as a hydraulic pressure, acting on a thick-walled concrete ring surrounding the reinforcing bar. The shear stress at the interface distributes into the surrounding material by compression under a certain angle and is balanced by tensile stress rings in the concrete, as shown in Figure 4. According to this, the radial stress due to bond action on the concrete, which is also regarded as the hydraulic pressure against a thick-walled concrete cylinder, can be calculated out via shear stress of the interface $\sigma_y = \tau \tan \alpha$.

![Figure 4 - Left: the radial components of the bond forces are balanced against tensile stress rings. Right: bond stresses in the concrete adjacent to the bar. After Tepfers (1979)](image)
For determination of the resistance against radial cracking, three different stress distributions are applied referred to as uncracked elastic, partly cracked, and uncracked plastic stage. The cross section of the deformed concrete beam and terminology in his mechanical model are shown in Figure 5.

Figure 5 - Left: Part of the tensile zone in a reinforced concrete beam. Right: the variation in tensile stress in the concrete cover transverse to the reinforcing bar 1) Elastic deformation; 2) Partial crack; 3) Completely plastic. After Tepfers (1979)

In the elastic stage, using thick walled cylinder theory, the tangential stress can be found (Figure 5-1):

\[
\sigma_t = \frac{(\phi/2)^2 \sigma_y}{(c_y + \phi/2)^2} \left[ 1 + \frac{(c_y + \phi/2)^2}{r^2} \right]
\]

(1)

When the bond reaches the plastic stage, the cylinder will not break until the stresses in the tangential direction at every part of the cylinder have reached the ultimate tensile strength. The tangential stress in the cylinder can be expressed as \( \sigma_t = \phi \cdot \sigma_y / 2c_y \) (Figure 5-3). In the intermediate stage of above two cases, the ring has internal partial cracks where the circumferential stresses have reached the ultimate tensile strength of concrete. The bond force is now transferred through the concrete teeth between the internal cracks to the uncracked part of the ring. He found that when the radius of cracked zone \( r_c = 0.486(c_y + \phi/2) \) the bond force capacity of the concrete ring reaches its maximum value (Figure 5-2).

This analysis is based on the specific bond failure mode, i.e. cone cracks and radial cracks. In the rockbolting system, however, it is not always the case due to grouting material which dominates the failure of the bond. The thick wall theory and associated methodology of elastic and plastic analysis used in his paper are admirable and employed in later research work on the fully grouted rockbolt composed of three phase material with two interfaces.

**FAILURE MODES OF CABLE BOLTING SYSTEM**

Yazici and Kaiser (1992) developed a conceptual model for fully grouted cable bolts (Figure 6), called “Bond Strength Model (BSM)”.

Figure 6 - Schematic diagram reflecting the geometry of a rough cable bolt
According to their theory the bond strength is mainly frictional and hence depends on the pressure build-up at the interface which in turn depends on the dilational movement against the confining grout or rock. The cable bolt surface was simplified to be zigzag (twisting of the cable is ignored), thus a bilinear dilation-dependent joint strength concept introduced by Patton can be applied:

\[ \tau = \sigma \tan (i_0 + \phi) \]

For small angles, the bond strength can be expressed in terms of friction and dilation angle:

\[ \tau = \sigma \tan \left( i_0 + \left( \frac{\sigma}{\sigma_c} \right)^{\beta} + \phi \right) \]  

(2)

where:
- \( \tau \) = shear or bond stress;
- \( \sigma \) = radial stress at the bolt – grout interface;
- \( i_0 \) = dilation angle at the bolt – grout interface given by surface geometry;
- \( \beta \) = reduction coefficient of dilation angle, \( \sigma_c \) = compressive strength of grout;
- \( \phi \) = friction angle between the steel and grout.

The BSM involves four main components; axial displacement, lateral displacement, confining pressure and bond strength. In Figure 6, the schematic diagram illustrates these interrelated components in four quadrants:

- The first quadrant shows the variation of bond strength with axial displacement. It represents the pullout test graph;
- The second quadrant relates the confining pressure at the bolt-grout interface to the bond strength using Equation (2);
- The third quadrant shows the relationship between axial and lateral displacements. Since the apparent dilation angle is not constant, the relation is non-linear and asymptotically approaches an ultimate lateral displacement;
- The dilation acts outward on the grout column and creates the interface pressure as illustrated by the fourth quadrant. The straight lines show that the grout may split under the dilational pressure.

In the fourth quadrant of the BSM, the dilatational behaviour of grout is: (1) elastic; (2) fully split; or (3) a transition zone of partially split with an elastic portion. In the elastic grout expansion (Figure 7), the radial displacement at the bolt-grout interface can be derived from the plane strain thick-walled cylinder equations:

\[ u_{2G} = \frac{(1+\nu_g)(1-2\nu_g)}{E_g} \frac{p_1 r_1^2 p_2 r_2^2}{r_1^2 - r_2^2} + \frac{(1+\nu_g)(p_1 - p_2) r_1^2 r_2^2}{E_g} \frac{1}{r_1} \]  

(3)

\[ u_{2G} = \frac{(1+\nu_g)(1-2\nu_g)}{E_g} \frac{p_1 r_1^2 p_2 r_2^2}{r_1^2 - r_2^2} + \frac{(1+\nu_g)(p_1 - p_2) r_1^2 r_2^2}{E_g} \frac{1}{r_2} \]  

(4)

The radial displacement of the rock, induced by an internal pressure in a circular hole of radius in an infinite medium, is given by:

\[ u_{2R} = \frac{p_2 r_2^2}{E_r} \left( 1 + \nu_r \right) \]  

(5)

Combining the above Equations (3), (4) and (5), the displacement at the bolt-grout interface can be expressed in terms of the internal pressure in the form:

\[ u_{1G} = \frac{(1+\nu_g)(1-2\nu_g)}{E_g} \frac{r_2^2 r_1^2}{r_2^2 - r_1^2} \frac{p_1}{r_1} + \frac{(1+\nu_g)(1-\nu_g)}{E_g} \frac{r_2^2 r_1^2}{r_2^2 - r_1^2} \frac{p_1}{r_1} \]  

(6)

where: \( X = \frac{1+\nu_g}{E_g} + \frac{1+\nu_g}{E_g(r_2^2 - r_1^2)} \left( 1 - 2\nu_g \right) r_2^2 + \frac{1+\nu_g}{E_g(r_2^2 - r_1^2)} \left( 1 - 2\nu_g \right) r_1^2 + r_2^2 \)
If the tangential stress exceeds the tensile strength of the grout, grout will fully split and the tangential stress in the grout column becomes zero. This changes the thick-walled grout cylinder to a wedge-shaped geometry. The new state of stress can be found according to the wedge theory as $p_1 r_2 = p_1 r_2$. Consequently, the difference of the displacements between the boundaries of the split grout column can be calculated:

$$u_{1s} - u_{2g} = \frac{1-v_g^2}{E_g} p_1 r_1 \ln \frac{r_2}{r_1}$$  \hspace{1cm} (7)

Substituting $u_{2g}$ with Equation 5, the displacement at the bolt-grout interface for the totally split grout cylinder is:

$$u_{1g} = r_1 \left( 1 + \frac{v_r}{E_r} + \frac{1-v_g^2}{E_g} \ln \frac{r_2}{r_1} \right) p_1$$  \hspace{1cm} (8)

In the transition zone of above two cases, the interface pressure is obtained via elasto-plastic behaviour:

$$p_1 = \sigma_T (r_2^2 - r_e^2) \left/ \left( \frac{r_t}{r_e} r_e^2 + r_2^2 \left( 1 - \frac{r_t}{r_e} \right) \right) \right.$$  \hspace{1cm} (9)

where: $r_e =$ radius of the cracked zone and $X$ is obtained using $r_1 = r_e$ in Equation (6)

Thus, the dilation for partially split grout is found by algebraically adding the displacements for split and intact grout:

$$u_{1g} = \left( \frac{r_1}{r_t} \left( \frac{1+v_g}{E_g} \left( \frac{r_t^2 - r_e^2}{r_t^2 - r_e^2} \right) r_e + \frac{1+v_g}{E_g} \left( \frac{1}{r_t^2 - r_e^2} \right) r_2^2 \right) + r_1 \left( \frac{1+v_r}{E_r} + \frac{1-v_g^2}{E_g} \ln \frac{r_2}{r_1} \right) \right) p_1$$  \hspace{1cm} (10)

This equation is only applicable for $r_t < r_e < r_2$ and $u_{tg}$ is not a linear function of $p_t$, because the length of crack $r_e$ is also a function of $p_t$. A closed-form solution could not be found and, hence, $u_{tg}$ is determined iteratively starting from $r_e=r_t$.

To complete BSM, “dilation limit” must be determined. While failure occurs, the area of the grout teeth in contact with the bolt decreases but the stress acting on an individual tooth increases leading eventually to complex modes of failure. An empirical model was chosen to describe it as:
\[ u_{1g} = u_0 \left(1 - \frac{p_1}{\sigma_c}\right)^{B/\sigma_c} \]  

(11)

Where \( u_0 \) is maximum dilation \( \approx \) teeth height; \( \sigma_c \) is compressive strength of grout and \( B \) is a constant which can be determined from pullout data.

Hyett et al. (1992; 1995; 1996) carried out series of laboratory and field pullout tests to investigate the major factors influencing the bond capacity of grouted cable bolts. All tests were conducted on 15.9 mm diameter 7-strand cable grouted using type 10 Portland cement pastes. Their results indicate that cable bolt capacity most critically depends on the cement properties, embedment length and radial confinement. They found that cable bolt capacity increased with embedment length although not in direct proportionality. Furthermore, in general higher capacities were obtained under conditions of higher radial confinement.

From pullout tests, two failure modes have been observed. One mode involves radial splitting of the concrete cover surrounding the cable, and the other shearing of the cable against the concrete. The radial splitting mechanism is induced by the wedging action between the lugs of the bar and the concrete. This exerts an outward pressure on the inside of the concrete annulus that is balanced by the induced tensile circumferential stress within the annulus. However, if the tensile strength of the cement is exceeded, radial splitting will occur, the circumferential stress in the concrete annulus will be reduced to zero as will the associated reaction force at the steel-concrete interface, so resulting in failure. The shearing mechanism involves crushing of the concrete ahead of the ribs on the bar, eventually making pull out along a cylindrical frictional surface possible. Thus, it can be concluded that as the degree of radial confinement increased the failure mechanism changed from radial fracturing and lateral displacement of the grout annulus under low confinement, to shear of the cement flutes and pull out along a cylindrical frictional surface under high confinement.

The successive stages in the failure during a pull test were summarized schematically as shown in Figure 8, Hyett et al. (1992). In the essentially linear response (stage 1), as the experimental initial stiffness is significantly less than that predicted from elastic solutions, Hyett argued that the adhesional bond between the cable and the cement is negligible because (1) the cement paste is porous, and (2) the bond is not continuous but instead comprises a series of point contacts. Consequently, the mechanical interlock and frictional resistance is related with the initial linear response during a pull test, although partial adhesion probably involves additional components. From stage 2, the failure mechanism is dependent on the radial confining pressure. The stress drop may correspond to radial fracturing of the grout annulus and/or shear failure through the grout flutes. From then on, as cable displacement increases, the radial confining pressure is controlled by the potential for greater geometric mismatch between the cable and cement flutes. How far the individual wedges that now comprise the grout annulus can be pushed aside is determined by the radial stiffness of the confining medium. When the radial stiffness is low the favourable failure mechanism is lateral displacement of the wedges; when it is higher, dilation is suppressed and failure is more likely to occur by shear of the grout flutes and pull out along a cylindrical frictional surface.

![Figure 8 - Successive stages in the failure during a pull test, after Hyett et al. (1992)](image-url)
develop a frictional-dilational model for cable bolt failure in a mathematical form which is amenable to implementation in numerical programs.

Figure 9 - Left: Boundary conditions of the cable-bolt system, and in this research a constant radial pressure is applied. Right: Terminology and sign convention.

In this model, the bond strength is frictional, so it depends on the pressure generated at the cable-grout interface, \( p_1 \), which in turn depends on the reaction force generated at the borehole wall caused by dilation during bond failure. The frictional resistance can be catalogued into:

1. for dilational slip: \( f = A \cdot p_1 \cdot \tan(\phi_{g-s} + i) \)  
   \[ (12) \]
2. for non-dilational unscrewing: \( f = (A \cdot p_1 \cdot \tan\phi_{g-s})/\sin\alpha + Q \)  
   \[ (13) \]
3. for shear failure of the cement flutes: \( f = A(c + p_1\tan\phi_g) \)  
   \[ (14) \]

In which \( i = \) dilation angle; \( A = \) interface contact area; \( \phi_{g-s} = \) sliding friction between grout and steel; \( \phi_g = \) internal angle of friction for grout and \( c = \) grout cohesion.

Micrographs reveal that shearing of the grout flutes only occurred within 75 mm of the exit point. The only viable explanation is that, along the majority of the test section, failure involves unscrewing of the cable from the cement annulus. To include the unscrewing effect, \( Q \) is introduced as the component of the pull out force required to untwist the free length of cable. Based on work considerations, the formula is given by:

\[
Q = 4\pi^2Cu_0/\left[2^2(u_a + L_f)\right]
\]

In which \( C = \) torsional rigidity of cable; \( l = \) pitch length and \( L_f = \) free length of the cable between test and anchor sections.

After 50 mm of axial displacement, the radial dilations measured at the midpoint of the test section are from approximately 0.15 mm for 1 MPa radial confining pressure, to 0.02 mm for 15 MPa. Since the dilation angles are small (\( i < 0.2^\circ \)), the pull force component related to dilational slip may be ignored. Thus, the axial pull out force, \( F_a \), may be approximately written as:

\[
F_a = \frac{l_2}{l} \cdot A\left(c + p_1\tan\phi_g\right) + \left(1 - \frac{l_2}{l}\right) \times \frac{Ap_1\tan\phi_{g-s}}{\sin\alpha} + Q
\]

In which \( L = \) embedment length and \( L_s = \) sheared length of the grout flutes. However, as the shear failure length is undeterminable, an average coefficient of friction (\( \phi' \)) over the whole test section is introduced, then:

\[
F_a = A \cdot p_1 \cdot \tan(\phi') + Q
\]

The average coefficient of friction angle can be evaluated as the slope of the linear portion of plot \( (F_a-Q)/A \) against confine pressure, which in turn is independent of confining pressure.

In the cable grout interface, the pressure dependent closure is assumed to be hyperbolic, and then the total dilation due to splitting may be written as:
Radial splitting dilation: 
\[ v_r = v_{r0} - \frac{p_1 v_{r0}}{K_{r0} v_{r0} + p_1} \]

Where \( v_{r0} \) is the dilation generated by splitting when \( p_1 = 0 \), and \( K_{r0} \) represents the radial stiffness (MPa/mm) of the cable-grout interface immediately following splitting. Appropriate values for \( K_{r0} \) and \( v_{r0} \) can be determined from the radial displacement-axial displacement plots.

The cable is not rigid, radial contraction of the cable due to the application of \( p_1 \) is considered and evaluated as \( p_1/K_{rc} \). Therefore:

Radial splitting dilation: 
\[ v_r = v_{r0} - \frac{p_1 v_{r0}}{K_{r0} v_{r0} + p_1} - \frac{p_1}{K_{rc}} \]

Based on this, Hyett then assumed a simple mechanical model to characterise the radial deformability of the cable-grout interface after the cable has been pulled by an amount \( u_{r1} \):

\[ u_{r1} = \frac{k_1}{p_1}(u_a - 1) + v_{r0} - \frac{p_1 v_{r0}}{K_{r0} v_{r0} + p_1} - \frac{p_1}{K_{rc}} \]

(18)

Where \( u_a \) is axial displacement and \( k_1 \) is empirical constant determined by best fit.

Combining Equations (17) and (18), a differential formulation for the deformability of the cable joint interface during bond failure, i.e. tangent stiffness matrix, can be obtained;

\[
\begin{bmatrix}
\frac{dF_s}{dp_1}
\end{bmatrix} = \begin{bmatrix}
K_{11} & K_{12} & K_{13} \\
K_{21} & K_{22} & K_{23} \\
K_{31} & K_{32} & K_{33}
\end{bmatrix} \begin{bmatrix}
\frac{du_r}{du_a}
\end{bmatrix}
\]

The behaviour of grout annulus is discussed in three scenarios based on the assumption that it has fully split after 1 mm of axial pull. Thereafter, the cement annulus will be unable to support a tensile tangential stress. That is, the fracture is free to open or close depending on confining pressure \( p_2 \) and dilation \( u_{r1} \).

While the tangential stresses are compressive, the grout annulus will behave identically to an intact hollow cylinder, and the plane strain elastic solution for a thick walled hollow cylinder can be applied. For the case when the radial fractures are fully open, a series of individual grout wedges are formed. The solution to the stresses and dilations are identical with the BSM model of Yazici and Kaiser (1992).

If the radial fractures are partially open, i.e. the outer annulus is in compression but the inner annulus is in tension, the tangential stress at the common boundary must be zero. Thus, the radius for which fractures are open can be solved:

\[ r_c = \frac{p_2 r_1^2 r_2 - p_1 r_1^2 - p_1 r_2^2}{p_1 r_1} \]

(19)

Consequently, the radial displacement equation and stiffness matrix can be formulated.

**FAILURE MODES OF INTERFACIAL SHEAR FAILURE**

It is known that the ultimate failure of rockbolts may occur: (a) in the bolt, (b) in the grout, (c) in the rock, (d) at the bolt-grout interface, (e) at the grout-rock or steel tube interface and (f) a combination of these failure modes, Ren et al. (2010). This section is concerned with the very common debonding failure at the bolt-grout interface. Under the debonding failure, zero thickness interface represents the materials adjacent to the critical surface where debonding occurs. The deformation of the surrounding rock or grout is often negligible, i.e. all deformations in the surrounding grout and rock outside the critical interface are lumped in the interface. As a result, the bolt can be assumed to be under uniaxial tension and the bolt-grout interface layer under interfacial shear deformation only.

For the steel bar bolting system, Indraranta and Kaiser (1990 a,b) point out that failure takes place along the weakest interface unless the bolt itself yields. The product of the hole diameter and the bond strength of the grout/rock interface is greater than the product of the bolt diameter and the bond strength of the bolt/grout interface. Hence, failure may occur by the bolt pulling out, as is sometimes observed.
in the case of smooth rebars. Such failure of grouted bolts can be prevented by shaping the bolt surface. However, failure may also initiate within the grout annulus or at the grout/rock interface, owing to impaired grout strength development or poor adhesion of grout to the borehole wall.

Aydan (1989) carried out a series of push and pull out tests to investigate the anchorage mechanism of grouted rock bolts and the effect of various parameters such as the ratio of the bolt to borehole diameter and the behaviour of the bolt to grout interface under triaxial stress. Two steel bars 13 mm and 19 mm in diameter were tested. The results showed that the load bearing capacity of bolts was 25% higher in push out tests than pull out tests. Similar tests carried out by Aziz, et al. (2009) resulted in a load bearing capacity increase at around 10% in favour of push tests. This increase in push test values was attributed to the Poisson’s ratio effect (the radial stress is of compressive character in the push out case while it tends to become tensile in the pull out tests). Ayden’s investigation showed that an increase in bearing capacity was attributable to the normal compressive stress resulting from the geometric dilation of the surface. Aydan (1989) suggested that shearing might occur along one of the surface of weakness in the rock bolt system (grout-rock interface and bolt-grout interface), and classified the failure modes in the push pull tests as follows:

1. Failure along the bolt-grout interface. This occurred in every test on bars with a smooth surface and deformed bars installed in a large borehole;
2. Failure along the grout-rock interface. This occurred in deformed bars installed only in smaller diameter boreholes;
3. Failure by splitting of grout and rock annulus.

Aydan (1989) observed that although shearing failure along one of the interfaces was the main cause, some samples split without confining pressure. This was attributed to geometrical dilation of the bolt-grout interface during shearing, which caused an internal pressure on the borehole.

Li and Stillborg (1999) developed an analytical model for predicting the behaviour of rock bolts under three different conditions 1) for bolts in pull out tests, 2) for bolts installed in a uniformly deformed rock mass and, 3) for bolts subjected to the opening of rock joints.

The development of these models was based on the description of the mechanical coupling at the interface between the bolt and grout medium for the grouted tests. Based on the exponential decay theory and decoupling of the bond, they constructed a model for the shear stress along a fully grouted bolt as shown in Figure 10.

**Figure 10 - Distribution of shear stress along a fully grouted rockbolt subjected to an axial load.**

Left: before decoupling occurs. Right: completely decoupled with a zero shear stress until \( x_0 \), partially decoupled with a residual shear strength \( s_r \) until \( x_1 \), from \( x_1 \) to \( x_2 \) the residual shear strength linearly increases to the peak strength \( s_p \) and then exponentially towards the far end of the bolt. After Li and Stillborg (1999)

Before decoupling occurs at the interface for fully grouted rockbolts, the attenuation of the shear stress is expressed as:

\[
\tau_b = \frac{a}{2} \sigma_{bo} e^{-2a \frac{x}{s_p}}
\]

where

\[
a^2 = \frac{2 \mu_b g}{E_b \left( \frac{\sigma_{bo}}{g} \ln \frac{\sigma_{bo}}{g} + \frac{\sigma_{bo}}{g} \ln \frac{\sigma_{bo}}{g} \right)}
\]
After decoupling occurs, for equilibrium of the bolt, the applied load $P_0$ should equal the total shear force at the bolt interface, i.e.

$$P_0 = \pi d_b \int_{x_0}^{x} \tau_0 \, dx = \pi d_b \left[ \tau_0 (x_1 - x_0) + \frac{1}{2} s_p \Delta (1 + \omega) + \frac{d_b}{2a} s_p \left( 1 - e^{-\frac{2a}{d_b}(L-x_2)} \right) \right]$$

(21)

And the maximum applied load can be expressed as:

$$P_{0-max} = \pi d_b s_p \left[ \omega \left( L + \frac{d_b}{2a} \eta_0 \Delta - x_0 \right) + \frac{1}{2} \Delta (1 + \omega) + \frac{d_b}{2a} (1 - \omega) \right]$$

(22)

Where: $L=$length of the bolt; $\Delta = x_2 - x_1$; $\omega = s_r / s_p$; $x_2 = x_0 + \frac{1}{2a} \left( \frac{2P_0}{\pi d_b s_p} - \frac{d_b}{a} - \Delta (1 + \omega) \right)$

Ground anchor is also one common reinforcing method in civil engineering. It can make effective use of the soil potential and enhance its self-stability. According to the modes of load transfer, anchorage can be divided into three types: tension, pressure and shearing. Tension anchor is more commonly used and its reinforcement mechanism is to transmit the supporting force from anchor to stable stratum through bonding resistance. There are three failure modes of the tension anchor, that is, 1) anchor breaking, 2) anchor and grout body bonding failure, and 3) the anchorage body and soil shear failure. The former two failure modes hardly occur in practice so the main task of soil anchorage design is to determine the side resistance distribution between the anchorage body and surrounding soil to avoid the last failure mode. Side resistance distribution of the anchorage segment is conventionally assumed to be uniform. However, some recent investigations have shown that the side resistance is not uniform but has a peak in the front part and then decreases gradually and finally approaches zero. Based on this, Xiao and Chen (2008) presented a model of the tensile anchor load transfer mechanism using shear displacement method.

In their work, the softening feature of the soil was considered. The shear stress-strain relationship of soil surrounding the anchorage body was simplified into a tri-lines model consisting of an elastic phase, an elasto-plastic phase and a residual phase (Figure 11).

![Figure 11 - Left: Schematic diagram of tension type anchor. Right: Relationship between shear stress and strain of soils, Xiao and Chen, 2008.](image)

The shear stress-strain relationship can be expressed as:

$$\gamma = \begin{cases} \frac{\tau}{G} & \text{elastic phase} \\ \frac{\tau_1}{G} + (\tau - \tau_1)/K_2 & \text{elasto - plastic phase} \\ \frac{\tau_2}{G} + (\tau_2 - \tau_1)/K_2 & \text{residual phase} \end{cases}$$

(23)

According to elastic theory, the shear stress and strain of soil surrounding a structural pile are

$$\tau = \tau_0 \frac{r_0}{r} \quad \text{and} \quad \gamma = ds/dr$$

(24)

Where $r_0 =$distance from any point in the soil to pile center; $r_0 =$radius of anchorage; $\tau_0 =$shear stress of anchorage surface; $s =$soil displacement. Consequently, at a depth $z$, the soil displacement at the anchor interface can be obtained as:
Where $r_m$ is the soil radius surrounding anchorage body where shear displacement can be ignored.

The shear displacement, $s$, is also a function of depth $z$. According to the definition, the shear displacement in the phase I and II shown in Figure 11 are

$$s = \begin{cases} \frac{\tau_0}{g} \ln \frac{r_m}{r_0} & \text{elastic phase} \\ \frac{\tau_1}{g} \ln \frac{r_m}{r_0} + \frac{\tau_2}{g} \ln \frac{r_0}{\tau_1} & \text{elastoplastic phase} \\ \text{uncertainty} & \text{residual phase} \end{cases}$$

(25)

The governing equation is still

$$\frac{d^2 s}{dz^2} - \frac{2}{r_0 k} \tau = 0$$

(27)

When only elastic deformation exists, this governing equation can be easily solved. When the surrounding soil enters into the elastoplastic phase, the equation is a transcendental equation but they can be solved in close form. The close form solution is doubtful nevertheless, they provided a method to potentially predict the full range behaviour of the anchorage.

Ren et al. (2010) developed a closed-form solution for the prediction of the full-range mechanical behaviour of fully or partially-grouted rockbolts under tension. In this solution, a tri-linear bond slip model is used to accurately model the interfacial debonding mechanism between the grout and the bolt. The full-range behaviour consists of five consecutive stages: elastic stage, elastic-softening stage, elastic-softening-debonding stage, softening-debonding stage and total debonding stage. For each stage, closed-form solutions for the load-displacement relationship, interfacial shear stress distribution and bolt axial stress distribution along the bond length were derived. The ultimate load and the effective anchor length were also obtained. Their analytical model was calibrated with two pullout experimental studies. The predicted load-displacement curves as well as the distributions of the Interfacial Shear Stress (ISS) and the bolt axial stress are in close agreement with test results.

This study adopts a tri-linear bond-slip model, as shown in Figure 12, in which an ascending branch up to the peak stress at $(\delta_f, \tau_f)$ followed by a softening branch down to $(\delta_r, \tau_r)$, and then a horizontal branch representing the non-zero residual frictional strength $\tau_r$ after complete debonding.

![Figure 12 - Tri-linear bond-slip model](image)

Let $k$ be the ratio of the residual strength $\tau_r$ to the peak stress $\tau_f$, so $\tau_r = k \tau_f$, the shear stress can be expressed as:

$$s = \begin{cases} \frac{\tau_0}{g} \ln \frac{r_m}{r_0} & \text{elastic phase} \\ \frac{\tau_1}{g} \ln \frac{r_m}{r_0} + \frac{\tau_2}{g} \ln \frac{r_0}{\tau_1} & \text{elastoplastic phase} \\ \text{uncertainty} & \text{residual phase} \end{cases}$$
Based on force equilibrium, the governing equations of the grouted rockbolt and the axial stress in the bolt are

\[
\frac{d^2\delta}{dx^2} - \frac{\delta_f}{\tau_f} \lambda^2 \tau(\delta) = 0
\]  

\[
\sigma_b = \frac{2\tau_f}{\delta_f E_b \lambda^2} \frac{d\delta}{dx}
\]

Where \( \lambda^2 = \frac{2\tau_f}{\delta_f E_b \lambda^2} \).

They can be solved once \( \tau(\delta) \) is defined. Figure 13 illustrates the evolution of ISS distribution and corresponding load-displacement curve.

Figure 13 - Left: Evolution of interfacial shear stress distribution and propagation of debonding. (a,b) Elastic stage; (c,d) elastic-softening stage; (e,f) elastic-softening-debonding stage; (g) softening-debonding stage; (h,i) debonding stage; I, II and III represent elastic, softening and debonding stress states respectively. Right: Typical full-range theoretical non-dimensional load-displacement curve. After Ren et al, (2010).

In the elastic stage, the solutions of the governing equations are

\[
\delta = \frac{\delta_s P A}{2\pi \tau_f \cosh(\lambda_1 L)}; \quad \tau = \frac{P A}{2\pi \tau_f \sinh(\lambda_1 L)}; \quad \sigma_b = \frac{P \sinh(\lambda_1 L)}{2\pi \tau_f \cosh(\lambda_1 L)}
\]

Where \( \lambda_1^2 = \frac{\delta_f}{\delta_s} \lambda^2 = \frac{2\tau_f}{\delta_s E_b \lambda^2} \).

The slip at the loaded end with \( x = L \) is defined as the displacement of the rockbolt and is denoted as \( \Delta \). The following load–displacement expression can then be obtained

\[
P = \frac{2\pi \tau_f \tan h(\lambda_1 L)}{\delta_1 A} \Delta
\]
The elastic stage ends when the shear stress reaches the bond shear strength \( \tau_r \) at a slip of \( \delta_1 \) at \( x = L \) (Point A in Figure 13). Setting \( \Delta = \delta_1 \), the load at the initiation of interface softening is found to be

\[
P_{\text{soft}} = \frac{2\pi r_0 \tau_f \tanh (\lambda_1 L)}{\lambda_1}
\]

(35)

As the pullout force increases, softening commences at the loaded end \( (x = L) \) and the peak shear stress is transferred towards the embedded end, as shown in Figure 13-c (state ii). With the development of the softening length \( a \), the load \( P \) continues to increase because more interface is mobilised to resist the pullout force. At the end of this stage (point B in Figure 13), \( P \) reaches the debonding load \( P_{\text{deb}} \). The following differential equations for the elastic–softening stage can be obtained:

\[
\frac{d^2 \delta}{dx^2} - \lambda_1^2 \delta = 0 \quad \text{when } 0 \leq \delta \leq \delta_1
\]

(36)

\[
\frac{d^2 \delta}{dx^2} + (1 - k) \lambda_1^2 \delta = \lambda_2^2 (\delta_f - k \delta_1) \quad \text{when } \delta_1 \leq \delta \leq \delta_f
\]

(37)

Where \( \lambda_2^2 = \frac{\delta_f - \delta_1}{\delta_f - \delta_1} \lambda_1^2 = \frac{2 \tau_f}{(\delta_f - \delta_1) E b r_f}

The boundary conditions are: \( \sigma_b = 0 \) at \( x = 0 \); \( \delta = \delta_1 \) or \( \tau = \tau_f \) at \( x = L - a \) and \( \sigma_b = \frac{P}{\pi b} \) at \( x = L \).

The solution for the elastic region of the interface with \( 0 \leq x \leq L - a \) is

\[
\delta = \frac{\delta_1 \cosh (\lambda_1 x)}{\cosh [\lambda_1 (L - a)]}, \quad \tau = \frac{\tau_f \cosh (\lambda_1 x)}{r_b \lambda_1 \cosh [\lambda_1 (L - a)]}; \quad \sigma_b = \frac{2 \tau_f \sinh (\lambda_1 x)}{r_b \lambda_1 \cosh [\lambda_1 (L - a)]}
\]

(38)

The solution for the softening region with \( L - a \leq x \leq L \) is

\[
\delta = (\delta_f - \delta_1) \left[ \frac{\lambda_2 \sin (\lambda_2 (x - L + a) \sqrt{1 - k}) \tanh [\lambda_1 (L - a)]}{\lambda_1 \sqrt{1 - k}} - \frac{\cos [\lambda_2 (x - L + a) \sqrt{1 - k}]}{1 - k} + \frac{\delta_f - k \delta_1}{(1 - k)(\delta_f - \delta_1)} \right]
\]

(39)

\[
\tau = -\tau_f \left[ \frac{\lambda_2 \sqrt{1 - k}}{\lambda_1} \sin [\lambda_2 (x - L + a) \sqrt{1 - k}] \tanh [\lambda_1 (L - a)] - \cos [\lambda_2 (x - L + a) \sqrt{1 - k}] \right]
\]

(40)

\[
\sigma_b = -\frac{2 \tau_f}{\lambda_2 r_b \sqrt{1 - k}} \left[ \frac{\lambda_2 \sqrt{1 - k}}{\lambda_1} \cos [\lambda_2 (x - L + a) \sqrt{1 - k}] \tanh [\lambda_1 (L - a)] + \sin [\lambda_2 (x - L + a) \sqrt{1 - k}] \right]
\]

(41)

And

\[
P = \frac{2 \pi r_0 \tau_f}{\lambda_2 \sqrt{1 - k}} \cos \left[ a \lambda_2 \sqrt{1 - k} \tanh [\lambda_1 (L - a)] + \sin \left[ a \lambda_2 \sqrt{1 - k} \right] \right]
\]

(42)

\[
\Delta = (\delta_f - \delta_1) \left[ \frac{\lambda_2 \sin [a \lambda_2 \sqrt{1 - k}] \tanh [\lambda_1 (L - a)]]}{\lambda_1 \sqrt{1 - k}} - \cos [a \lambda_2 \sqrt{1 - k}] + \frac{\delta_f - k \delta_1}{(1 - k)(\delta_f - \delta_1)} \right]
\]

(43)

Debonding initiates at the loaded end when \( \tau \) reduces to \( \tau_r \), and \( x = L \) into shear equation leads to

\[
\frac{\lambda_2 \sqrt{1 - k}}{\lambda_1} \sin [\lambda_2 a \sqrt{1 - k}] \tanh [\lambda_1 (L - a)] - \cos [\lambda_2 a \sqrt{1 - k}] = -k
\]

(44)

Thus the softening length \( a \) at the initiation of debonding at the loaded end, denoted as \( a_d \), can be solved as:

\[
a_d = \frac{1}{\lambda_2 \sqrt{1 - k}} \sin^{-1} \left[ \frac{\sqrt{\delta_f - \delta_1} \left[ \delta_f - k \delta_1 - k \delta_1 / (\delta_f - \delta_1) \right]}{\delta_f - k \delta_1} \right]
\]

(45)

Debonding load \( P_{\text{deb}} \) can be found as

\[
P_{\text{deb}} = \frac{2 \pi r_0 \tau_f}{\lambda_2 \sqrt{1 - k}} \left[ \frac{\lambda_2 \sqrt{1 - k}}{\lambda_1} \cos [a_d \lambda_2 \sqrt{1 - k}] + \sin \left[ a_d \lambda_2 \sqrt{1 - k} \right] \right]
\]

(46)
Once the shear stress decreases to $\tau$, at $x = L$, debonding initiates at the loaded end. As debonding propagates, the peak shear stress continues to move towards the embedded end. Thus there are three possible stress states within the bond length: the elastic state (state I), the softening state (state II) and the debonding state (state III) (Figure 13-e). The debonded length is denoted by $d$ and the solution for the elastic-softening zones, i.e. Equation (39) (40) and (41), are still valid if $L$ is replaced by $(L - d)$. The differential equation for debonding zone can be obtained by substituting Equation (30) into Equation (31). The solution for the debonding region with $L - d \leq x \leq L$ is given by

$$\delta = \frac{\delta_f}{2\lambda_2\lambda_2\sqrt{1-k}} \left[ \lambda_1^2 \lambda_2 \sqrt{1-k} (2 + kd^2\lambda^2 + k\lambda^2(L - d)^2 - 2kd\lambda^2(L - x)) - 2\lambda^2(L - x - d)\sin[\pi(\lambda_2 a_d + k) - 2\lambda_2 a_d x - d\cos[\pi(\lambda_2 a_d + k)]\tanh[\lambda_1(a_d + d - L)] \right]$$

$$\tau = k\tau_f$$

$$\sigma_b = \frac{2\tau_f}{\tau_b} \left[ k(d - L + x) + \frac{\sin(\lambda_2 a_d \sqrt{1-k})}{\lambda_2 \sqrt{1-k}} - \frac{\cos(\lambda_2 a_d \sqrt{1-k}) \tanh[\lambda_1(a_d + d - L)]}{\lambda_1} \right]$$

And

$$P = \frac{2\pi\tau_b}{\lambda_2 \sqrt{1-k}} \left[ \lambda_2 \sqrt{1-k} \cos(a_d \lambda_2 \sqrt{1-k}) - \sin(a_d \lambda_2 \sqrt{1-k}) \right] + 2\pi\tau_b \tau_f d$$

$$\Delta = \frac{\delta_f}{2\lambda_1 \lambda_2 \sqrt{1-k}} \left[ \lambda_1 \lambda_2 \sqrt{1-k} (2 + kd^2\lambda^2 + 2\lambda^2d\sin[\pi(\lambda_2 a_d + k)]) - 2\lambda^2(\lambda_2 \sqrt{1-k}) \cos[\pi(\lambda_2 a_d + k)] \tanh[\lambda_1(a_d + d - L)] \right]$$

$P$ reaches its maximum value when

$$\tanh[\lambda_1(L - d - a_d)] = \frac{\cos(a_d \lambda_2 \sqrt{1-k}) - k}{\cos(a_d \lambda_2 \sqrt{1-k})}$$

Solve $d$ at the ultimate load, denoted as $d_{ult}$ as

$$d_{ult} = L - a_d - \frac{1}{\lambda_1} \tanh^{-1} \sqrt{\frac{\cos(a_d \lambda_2 \sqrt{1-k}) - k}{\cos(a_d \lambda_2 \sqrt{1-k})}}$$

The above analysis shows that the full-range mechanical behaviour of rockbolts under tension consists of five distinct stages. The important points are point A($P_1$, $u_1$) corresponding to the initiation of interface softening, point B($P_2$, $u_2$) corresponding to the initiation of debonding, and point C($P_3$, $u_3$) corresponding to the ultimate load. These three points may be identified from an experimental load–displacement curve, and used to calibrate the parameters in the tri-linear bond-slip model (Figure 14).

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Figure 14 - Load-displacement curve used to calibrate the parameters in the tri-linear bond-slip model, after Ren et al. (2010).
DISCUSSION AND CONCLUSIONS

Rockbolts are widely used in mining and tunneling engineering to support underground excavation or to stabilise a jointed rock mass. In this work up-to-date failure modes of several kinds of rockbolting reinforcement system were presented and discussed. The stress state in a concrete beam due to bond forces from a steel bar is analysed. Upon the specific bond failure modes, i.e. cone cracks and radial cracks, the stresses were calculated for an elastic stage, a plastic stage and an elastic stage with internal cracks.

For fully grouted cable bolts, the BSM explains observations from laboratory pull-out tests by predicting the elastic, partially split and fully split grout behaviour. The load level during splitting of grout is a function of the grout confinement by the rock, grout stiffness and grout strength. The grout column of cable bolts confined by relatively soft or disturbed rock, with soft or low tensile strength grout will be susceptible to grout splitting which in turn leads to a reduction in ultimate bond strength.

Hyett et al. (1995) emphasizes that the failure involves unscrewing of the cable from the cement annulus. This type of failure mechanism is due to the helical form and low torsional rigidity of a seven-wire strand, and it distinguishes the mechanical behaviour of cable bond failure from a solid deformed bar.

Finally, a closed-form solution for predicting the full-range behaviour of rockbolts under tension based on a tri-linear bond-slip model is presented. Formulations for the shear slip and shear stress on the grout–bolt interface, the load–displacement relations, and the axial stress in the bolt, have been derived for each of the five distinct loading stages. The control parameters in the solution can be calibrated from pullout test data. It offers a theoretical basis of the rockbolt behaviour under tension, and provides practical application. Once the bond-slip model is calibrated using the analytical solution from pullout tests, it can be used in the modelling of complex engineering problems.

REFERENCES


DESIGNING COAL MINE DEVELOPMENT GALLERIES FOR ROOM AND PILLAR MINING FOR CONTINUOUS MINER OPERATIONS - INDIAN EXPERIENCE

Mani Ram Saharan¹, Prabir Kumar Palit² and Kasaraneni Ramachandra Rao³

ABSTRACT: Most of the about 300 underground coal mines in India operate with room and pillar mining method using drill and blast cyclic operations. Output per man shift from these mines has been stagnant since decades and a cause of concern. Introduction of continuous miner technology, though it works for 10% of its cycle time, is considered as an appropriate technology to boost productivity from already developed coal mining properties. This paper briefly describes Indian experience with using the continuous miner technology in a few of its mines. The paper also projects geo-technical conditions for the mines planned to use this technology. A case study is explained for geotechnical aspects of designing development galleries of a coal mine. The design procedure includes empirical rock mass characterisation, performance appraisal of the proposed roof support system, geotechnical instrumentation to characterise roof rock behaviour and numerical modelling for designing the operations.

INTRODUCTION

Energy sector demand for India is rising at a pace of 10-12% per annum. At present the coal sector is contributing more than 55% of the energy demands for the country. It is projected that the national demand will reach 731 million tonnes in 2011-12 whereas the domestic supply will have to be stretched to 680 million tonnes to meet the energy requirements of the country. A major thrust for capacity creation in the nationalised coal sector has been implemented to achieve 680 million tonnes of coal production during terminal year XIth plan. As far as underground is concerned, infusion of modern technology power support longwall working, continuous miners, mechanisation of support system has been envisaged. The Continuous Miner (CM) is considered as the most appropriate intermediate technology.

The efficiency of coal production from underground coal mines is evaluated with Output per Man Shift (OMS) in coal engineering parlance and OMS from Indian underground mines has been stagnant at around one for a long. This OMS figure is considerably lower in comparison to other countries where OMS of more than 20 is a normal figure. The low figure of OMS from Indian coal mines is due to the fact that the mines are operating with work force intensive technology with drill and blast cyclic operations.

Mass production technology using CM is one of the suitable alternatives for Indian coal mines in order to efficiently boost the coal production from underground mines. The scenario of a higher production share from surface mines is not going to be sustainable because of reduced near surface coal reserves and other concerning issues attached with surface mining. Considering these restrictions the two state owned coal companies, Coal India Limited (CIL) and Singareni Collieries Company Limited (SCCL), have taken a lead to boost the coal production from underground mines through CM mining technology. At present five mines under different geo-mining conditions are extracting coal from previously developed square pillars with CM technology and the majority of them experienced unexpected roof fall incidents perhaps due the geo-mining conditions that were not appropriately anticipated and accounted during the planning stage. Four of the mines are using the pocket-and-fender method for coal extraction which is the least favoured method with CM technology due to safety reasons (Mark, et al., 2002). Five mines are developing coal blocks using CM technology. Three of the mines introduced CMs with a cutting drum width of 3.3 m and two have cutting drum widths of 2.7 m. This means that for economical reasons two mines shall operate with 5.4 m wide rooms and rest of the mines operate with 6.6 m room width. OMS from all these mines has shown a threefold to tenfold increase in comparison

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² Directorate General of Mines Safety (DGMS), India
³ SMS Infrastructure Limited (SMSIL), India
to the conventional mining practices and there is potential to further increase productivity from these mines should proper geotechnical planning be considered for the final extraction program.

**INDIAN EXPERIENCE WITH CONTINUOUS MINER TECHNOLOGY FOR CREATION OF ROOMS IN ROOM AND PILLAR MINING**

There are five mines, namely - GDK11, Tandsi, Kumbharkhani, Rani Atari and Chirimiri, operating with continuous miner technology in India where creations of rooms is being undertaken. Additionally, the Western Coalfields Limited (WCL) will implement continuous miner technology at its more underground (UG) mines apart from the operating two mines of Tandsi and Kumbharkhani in two phases. The mines are indicated in Table 1. The new method is more machine-oriented than the conventional mining method involving drill and blast cycles. Two of the operating mines have CMs with cutting drum width as 2.7 m implying that economic reasons dictates room width shall be at least 5.4 m while the other three mines have CM cutting drum width at 3.3 m giving the possibility for 6.6 m wide rooms. Geo-technical conditions dictating the room width can easily be ascertained by the stand-up time concept given by Bienawski (Bieniawski, 1976). Figure 1 illustrates the stand-up time concept with Rock Mass Rating (RMR) values plotted on it for some of the operating mines and planned mines. The statutory permitted room width for Rani Atari and Kumbharkhani mine is 5.4 m while Tandsi Mine is forced to work under 4.5 m room width due to poor geo-technical conditions. Chirimiri and GDK11 mine are permitted for 6 m wide room creation. Study from Figure 1 reveals that the decision to introduce CM with 3.3 m wide cutting drum for Tandsi mine was not a proper decision. The mine has a severe issue of ground control related problems caused by high horizontal stresses and a solution to deal with the stress regime should be addressed along with the creation of rooms. A proper study prior to introducing the CM technology would have helped the mine management. Figure 1 also suggests that the room widths of more than 6 m with a cut-out distance of 12 m can easily be operable parameters for the planned mines except the Nand I Mine. Rani Atari and Kumbharkhani mine has developed more than 20 km of development in the respective mines without an incident related to roof fall and both the mines used the stand-up concept to design the room width. The concept dictates that the maximum room width shall be designed in such a manner that the roof shall not fall within a period of 48 h prior to installation of the rock reinforcement measures. The critical time period of 48 h is kept in case the reinforcement measures could not be applied due to some technical problems in the mine.

**Table 1 - Mines of WCL approved for continuous miner technology adoption**

<table>
<thead>
<tr>
<th>No</th>
<th>Mine</th>
<th>Mining Area</th>
<th>Operating/ Future Project</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Saoner No I</td>
<td>Nagpur</td>
<td>Operating</td>
<td>1st Phase</td>
</tr>
<tr>
<td>2</td>
<td>Maori</td>
<td>Kanhan</td>
<td>Operating</td>
<td>1st Phase</td>
</tr>
<tr>
<td>3</td>
<td>Tawa II</td>
<td>Pathakhera</td>
<td>Operating</td>
<td>2nd Phase</td>
</tr>
<tr>
<td>4</td>
<td>Nandan–II</td>
<td>Kanhan</td>
<td>Future</td>
<td>2nd Phase</td>
</tr>
<tr>
<td>5</td>
<td>Dhankasa</td>
<td>Pench</td>
<td>Future</td>
<td>2nd Phase</td>
</tr>
<tr>
<td>6</td>
<td>Jamunia</td>
<td>Pench</td>
<td>Future</td>
<td>2nd Phase</td>
</tr>
<tr>
<td>7</td>
<td>Nand – I</td>
<td>Umrer</td>
<td>Future</td>
<td>-</td>
</tr>
</tbody>
</table>

**Figure 1 - Application of Bieniawski’s RMR for room width/cut-out distance estimation**
DESIGN OF ROOMS FOR ROOM AND PILLAR MINING - A CASE STUDY

Details of the mine

The mining area is covered by Survey of India Topo Sheet No. 64 J/5 (R.F. 1:50000). The coal reserve is known as the Vijay West Block and it is situated in the western part of Sendugarah Coalfields. The winnable reserve of Seam I, which is 11.30 Million Metric Tonnee, is grouped in four blocks based on their respective thickness of < 2.0 m, 2.0 m-2.5 m, 2.5 m-3.0m and >3.0m excluding the area under the 15 m hard cover statutory mining line, respectively. The coal block has the seam thickness varying between 2.0 m to 3.0 m with an average depth of 40 m from the surface. The seam is overlain by competent medium grained sandstone of varying thickness of Barakar Formation. Medium grained sandstone to shaly sandstone constitutes the seam floor in the mine. The coal seam before the experimental block is developed along the seam floor using room-and-pillar mining method with blasting-off-the-solid excavation technology in 4.2 m wide room dimensions and square pillars of 21 m centre to centre. The changing placement method of coal development with the CM technology is proposed for the development of the experimental coal block with 21 m square coal pillars (centre-to-centre) for room-and-pillar mining operations. The pillar size is based on Coal Mine Regulations those framed with considerations of drill and blast cyclic operations.

Authors of this paper are of opinion that smaller and rectangular pillars shall be preferred for CM technology to devise safer final extraction methods. Major design needs for the proposed method are namely, (a) a suitable room width, (b) a safe cut-out distance under which the machine can work for a limited time period without supports and (c) an effective roof rock reinforcement system for the development headings.

Geotechnical parameters and rock mass characterisation

Basic and applied geotechnical parameters for different coal measure rocks and coal has been obtained through field measurements and laboratory testing. The basic parameters include density, Young’s modulus, Poisson’s ratio, uniaxial compressive strength and sound wave velocities for different rocks. Core samples are obtained from the mine for the purposes. Applied geotechnical parameters, such as, joints persistence, joint conditions, number of joints, joint spacing and water seepage have been estimated through field measurements. These parameters are used for rock mass characterization and numerical modelling.

The basic geotechnical parameters are summarized in Table 2. Measured density of coal, fine grained sandstone and coarse grained sandstone is found as 1.29 t/m³, 2.23 t/m³ and 1.78 t/m³, respectively. The first cycle slake durability index values of 97% for coal and 93% for fine grained sandstone, 93% for medium grained sandstone and 83% for coarse grained sandstone are measured. The UCS values are obtained through Point Load Index testing on core samples following guidelines by International Society for Rock Mechanics (ISRM, 1985) and Bureau of Indian Standards (BIS:8764, 2003). The core samples are tested for diametral and axial strengths; UCS values for fine grained sandstone, medium grained sandstone and coarse grained sandstone are obtained as 22.6 MPa, 16.3 MPa and 9.0 MPa, respectively. P-wave values are also obtained for the samples in order to indirectly assess the rock strength following the suggested procedure by International Society for Rock Mechanics (ISRM, 1978). The average P-wave velocities for fine grained sandstone, medium grained sandstone, coarse grained sandstone and coal samples along the axial direction are obtained as 1.44 km/s, 1.73 km/s, 2.25 km/s and 0.40 km/s, respectively. Estimated values for Young’s Modulus for coal, fine grained sandstone, medium grained sandstone and coarse grained sandstone are 4 GPa, 4 GPa, 7 GPa and 2 GPa, respectively. Estimated values of Poisson’s ratio for coal, fine grained sandstone, medium grained sandstone and coarse grained sandstone are 0.27, 0.41, 0.31 and 0.43, respectively. Bedding is the only joint found in all categories of sandstones. These joints are found tight, devoid of infillings, persistent with a joint spacing of 0.3 m to 0.7 m for both fine grained and medium grained sandstone while the joint spacing in coarse grained sandstone found to be 0.6 m to 0.9 m. Coal has two more joint sets apart from its cleats. The average spacing of cleats in coal is varying between 0.1 m to 0.15 m. Water seepage in the mine has been found below 20 ml/min.

Rock mass characterisation for coal and coal measure rocks of the mine has been done by using Geomechanics classification of Coal Measure Rocks (Coal Mine RMR by CMRI; CMRI, 1987), Bieniawski’s RMR (Bieniawski, 1976) and NIOSH’s Coal Mine Roof Rating CMRR (Molinda and Mark, 1993). These rock mass characterisation parameters are utilized for prediction of different geomechanical...
conditions for the proposed continuous miner operations in the mine. Ratings for CMRR approach of rock mass characterisation is summarised in Table 3.

Table 2 - Engineering properties of coal measure rocks

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Engineering Property</th>
<th>Mean Value of the Property</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal</td>
<td>Mass Density, kg/m$^3$</td>
<td>1290</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>First Cycle Slake Durability Index</td>
<td>97%</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
<td>4</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.27</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>UCS, MPa</td>
<td>28.5</td>
<td>Tested value</td>
</tr>
<tr>
<td>Fine Grained Sandstone/Shaly Sandstone</td>
<td>Mass Density, kg/m$^3$</td>
<td>2230</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
<td>4</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.41</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>First Cycle Slake Durability Index</td>
<td>93%</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>UCS, MPa</td>
<td>22.6</td>
<td>Tested value</td>
</tr>
<tr>
<td>Medium Grained Sandstone</td>
<td>Mass Density, kg/m$^3$</td>
<td>2230</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
<td>1</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.31</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>First Cycle Slake Durability Index</td>
<td>93%</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>UCS, MPa</td>
<td>16.3</td>
<td>Tested value</td>
</tr>
<tr>
<td>Coarse Grained Sandstone</td>
<td>Mass Density, kg/m$^3$</td>
<td>1780</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
<td>1</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.43</td>
<td>Estimated</td>
</tr>
<tr>
<td></td>
<td>First Cycle Slake Durability Index</td>
<td>83%</td>
<td>Tested value</td>
</tr>
<tr>
<td></td>
<td>UCS, MPa</td>
<td>9</td>
<td>Tested value</td>
</tr>
</tbody>
</table>

Table 3 - Estimated CMRR values for a coal mine where CM technology inducted

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Range</th>
<th>Value</th>
<th>Range</th>
<th>Value</th>
<th>Range</th>
<th>Value</th>
<th>Range</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stress, MPa</td>
<td>23.5MPa</td>
<td>15</td>
<td>22.5MPa</td>
<td>15</td>
<td>19.3MPa</td>
<td>10</td>
<td>9MPa</td>
<td>10</td>
</tr>
<tr>
<td>Layer thickness, cm</td>
<td>7 cm to 15 cm</td>
<td>27</td>
<td>0.5m</td>
<td>32</td>
<td>0.5m</td>
<td>32</td>
<td>0.7m</td>
<td>36</td>
</tr>
<tr>
<td>Discontinuity/Shear Strength Rating</td>
<td>Planar, rough, moderate to moderate coherent joints</td>
<td>25</td>
<td>Planar, tight joints with small laminae or shales</td>
<td>16</td>
<td>Planar, rough and tight shales/plates</td>
<td>55</td>
<td>Planar, rough and very loose/weak</td>
<td>16</td>
</tr>
<tr>
<td>Moisture saturation (SD%)</td>
<td>97%</td>
<td>0</td>
<td>92%</td>
<td>-3</td>
<td>32%</td>
<td>-3</td>
<td>60%</td>
<td>-10</td>
</tr>
<tr>
<td>Water surcharge adjustment</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
<td>0</td>
</tr>
</tbody>
</table>

CMRR 57 (Good) 60 (good) 74 (Good) 51 (Moderate)

Prediction of ground conditions using empirical approaches

Room width: Room width or gallery width for continuous miner operations are largely dependent on the cutting drum width of the continuous miner employed. It is generally twice the cutting width to facilitate ease in broken coal gathering and better economic returns. The mine management desired to introduce a continuous miner which has a 2.7 m wide cutting drum and requires at least 4.2 m wide galleries for making square pillar geometries. Economical operations of the machine demand a gallery width of 5.4 m (twice the cutting drum) so that two cuts can be achieved without changing the place of the machine. Junctions carry greater opening dimensions than the galleries in coal mines and hence carry greater risks of roof fall. Though coal junctions are always supported prior to their opening, it is imperative that a safe design should be based on safe junction geometry. Mark et al. (2001) proposes the following relationship to estimate maximum diagonal distance of a coal mine junction based on CMRR.
IsG = 20+0.26(CMRR) \hspace{1cm} (1)

Where IsG is the diagonal distance of a junction in feet.

Based on the above relationship, a safe diagonal distance of 12 m for a junction is possible. This safe distance makes board width as 8.3 m, which is greater than the practical requirement of 5.4 m. Bieniawski (1989) uses concept of stand-up time and unsupported span which can be used to design safe gallery width for continuous miner operations. Further, British Coal Board considers 48 hrs of stand-up time necessary for design of cut-out distance. It may be noted that the concept of unsupported span by Bieniawski is one dimensional parameter. It considers either gallery width or the cut-out-distance (in the present case) as an unsupported span. Nomogram by Bieniawski (1989) for stand-up time is given in given in Figure 1. A gallery width of about 5.4 m will certainly be safe as per the nomogram for RMR value of 65 and stand-up time of 48 h.

Cut-out distance: Globally, there are two terminologies applied for permissible unsupported span by a continuous miner. Australia and UK favours single terminology of cut-out distance while S. Africa defines extended-cut as a cut-out distance more than 12 m and in USA, extended-cut is defined as a cut-out distance more than 6 m for remote controlled continuous miners. It is pertinent to note that limitation imposed on the permissible extent of cut-out distance in various countries is largely based on human and ventilation factors rather than issues related with roof instability (Canbulat and van der Merwe, 2000). Technically, roof dilation/bed separation stops once the face moved beyond a distance twice of the bord width (Canbulat and van der Merwe, 2000; Mark, 2007). Empirically, two approaches, namely, Bieniawski’s RMR (1976) and CMRR by NIOSH (Mark, 1999) can be used to delineate cut-out distance. A cut-out distance of 18 m can be predicted for a bord width of 5.4 m and stand-up time of 48 h using the concept of unsupported span as shown in Figure 1. There is, however, a practical limitation on this cut-out distance. A cut-out distance should only be practiced when there is a minimal chance of the CM operator stepping into the unsupported area for identification of variations in roof conditions. Bauer (1998) proposed the following relationship for a safe cut-out distance during pre-approval stage of a mine based on NIOSH’s CMRR approach.

$$\text{Cut Depth} = 8.1 + 0.564 \times \text{(CMRR)} - 0.152 \times \text{(Bord Width)} - 0.0029 \times \text{(Overburden)} \hspace{1cm} (2)$$

Where bord width and overburden are in feet.

Using the above relationship, cut-out distance comes out to be 14 m for a bord width of 5.4 m for a CMRR value of 74 for the mine. Mark (1999) reports that 12 m extended cuts will always be stable for a CMRR value higher than 55. The above two calculations corresponds to US experience. One striking difference between US data and this particular case is that the US mines have more than one lithological unit within the strata to be rock bolted whereas the present case has only one unit of coal itself. Keeping intact more lithological units than one has been a more difficult task in underground coal mining (Karmis and Kane, 1984; Kester and Chugh, 1980). Based on these findings, it may be safe to predict safe operations of the continuous miner in the present case with a cut-out distance beyond 12 m. The limit on cut-out distance beyond the machine length should include considering ventilation factors (dust and gases generation and their impact on the health of CM operator and chance of explosion in the mine) and human factors (chances of CM operator to step into unsupported area for visualising variation in roof rock conditions).

Prediction of ground conditions using numerical modelling

Three dimensional numerical models were prepared to evaluate stability of roof rock under various conditions and also to make predictions for continuous miner operations in 5.4 m wide galleries. All lithological units with their respective rock mass properties were used for the modelling. Corresponding materials, as per the typical lithologs, have been considered to follow Mohr-Coulomb’s elasto-plastic rock failure model with non-associated flow rule. Various rock mass properties and corresponding rock properties are given in Table 4. The basis of conversion of the properties into rock to rock mass has been given by Sheorey (1997) and others (Bieniawski, 1978; Serafim and Pereira, 1983; Singh, 1979). The prepared models were provided with gravity loading only as initial load conditions for the reason that the mine is under a shallow depth cover of 30 m and there is no sign of distress due to in situ stresses. Model geometry prepared and used for the modelling is given in Figure 2. Model boundaries are truncated using the advantage of symmetric planes. Appropriate roller boundaries are placed at the far field model boundaries. Two categories of models are prepared. The models with a 4.2 m wide gallery...
were prepared for validation of the modelling while the models with a 5.4 m wide gallery were prepared for prediction of roof conditions during continuous miners operations. All simulations have been solved following two stages. The staged excavation of mining steps were incorporated in the modelling after gravitational load condition is imposed and solved in an initial load condition. Staged excavation with 1 m mining steps covering 12 mining steps were introduced during the simulations. Models behaviour was evaluated after each simulation through observations of roof rock deformations, material failure state and safety factor contours. Numerical modelling results are compared with observed deformation values at 28 L/2D of the mine by multi-point borehole extensometers (MPBX). Corresponding predicted deformation values through the modelling and the observed deformation values are compared and shown in Figure 3. Comparison of the deformation values shows a correlation coefficient of 86% with the slope of the trend line as 25.64. The high correlation coefficient indicates that the prepared numerical models are accurate enough to provide reasonable trends for the mining conditions.

### Table 4 - Rock mass properties used for Mohr-Coulomb material

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Engineering Property</th>
<th>Property of the rock</th>
<th>Property of the rock mass</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal (RMR=56)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>UCS, MPa</td>
<td>28.5</td>
<td>3.16</td>
</tr>
<tr>
<td></td>
<td>Tensile Strength, MPa</td>
<td>3</td>
<td>0.6</td>
</tr>
<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
<td>7</td>
<td>1.4</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.27</td>
<td>0.27</td>
</tr>
<tr>
<td></td>
<td>Cohesion, MPa</td>
<td>-</td>
<td>0.72</td>
</tr>
<tr>
<td></td>
<td>Friction, Degree</td>
<td>-</td>
<td>41.5°</td>
</tr>
<tr>
<td>Fine Grained Sandstone/Shaly Sandstone (RMR=47)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>UCS, MPa</td>
<td>22.59</td>
<td>1.6</td>
</tr>
<tr>
<td></td>
<td>Tensile Strength, MPa</td>
<td>3</td>
<td>0.44</td>
</tr>
<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
<td>4</td>
<td>0.6</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.41</td>
<td>0.41</td>
</tr>
<tr>
<td></td>
<td>Cohesion, MPa</td>
<td>-</td>
<td>0.69</td>
</tr>
<tr>
<td></td>
<td>Friction, Degree</td>
<td>-</td>
<td>35°</td>
</tr>
<tr>
<td>Medium Grained Sandstone (RMR=65)</td>
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<td></td>
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<tr>
<td></td>
<td>UCS, MPa</td>
<td>16.33</td>
<td>2.84</td>
</tr>
<tr>
<td></td>
<td>Tensile Strength, MPa</td>
<td>2</td>
<td>0.55</td>
</tr>
<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
<td>7</td>
<td>1.96</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.31</td>
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<tr>
<td></td>
<td>Cohesion, MPa</td>
<td>-</td>
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</tr>
<tr>
<td></td>
<td>Friction, Degree</td>
<td>-</td>
<td>41°</td>
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<tr>
<td>Coarse Grained Sandstone (RMR=41)</td>
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<td>UCS, MPa</td>
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<td>Tensile Strength, MPa</td>
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<tr>
<td></td>
<td>Young’s Modulus, GPa</td>
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<td>0.3</td>
</tr>
<tr>
<td></td>
<td>Poisson’s Ratio</td>
<td>0.43</td>
<td>0.43</td>
</tr>
<tr>
<td></td>
<td>Cohesion, MPa</td>
<td>-</td>
<td>0.7</td>
</tr>
<tr>
<td></td>
<td>Friction, Degree</td>
<td>-</td>
<td>37.9°</td>
</tr>
</tbody>
</table>

Material failure state plot (Figure 4) and safety factor contours (Figure 5) are evaluated to make predictions for the roof behaviour during the continuous miner operations under 5.4 m wide galleries. The minimum safety factor contour value is of 1.97 at the face while roof level has the safety factor value more than ten. Evaluation of the modelling results for change in material state conditions (failure plots) did not reveal any material change in conditions for the mine from the gallery width widening from 4.2 m to 5.4 m even after 12 m of staged excavation simulation steps. Further, comparisons of deformation values and support pressure values between corresponding excavation stages of 4.2 m and 5.4 m wide galleries indicate that there will be 26% increase in deformation values and no change in support pressure values. An increase of 26% in deformation means that the deformation values will remain less than 1 mm for 5.4 m wide galleries. This miniscule change in the deformations will not result into any change in support pressure. These observations, like the empirical predictions, predict that the 5.4 m wide galleries with a cut-out distance selection based on human factor and ventilation factor will be safe for the mine.
Figure 2 - Basic numerical model with far field boundary conditions

Figure 3 - Training of numerical simulations

Figure 4 - Material failure state in coal seam for 5.4 m wide gallery
CONCLUSIONS

Concepts of designing room and pillar mining with continuous miner technology with respect to geomechanics issues are explained and a case study presented in this publication. The continuous miner technology is a viable technology to boost production and replace work force intensive technology of the drill and blast cycle for room and pillar coal mining method. The CM technology needs a proper assessment of geominning conditions prior to introduction of a particular type of continuous miner in the mine. It is experienced that one of the Indian mines introduced a continuous miner under adverse mining conditions and the machine is under-performing. Two of the mines in India where CMs were introduced after a proper study for design of room width and cut-out distance are operating without any geotechnical issues from the last five years. It may be, however, noted that both the mines were designed with concentration of rooms only which shall not be a part of the design. Pillars shall be smaller and rectangular for the CM technology in confirmation of the need for the final mining operations.

ACKNOWLEDGMENTS

The authors sincerely appreciate the respective affiliated institutes for permission to present the case study and views. The views expressed in this paper are of the authors and shall not be construed as the views of the respective institute to which they belong.

REFERENCES


RELATIONSHIP BETWEEN TWIN TUNNELS DISTANCE AND SURFACE SUBSIDENCE IN SOFT GROUND OF TABRIZ METRO - IRAN

Saied Mohammad Farouq Hossaini\textsuperscript{1}, Mehri Shaban\textsuperscript{2} and Alireza Talebinejad\textsuperscript{3}

\textbf{ABSTRACT:} This paper presents a series of three-dimensional finite distinct element analyses carried out for line 1 of Tabriz metro tunnels. Interaction between these circular parallel twin tunnels excavated by an Earth Pressure Balance machine in a soft ground has been studied. The Influence of the distance between twin tunnels on the surface subsidence, bending moment and axial forces in the segmental lining of the first tunnel have been particularly, investigated. Advancing of the second tunnel affects the surface subsidence, bending moment and internal forces in the lining of the first tunnel. These effects relate directly to the width of the pillar separating the twin tunnels. It was found that the location of the maximum subsidence is offset from the centreline of the first tunnel. The offset increases with decrease in the distance between the tunnels. Also, moment and axial forces of the first tunnel decrease by increasing the space between the tunnels. The interaction between the tunnels has been quantified and classified in accordance with various tunnel distances.

\textbf{INTRODUCTION}

The use of underground spaces for transport infrastructures is required in development of large cities. In some cities, the geotechnical and underground conditions require the construction of new tunnels close to existing ones. In other cases, the solution of twin tunnels presents major advantages, such as reduction of both tunnels diameter and soil movement resulting from the tunnel construction (Chen, \textit{et al}., 2009).

Ground movements are an inevitable consequence of excavating and constructing tunnels. Tunnel excavation causes relaxation of \textit{in situ} stress, which is only partially restricted by the insertion of the tunnel support. In fact it is not possible to create a void instantaneously and provide an infinitely stiff lining to fill it exactly. Hence, a certain amount of the deformation of the ground will take place at the tunnel depth; this will trigger a chain of movements, resulting in settlements at the ground surface, which are more significant at shallow tunnel depth (Moller and Vermeer, 2008).

Numerical modelling and \textit{in situ} observations were used to analyse the interaction between twin tunnels. Results show that in some configurations, the interaction could largely affect the soil settlement and that the design of twin tunnels requires numerical analysis associated to monitoring during the design phase (Hage Chehade and Shahrour, 2008). The construction of the first tunnel may notably affect the soil conditions: reduced confinement, stress release and reduction of the strength parameters of the soils. Consequently, the second tunnel will be excavated through a different material and the induced settlements related to the second tunnel will be generally greater (Guglielmetti, \textit{et al}., 2007).

This paper presents 3D numerical analysis conducted to investigate the influence of twin tunnel spacing on the surface settlement and internal forces resulting from the tunnel excavation. Analysis was carried out for three different tunnel distances namely 0.5D, 1D and 1.5D where D is the tunnel's diameter.

\textbf{TABRIZ METRO LINE 1}

Tabriz is a large city in the north west of Iran with a population of about two million. Tabriz metro is designed in three lines. Line 1 starts at south east of the city and after passing the city centre ends at the south west. In this line two parallel circular tunnels are excavated by two EPB machines. The length of tunnels is 8 km and their diameter is 6.88 m. Figure 1 shows three lines of Tabriz metro.

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Geotechnical condition is divided into four different sections in the tunnel alignment. Analysis was carried out for sections 1 and 2 of line 1 because of the critical conditions of these parts regarding geotechnical specification, water table depth and existence of old buildings. There is one sedimentary layer in section 1 and four different layers in section 2 including two silt layers with different properties, a sand layer and a man filled layer. The properties of different layers are summarized in Table 1.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Section 1</th>
<th>Section 2</th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Fine sedimentary layer</td>
<td>Manfilled</td>
<td>silt</td>
<td>sand</td>
<td>silt</td>
<td></td>
</tr>
<tr>
<td>Friction angle (degree)</td>
<td>35</td>
<td>5</td>
<td>15</td>
<td>33</td>
<td>34</td>
<td></td>
</tr>
<tr>
<td>Cohesion (MPa)</td>
<td>0</td>
<td>0</td>
<td>0.015</td>
<td>0</td>
<td>0.025</td>
<td></td>
</tr>
<tr>
<td>Bulk modulus (MPa)</td>
<td>33.34</td>
<td>15.62</td>
<td>22.3</td>
<td>33.34</td>
<td>33.34</td>
<td></td>
</tr>
<tr>
<td>Shear modulus (MPa)</td>
<td>11.12</td>
<td>5.6</td>
<td>7.4</td>
<td>15.4</td>
<td>11.2</td>
<td></td>
</tr>
<tr>
<td>Thickness (m)</td>
<td>-</td>
<td>5</td>
<td>8</td>
<td>15</td>
<td>20</td>
<td></td>
</tr>
<tr>
<td>Tunnel depth (m)</td>
<td>10</td>
<td>14</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Water depth (m)</td>
<td>20</td>
<td>6</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

The depth of tunnels from the ground surface is 10 m and 14 m in sections 1 and 2, respectively. In section 1 ground water table is 8 m above the tunnels crown but it is below the tunnel invert in section 2 (Jahad-e Tahghighate Sahand, 2005).

THREE DIMENSIONAL MODELS

Figure 2 shows the model used for analysis of horizontally aligned tunnels with 6.88 m diameter and 7 m pillar width. The boundary of the model is extended to a distance where there is no effect of tunnel construction on the lateral border of the model. This distance is 5.5D equal to 38 m from each tunnel's centre.

The length, width and height of the model are 56 m, 90 m and 40 m, respectively. Layers of section 2 and their properties are also shown in Figure 2. The model contains 57,280 elements and 60,147 nodes. The mesh size increases gradually when the distance from the tunnel increases. The mesh size in the...
direction of the tunnel axis is equal to the segment length which is 1.4 m. Concerning the boundary conditions, the displacements are constrained in three directions at the bottom, while zero horizontal displacement is imposed at lateral boundaries.

![Image of mesh used for modelling](image)

**Figure 2 - The mesh used for modelling**

The ratio of the horizontal to vertical stress (k) is 0.43 and 0.74, respectively for sections 1 and 2. Initial vertical stress is calculated as the weight of the layers. The stress induced by surface structures and traffic is applied as planner load on the top boundary of the model. This stress is assumed to be 24 000 N/m². The soil behaviour is described by an elastic perfectly plastic Mohr-Coulomb criterion.

Modelling of the twin tunnels construction is carried out in the following steps:

- Construction of the first tunnel 1.4 m, applying face pressure and installation of shield elements. This cycle is repeated till 9.8 m of the tunnel is excavated (7 cycles). There would be no lining installed up to this length.
- Continuing step one followed by installing lining elements and then injection of grout behind the shield.
- Repetition of step two till 35 m of tunnel 1 is excavated
- Starting the construction of the second tunnel in the same way performed for the first one.

The shield is modelled as a rigid cylinder by means of shell elements with external diameter of 6.86 m and length of 9.8 m. Segmental lining is modelled with shell elements with thickness of 0.3 m and internal diameter of 6 m. The behaviour of the shield and segmental lining is assumed to be linear-elastic. Properties of the shield and lining are shown in Table 2.

**Table 2 - Properties of shield and segmental lining**

<table>
<thead>
<tr>
<th>Type of support system</th>
<th>Elastic modulus (Gpa)</th>
<th>Poisson’s ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shield</td>
<td>200</td>
<td>0.25</td>
</tr>
<tr>
<td>Segmental lining</td>
<td>25.2</td>
<td>0.2</td>
</tr>
</tbody>
</table>

**Results**

Analysis was conducted for three various pillar width of 3.5 m (0.5D), 7 m (D) and 10.5 m (1.5D). Figure 3 shows the surface settlement for different tunnel distances in section 1. The pattern and magnitude of the settlement depend on the distance between tunnels. Maximum settlement is about 2.65 cm, 2 cm and 1.9 cm for the pillar width of 3.5 m, 7 m and 10.5 m, respectively. By increasing pillar width from 3.5 m to 7 m, maximum settlement decreases about 25% but it decreases about 5% when pillar width
increases from 7 m to 10.5 m. The maximum soil settlement is observed for tunnels with the narrowest pillar width (i.e. 3.5 m) where maximum settlement occurs in the centre part of the pillar. When distance between the tunnels increases the settlement in this part decreases because of decreasing the interaction between tunnels.

![Figure 3 - Surface settlement for different pillar width between tunnels in section 1](image)

Interaction between tunnels leads to increase in soil movement in the pillar between them. As Figure 4 shows in pillar width of less than 10.5 m maximum settlement offsets from the centre line of the first tunnel. The magnitude of offset increases when distance between tunnels decreases. In pillar width of 3.5 m maximum settlement occurs at 5.19 m far from the first tunnel axis.

![Figure 4 - Offset of maximum settlement from the center line of the first tunnel in section 1](image)

Figure 5 shows the settlement curves in section 2. In this section, for pillar width of 3.5 m, maximum settlement doesn’t occur in the central part of the pillar but is near to the first tunnel (i.e. 4.44 m from centre of the first tunnel). For the pillar width of 7 m maximum settlement is in the pillar zone with 0.75 m offset from pillar centre, but for 10.5 m pillar width maximum settlement is 3 m from the centre part of pillar (2.25m from the first tunnel wall). The shape of curve for this case shows that the interaction between tunnels decreases. Maximum settlement is 3.9, 3.5 and 2.75 cm for 3.5, 7 and 10.5 m pillar width, respectively. Therefore, increasing pillar width from 3.5 m to 7 m leads to decrease of 11% and increasing pillar width from 7 m to 10.5 m leads to another decrease of 27% in the ground settlement.

The maximum settlements estimated for pillar width of 3.5 m and 7 m are not allowable in urban area. Therefore, the pillar width of 10.5 m is suitable as it decreases the settlement to an allowable amount in section 2.

Excavation of the second tunnel changes the bending moment and axial forces in the segmental lining of the first tunnel. Tables 3 and 4 show the quantities of these parameters for various tunnels distances. As shown in these Tables, for lowest distance, the bending moment increases 17% and 11.5% and axial force increases about 6.5% and 9.5% in sections 1 and 2, respectively. Bending moment and axial forces in the lining of the first tunnel decrease when the distance between tunnels increases. In both
sections the effect of advancing of the second tunnel on the lining of the first tunnel is negligible for 10.5 m pillar width.

![Figure 5 - Surface settlement for different pillar width between tunnels in section 2](image)

### Table 3 - Bending moment on the lining of the first tunnel after excavating the second one

| Pillar width (m) | Section 1 | | Section 2 | |
|------------------|-----------|------------------|-----------|
|                  | Moment in segments (KN-m) | Changes compare to single tunnel (%) | Moment in segments (KN-m) | Changes compare to single tunnel (%) |
| 3.5              | 0.884208 | 17               | 2.13057 | 11.5 |
| 7                | 0.831688 | 10               | 1.99667 | 6.5  |
| 10.5             | 0.781443 | 3.5              | 1.94025 | 1.5  |

### Table 4 - Axial forces in the first tunnel lining after excavating the second one

| Pillar width (m) | Section 1 | | Section 2 | |
|------------------|-----------|------------------|-----------|
|                  | Magnitude of forces (KN) | Changes compare to single tunnel (%) | Magnitude of forces (KN) | Changes compare to single tunnel (%) |
| 3.5              | 664.94 | 6.5               | 1222.2 | 9.5    |
| 7                | 644.45 | 3                  | 1152.6 | 3.5    |
| 10.5             | 625.84 | 0.06              | 1127.4 | 1.1    |

### CONCLUSIONS

The following conclusions can be drawn from this numerical study:

- Settlement decreases with increasing the distance between the tunnels for both sections. In section 1, the magnitude of settlement is 2.65, 2 and 1.9 cm for pillar width of 3.5, 7 and 10.5 m, respectively. In section 2, settlement changes from 3.9 to 2.65 cm when pillar width change from 3.5m to 10.5 m;

- Offset of maximum settlement from the centre line of the first tunnel decreases by increasing the tunnels’ distance. It is 5.19m and 4.44 m for section 1 and section 2, when the pillar width is 3.5m. The offset reaches zero for pillar width of 10.5 m;

- The moment in the segmental lining of the first tunnel decreases by increasing the distance between the tunnels. In section 1, it increases 17% for pillar width of 3.5 m while it increases 3.5% for pillar width of 10.5 m. These amounts are 11.5 and 3.5% for section 2;

- The axial force decrease by increasing of tunnels distance. It changes 6.5% and 9.5% for section 1 and section 2 when the pillar width is 3.5 m. These changes are 0.06% and 1.1% for pillar width of 10.5 m.
In section 1, interaction between tunnels is negligible for pillar width of 7 m (D) and further.

In section 2, interaction between two tunnels is negligible for pillar width of 10.5 m (1.5D) and further.

REFERENCES


PHYSICAL SCALE MODELLING OF ROADWAY ADVANCE WITH WEDGE CUT BLASTING FOR SOUTH WING RAIL AT TUNLIU MINE

Y.Q. Yu\(^1\) and H.S. Mitri\(^2\)

ABSTRACT: Blasting with wedge cut is the key to the efficiency of roadway excavation in coal mines, particularly where the rock face is highly laminated. The quality of the wedge cuts can directly affect blasting results. A series of model experiments were carried out in the Blasting Laboratory of Henan Polytechnic University, China, in order to improve the quality of wedge cuts and increase roadway development rates in the South Wing rail of Tunliu mine. Three groups of twelve wedge cut model experiments were conducted, which showed that the wedge cutting boreholes angle can affect the depth and volume of blasted zone. It was found that; symmetrical cut pattern was beneficial, and effective stemming of the blasting holes can enhance explosive energy efficiency and ensure effective blasting. Suitable roadway wedge cut blasting parameters were determined through the analysis of test results.

INTRODUCTION

Blasting with a wedge cut is the key to the efficiency of roadway excavation in coal mines. The quality of wedge cut can directly affect the blasting results. The depth of the wedge has a direct impact on the length of the face advance. The quality of the wedge cut is related to the geological conditions of the host rock and the type and quantity of explosives and the detonation sequence (Xiaolin and Yu, 2009; Huateng, 1984; Shouzhong, 1988). Wedge cut blasting design model experiments were conducted in the Blasting Laboratory of Henan Polytechnic University in order to improve the quality of blasting with wedge cut and increase roadway development rate in South Wing rail of Tunliu mine. According to the physical and mechanical parameters of rock South Wing rail of Tunliu mine, three groups of tests, comprising a total of twelve wedge cut blasting model experiments were conducted to determine suitable roadway cutting blasting parameters.

BLASTING WITH WEDGE CUTS

Various drilling patterns have been developed for blasting solid rock faces for development mine drifts, ramps and roadways. One of these patterns is the V type wedge cut. In this pattern, the blast holes are drilled at an angle to the face forming a uniform wedge in the middle of the rock face. Upon blasting, the wedge cut is effectively ejected from the rock face, and the wedge is further widened to the full width of the drift in subsequent blasts, each blast being fired with detonators of suitable delay time. This type of cut is particularly suited to coal mines, where the rock is well laminated (Gabshen, 2009; Zhongjie, 2010). In effect, the ejection of the wedge creates a new free surface that helps produce better quality fragmentation when the rest of the blast holes are fired. The number of blast holes needed to form a wedge is usually two to four pairs, and they should be symmetrically drilled with respect to the rock face; see Figure 1. The spacing between the holes is 0.2 to 0.3 m at the wedge bottom. The wedge strike can be either vertical (Figure 1a) or horizontal (Figure 1b). The wedge apex angle is 60\(^\circ\) to 30\(^\circ\), which makes the angle of the drill holes with the rock surface in the range of 60\(^\circ\) to 75\(^\circ\) respectively. When the rock is particularly strong and brittle and the required depth is more than 2 m deep, the wedge pattern can be repeated as shown in Figure 1c to form a double wedge (Yu, 2009).

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\(^2\) Department of Mining and Materials Engineering, McGill University, Montreal Canada
MODEL EXPERIMENTS

Objectives

The main goals of the experimental study were:

- Investigate the effect of wedge apex angle or the inclination angle of the blast holes with respect to the rock face;
- Determine the most suitable wedge cut pattern;
- Verify the role of blast-hole block;
- Verify the rock blasting mechanism.

Experimental design

According to the rock physical and mechanical properties of South Wing rail of Tunliu min, the roadway rock strength was divided into three groups at 38.5 MPa, 50 MPa and 25 MPa. The 38.5 MPa strength group accounted for 80% of the roadway rock. According to the criteria of “similarity”, as specified by the Chinese Academy of Science (1986), the physical scale and model dimensions were 1:20 and 300×300×300 mm (actual engineering 6000×6000×6000 mm) respectively. The models were made from a mixture of water, cement and sand. Three materials were prepared with the following proportions of water: cement: sand - 1:1.9:8.4, 1:2.3:7.2 and 1:2.5:8.0. The cement strength grade used for the models was 32.5 (R) ordinary Portland cement. Testing was carried out in the Blasting Laboratory of Henan Polytechnique University, China. Twelve physical scale models were made. All models were given a curing period of 28 days. Nine models were used to examine the effect of angle of inclination of blast holes with respect to the model face (Table 1). The number of blocks tested which were used to verify the role of blast-hole block is three. The split surface of each model tested was observed.

Table 1 - Experimentation design of physical scale models

<table>
<thead>
<tr>
<th>Number</th>
<th>Strength (MPa)</th>
<th>Depth of hole (mm)</th>
<th>Hole inclination (°)</th>
<th>Dose per hole (g)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>38.5</td>
<td>70</td>
<td>70 70 70 70</td>
<td>0.1</td>
</tr>
<tr>
<td>2</td>
<td>38.5</td>
<td>75</td>
<td>75 75 75 75</td>
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</tr>
<tr>
<td>3</td>
<td>38.5</td>
<td>80</td>
<td>80 80 80 80</td>
<td>0.1</td>
</tr>
<tr>
<td>4</td>
<td>38.5</td>
<td>70</td>
<td>70 70 80 80</td>
<td>0.1</td>
</tr>
<tr>
<td>5</td>
<td>38.5</td>
<td>80</td>
<td>80 80 80 80</td>
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</tr>
<tr>
<td>6</td>
<td>38.5</td>
<td>70</td>
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<td>0.1</td>
</tr>
<tr>
<td>7</td>
<td>38.5</td>
<td>80</td>
<td>80 80 80 80</td>
<td>0.1</td>
</tr>
<tr>
<td>8</td>
<td>38.5</td>
<td>70</td>
<td>70 70 70 70</td>
<td>0.1</td>
</tr>
<tr>
<td>9</td>
<td>38.5</td>
<td>80</td>
<td>80 80 80 80</td>
<td>0.1</td>
</tr>
</tbody>
</table>
Experimental methods

The holes were made with an electric drill (diameter of 5 mm). A protractor and ruler were used to ensure the accuracy of the hole angle, the type of explosive matching the material should used was in accordance to the impedance principle. Taking into account the smaller size of the model dimensions and drill hole size, and in order to more easily observe blasting effect, a high detonation velocity elemental RDX explosive was used. Each hole charge was 0.1 g, determined according to the model of rock mass strength calculation, and taking into account misfire factors. The detonation means of model was by a gunpowder head. The major risk factors for the model blasting experiment were based on noise, blast shock wave and flying rocks. Because of the small amount of explosive, the noise and blast shock wave were at a safe range for the laboratory use. In order to protect laboratory personnel and facilities from flying rocks, a rubber cover was used. The cutting depth was measured after each blast. The volume of cut cavity was determined by placing a plastic bag in the cavity, and then pouring water into it. The volume of water needed to fill the cavity was recorded. After model blasting, the filling effect of blast-hole and the split plane of fracture and damage model were directly observed.

![Figure 3 - Physical scale model](image)

![Figure 4 - Blast-hole pattern](image)

![Figure 5 - Measuring wedge cut cavity volume](image)

![Figure 6 – Measuring the cavity depth](image)

Experimental results

Experimental results parameters include wedge cut cavity depth, volume and blast-hole utilisation and fragmentation. The recorded results are shown in Table 2. Figure 7 shows the groove cavity depth and volume comparison chart (model strength 38.5 MPa). Figure 8 is groove cavity depth comparison for the inclination angles of 70° and 80° for three different strength models. Figure 9 to Figure 13 present the model blasting effect diagrams. Figure 14 presents a comparison between stemming and no stemming of blast holes. Figure 15 shows the split surface of the blasted model.

### Table 2 - Experimental results

<table>
<thead>
<tr>
<th>Model</th>
<th>Groove depth (mm)</th>
<th>Groove cavity volume (ml)</th>
<th>Hole utilization (%)</th>
<th>Degree of stemming</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>61</td>
<td>150</td>
<td>92.8</td>
<td>1</td>
<td>a wide range of fragments around the eight models groove cavity.</td>
</tr>
<tr>
<td>2</td>
<td>65</td>
<td>190</td>
<td>96</td>
<td>1</td>
<td></td>
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<tr>
<td>9</td>
<td>63</td>
<td>190</td>
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</tbody>
</table>
Figure 7- Groove cavity depths and volume comparison chart of model (38.5 MPa)

Figure 8 - Model groove cavity depth for different hole inclinations and material strength

(a)  
(b)  
(c)  

(d)  
(e)  
(f)  

Figure 9 - Fragmentation pattern (38.5 MPa)

Figure 10 - Fragmentation pattern (50 MPa)  
Figure 11 - Fragmentation pattern (25 MPa)
MODEL EXPERIMENTAL FINDINGS

The effect of inclination angle on groove cavity depth and volume was shown in Figure 7. As can be seen, Model 3 (38.5 MPa) with an inclination angle of 80° shows the maximum cavity volume of 220 ml. Meanwhile, as can be seen from the groove cavity volume results reported in Table 2, the asymmetric inclination of the wedge cutting angle (Models 4 and 5) are less efficient than symmetric models with groove cavity volumes of 140 ml and 150 ml for Models 4 and 5 respectively. Figure 8 shows that for the same cutting angle, the groove cavity depth of varies for different strength models. However, as the cutting angle increases from 70° to 80°, the groove depth has increased. The fragmentation patterns of selected samples are shown in Figures 9 to 13 for the three rock types tested. Figure 14 demonstrates the difference between the fragmentation pattern of a block with stemmed blast-hole (Figure 14a), and another with no stemming (Figure 14b). Clearly, fragmentation is much more effective with stemming. Figure 15 shows that there is a smaller range of crush area around the blast holes and some traces caused by the detonation gas splitting into the surrounding area.

CONCLUSIONS

In the light of the results obtained from this experimental study a number of conclusions can be drawn. It appears that the best cutting angle for maximum cavity volume is 80° for the 38.5 MPa rock. The angle of inclination affects the cutting depth and volume; the ideal angle is 80° for the 38.5 MPa rock group, 70° for the 50 MPa rock group, and 85° for the 25 MPa rock. The tests show that asymmetric wedge cut produces shorter groove depth and less cavity volume, and therefore symmetrical cutting must always be used in field applications. In order to enhance the explosive energy efficiency, the blast-holes need to be well stemmed. Detonation gases will not work effectively if the holes are not stemmed to provide full fragmentation potential of the stress waves and detonation gases. Finally, it was found that there was a small crushed area around the blast holes and some shattering caused by the blast detonation.
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PREVENTION OF FRICTIONAL IGNITION IN COAL MINES USING CHILLED WATER SPRAYS

B Belle¹, D Carey² and B Robertson³

ABSTRACT: A key objective of this paper is to share application of chilled water sprays for prevention of frictional ignitions in coal mines to provide a safe occupational environment. Frictional ignitions in underground mines if not managed adequately may lead to major explosions. Historic statistics have indicated that the greatest explosion risk originates from frictional ignitions. Methane and dust related to explosions have resulted in over 7500 lives lost in the last ten years worldwide. The experiences of chilled water sprays used for managing heat in deep metal mines in South Africa is highly relevant to frictional incendive/face heat management in Australian coal mines.

Mine ventilation and water sprays (used for dust suppression and dilution ventilation) are established technologies that are widely used in coal mines for the prevention of frictional ignitions. However, greater benefits of cooling or the sharp reduction in incendive heat from cutting picks by the use of chilled water sprays outweighs cooling using the current practice of using warm service water. Chilled water spray droplets have the potential to become ‘improved last line of defense’ against gas ignitions. For example, introduction of chilled water sprays with millions of fine chilled water droplets around picks and face area would provide a simple, reliable and rapid cooling power of 210 kW and 1 005 kW for continuous miners faces and Longwall shearer face respectively. A US study has noted that 90% of all frictional ignitions occurred in coal mines that liberated at least 0.39% of CH₄ through their mine ventilation air methane system. However, analyses of South African statistics indicate that frictional ignition have occurred in coal and gold mines that liberated even with lower emissions between 0.02% to 0.05% of CH₄ through their mine ventilation air methane system. Therefore, it is important to raise awareness that frictional ignition risks are ever present in coal mines regardless of gas contents or gas emissions. The application of wet head systems, proactive ventilation and methane monitoring, active suppression systems, and frequent safety interactions for preventing frictional ignitions are discussed.

INTRODUCTION

Frictional ignitions (FI) in mines are an operational safety threat that may lead to major explosions. A FI is an ignition of flammable gases initiated by a heat energy source resulting from frictional means. It may involve friction between different media, but commonly involves steel cutting elements rubbing against rocks with incendive (heat generating) properties such as high quartz or pyritic content. It is believed that the semi-molten trail of rock and pick metal left behind the tool (not the shower of orange sparks usually associated with rock cutting) is the primary source of heat that is igniting methane.

Knowledge of FI has been known for at least past 330 years. A FI review by Phillips (1997) noted that in July 1675, a Mr. Jessop of Broomhall, Yorkshire communicated the problem of FI with the Royal Society. In South Africa, the first FI was recorded in 1968. In addition, during the 1970’s seven FIs were recorded and 31 were recorded in 1980’s. In the last ten years, 10 FIs were recorded. A valuable FI research was conducted in Australia through ACARP project C7029 (2000). In the last two years, there have been reported incidents of FIs in Australian mines.

FIs are mostly associated as a pre-cursor to the deadly methane and coal dust explosions. Statistics have indicated that the greatest explosion risk comes from FIs, and that the most likely site for an ignition to become a major incident is in a development drivage (Browning and Warwick, 1993). However, recent statistics have highlighted that longwall faces are not immune to FI risks.

Possible energy sources for gas ignitions are lightning strike, spontaneous combustion, electrical ignition, mechanical heat sources, contraband, and frictional heat. FI controls exist in the form of mine safety systems and procedures and in particular environmental monitoring, ventilation systems, trigger action response plans.

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³ Carabella Resources Limited
A study on changing pattern of ignition source (Figure 1) due to changes from conventional drill and blast to mechanized mining have been carried out (Phillips, 1996). There was a recorded case of gas ignition leading to flames that may have been caused by short circuiting or incomplete detonation in a blasting face in Ermelo mines in South Africa (Stone, 1990). As seen from the statistics, other than the lightning as the key contributor in gas ignition, all other major factors are not contradictory or debatable. It was noted that an increased mechanisation has also increased FI incidents. There is known evidences of lightning in the explosion of methane gases. However, the debate surrounding the role played by lightning in the gas ignition may not subside soon, but adequate lightning protection control procedures will aid in reducing the FI related incidents.

Figure 1 - Sources of FI in South African Collieries for the period between 1984 and 1993 (Phillips, 1996)

Another interesting statistics that is essential in understanding the ignition risks and control status is the impact of seasonal variation such as winter months (Figure 2). Unfortunately, no statistics could be found on Y-axis.

Figure 2 - Number of methane ignitions and season of the year (South African Experience)

FI’s are of regular nature and not unique to developed or developing countries, low or high inherent methane coal seams, low or high Ventilation Air Methane (VAM) from shafts. The seriousness of the methane ignition and the resulting methane and coal dust explosions are witnessed by the series of explosions in the coal mines worldwide (China, South Africa, Ukraine, USA, and New Zealand). Figure 3 shows the explosion statistics collated from the public domain. In South Africa, there are known cases of ignition of flammable gases from gold, platinum and coal mines leading to 78 accidents in the past seventeen years (1988 to 2005) with loss of multiple lives and injuries.

Some of the recognisable pattern of factors contributing to such events can be summarized as follows:

- Inadequate understanding of the inherent ignition and explosion risks;
- Inadequate and at risk installation, quality assurance and maintenance practices of monitoring hazardous environment using early warning devices;
• Inadequate management of ignition controls;
• Inadequate management and control of ignition sources;
• Inadequate integration of explosion risks into the overall mine design, control and systems;
• At-risk behavior, poor or non-implementation of risk management standards and procedures.

![Image of Figure 3 - Statistics on global mine explosions]

**CONDITIONS FOR FRICTIONAL IGNITION**

Prediction of transient methane explosive mixtures present in a mine atmosphere in and around coal cutting environment or face area is complex and thus prevention of its possible contact with frictional/incendive heat energy sources is a means of management to prevent FI in mines. Presence of methane or scenarios leading to FI or explosions is often termed as 'abnormal' situations. It is the adequate management of these ‘abnormal’ situations that will aid the mining industry in prevention of ignitions and loss of lives.

The literature review on FI highlights a number of pertinent facts (Robertson, 2010):

• Whilst the explosive range of methane in air is approximately 5-15%, the maximum sensitivity exists for 6.5-7.5% mixtures. The minimum ignition temperature is thought to be around 650 to 750 °C, but this increases to over 1 000 °C as the size of igniter reduces. The time to initiate an explosion is around 0.3 sec for 9.6% methane composition, increasing to 8 seconds at 12 to 14% mixtures. These facts suggest that for an incendive spark to ignite a gas mixture, the concentration would need to be optimal, the temperature quite elevated (approaching 2 000 °C) and the contact time of several seconds.

• Above dynamics support the concept of “hot spots” as the source of ignition. The area of ‘hot spots’ as the source of ignition, is a function of the nature of the streak developed behind the tool; wide streaks result from blunt tools and hot streaks develop when fused quartz from the rock adheres to steel on the tool and then rubs on the rock surface. Rocks with greater than 50% quartz content are highly incendive, especially where the quartz grain size is greater than 70 micron. Tests have indicated that even under these conditions, temperatures in excess of 1 000 °C are needed to cause an ignition.

• Rocks can be categorised by allocating a so-called Ignition Potential Categorisation (IGCAT) rating in accordance with physical and mineralogical properties. This solution is impractical considering the wide and complex geological conditions.

• Pyritic inclusions in the rock exacerbate incendiency due to raised temperatures during oxidation of pyrite when cut.

• Worn picks dissipate more energy in friction than sharp picks increasing the chances of an ignition occurring.

• Pick back clearance angles of greater than 12° are necessary to avoid the likelihood of steel shank material contacting rock.
Steel rubbing on such rocks creates the necessary frictional heat whereas hard tool coatings such as tungsten carbide, ceramics or diamonds produce less heat. A frozen (non-rotating) conical pick can cause an ignition with tungsten carbide contact if a large enough flat is worn.

There is variable evidence as to the importance of force and cutting speed but current thinking indicates that pick speeds below 1.5 m/s are less likely to create FI conditions, with lower forces also less conducive.

The probability of the appropriate juxtaposition of both methane condition and cutting condition to cause a FI is enhanced by the myriad of pick strikes developed from rotation of a laced drum in different face configurations.

**Common FI causes**

Globally, known causes of FI are not new and include the following absent and failed defenses:

- Blunt and worn out picks on coal cutting machine head;
- Missing Pick blocks;
- Cutting a large quantity of roof rock;
- Hard pyritic stone in the roof;
- Failure to identify abnormal geological conditions;
- Failure of picks to rotate and design of pick angle on cutting drums;
- Current pick speed may be too fast for mining conditions;
- Missing, blocked and ineffective water sprays;
- Ventilation/gas drainage ineffective in dissipating local gas make;
- Missing rubbers on ventilation duct reducing face ventilation;
- Not aware that in-seam or exploratory horizontal or vertical boreholes had been cut through;
- Current gas drainage methodologies do not eliminate residual gas make from borehole stubs;
- Not all crew members have current awareness of frictional ignition procedures in detail;
- Inaccessible and complex Safety and Health management systems;
- Latest wet head technology or active suppression systems not fitted (currently not available).

The occurrence of recent Australian FIs indicates that there are opportunities to improve on existing controls (to eliminate human interventions), and for improving the risk profile the following are suggested:

- Review and upgrade of the current FI SOPs and Safe Work procedures;
- Ensure adequate stone dusting for prevention of coal dust explosions;
- Adequately maintain coal cutting machine;
- Regularly change worn out picks and keep all sprays in designed condition;
- Create greater awareness of FIs for all workers;
- Investigate the viability of introducing “wet head” technologies;
- Review the configuration of existing underground UIS drainage hole monitoring;
- Investigate the viability of sealing boreholes;
- Investigate options of roadway geometry including cutting less roof and more floor;
- Research other mines recent experience with FIs.
**FRICIONAL IGNITION LEADING PRACTICES**

Energy needed from literature states that as little as 0.3 milli-joules of electrical energy is required to ignite a methane air mixture. This is equivalent to $1/120\,000\,000$ of the energy used in 1 second by a 37 kW motor or about one-fiftieth of the static electricity accumulated by an average size man walking on a carpeted floor on a dry day (Du, 1994). Therefore, any improvements in eliminating or further reducing the heat energy source (Figure 4) would help in succeeding the challenges over FIs. The section hereafter discusses the cross-pollination of technology and in particular use of chilled water sprays for prevention of FI commonly used in South African gold and metal mines.

Managing human behaviour including lack of discipline is difficult but key contributing factors in failure of series of controls leading to FIs. In the case of frictional ignitions, failure of an individual could have significant multiple consequences. Therefore, where possible, introduction of new technologies that would overcome such breaches due to human factors would be helpful to the mining industry.

![Figure 4 - FI fault tree analyses (Robertson, 2010)](image)

The general principles of the frictional ignition control, viz., ventilation, water sprays has been researched and documented (BCC, 1990) and practiced in the mining industry globally. The generic mitigation strategy based on reducing the friction, cooling the cut surface and/or reducing local methane concentrations can be summarised as follows:

- Ensure that adequate ventilation is always in place in development headings such that any ignition is contained to a small a methane source as possible. Explosion cannot occur in the presence of adequate ventilation;
- Reduce and dilute methane make from the face area to minimise the development of pockets of explosive methane;
- Design, maintain and operate cutting systems that minimise friction when cutting rock so as not to create ignition conditions;
- Maintain and operate effective back pick flushing sprays that cool rock surfaces behind the pick.

**Cutting pick condition management**

Globally, according to various reports, cutter pick condition is the most important factor because nearly 70% of all FIs initiated by this means. All coal cutting heads are equipped with picks (Figure 5) featuring hard cutting tips for wear resistance and these picks wear quickly when cutting hard rock. Picks are fitted to pick blocks welded to the cutting drum, often via a removable adaptor piece or pick sleeve. The lacing of the drum determines the spacing and sequence of pick contact on the rock surface and the set-up of the tool determines the angle of attack, pick body clearance etc.
positive note, recent technologies have created high temperature stable diamond composite materials that offer encouraging life.

![Cutter head with picks (left) and Pick condition reference guide (right)](image)

**Figure 5** - Cutter head with picks (left) and Pick condition reference guide (right)

**Face environmental monitoring**

Other key feature of early prevention practices in prevention of FI is detection of hazard. In the prevention of FI, critical monitoring parameters of interest are methane, section or face air velocity, alarm settings of these monitors. Studies (Kissell, *et al.*, 1986) have suggested that the methane monitor on the mining machine is an essential control for early detection of gas which would shut off the machine sooner. Depending on the machine type and data communication systems, it would useful to understand the trends of gas emissions from face area. This is one of the FI control barriers. In all or most of FI incident investigation reports it is reported that there was a failure to analyse the pre-ignition gas trend due to limited manual gas records or unconnected real-time recording and data collection system. Improvement in collection of this crucial information is worth the effort for improved understanding and management of FI risks in the face area.

**Application of chilled water sprays**

It is noted that frictional energy and thus the thermal environment near the picks is an important contributing factor in initiating an FI. It is recognised that FI is not caused by the orange sparks which are observed as particles which are torn off during mechanical process and ejected glowing visibly. The cause of FI is the hot spot left behind the trailing edge of the pick on the rock. The hot spot is formed by a thin layer of metal wiped on to the rock, or rock adhering to the metal and then smeared. Depending on its lifetime, size of smear and temperature, this smear ignites methane if present. The temperature at which both Tungsten carbide and quartz liquefies is about 1710 ºC whilst the melting point of carbon steel is 1500 ºC. Therefore, chilled water sprays with arrival temperature of 8 ºC to 10 ºC may present as a simple, rapid, reliable, and proven technology for FI prevention.

Effects of hot thermal environment have been studied and effectively managed in coal mines elsewhere in South Africa, USA, Europe and Australia. The first use of cooling plant was in Brazilian gold mines in the late 1800s. In the mid 1930’s, first surface bulk air cooling was first started in South African mines. Improvement in cooling technology has resulted in current use of slurry ice and chilled water on a daily basis in those hot underground mines. Chilled water technology has become mature and its application in Australia is just a matter of acceptance and time.

In Moranbah region coal mines at a depth of 300 m, the thermal environment at a coal face is the same as met by those gold mines of South Africa at a depth of 4 000 m and Platinum mines at a depth of 1500 m. Currently, bulk air cooling is the strategy that is being implemented in Australian coal mines. Current BAC discharge air temperatures vary from 10 ºC to 13 ºC with mixed air with temperature at the shaft bottom of 18 ºC. As known from Australian experiences, ventilation air alone will not contribute a significant cooling impact. The challenge is the arrival temperature of cooled air at coal face which has to travel long lengths of roadways with positional efficiency of cooled air at its lowest at coal face. Therefore, alternative cooling strategy such as chilled water is a practical opportunity and would aid in cooling and avoiding FI environment.

Traditionally, mine cooling is commonly achieved in the following order in the mining industry:

- Ventilation;
- Surface bulk air coolers (direct spray heat exchangers);
- Underground bulk air coolers;
- Spot cooling coils;
- In-stope coolers;
- Chilled water;
- Surface ice plant for slurry ice cooling.

The experiences of chilled water sprays used for cooling deep hot mines in South Africa is highly relevant to FI prevention and incendive/face heat management in Australian coal mines. From a FI prevention perspective, ventilation and water sprays (used for dust suppression and methane dilution through ventilation) are established technologies that are widely used in coal mines. However, greater benefits of cooling or sharp reduction in incendive heat from cutting picks by the use of chilled water sprays outweighs the current practice of service water. For example, introduction of chilled water sprays and millions of fine chilled water droplets at temperatures of 8 °C to 10 °C would provide a rapid heat removal rate of 210 kW and 1 005 kW Continuous Miner faces and Longwall (LW) shearer faces respectively (Figure 6). On the other hand, CM and LW face ventilation alone may provide a heat removal range between 70 kW and 350 kW. Thus, chilled water spray droplets have the potential to become ‘improved last line of defense’ against gas ignitions.

Figure 6 - Use of chilled water sprays behind the picks

It is simple operational practice in mines with Virgin Rock Temperature (VRT) of 55 °C to use jet sprays using 4 to 5 L/s of chilled water to rapidly fix and improve hot face environment conditions. Typical daily chilled water consumption in those metal mines is approximately 200 L/s against the Australian coal mine conditions that may require up to 50 L/s of chilled water. Unlike laboratory environment, due to complexity of mining conditions, stand alone benefits of chilled water sprays have been difficult to quantify other than operational qualifications.

In the case of FI, concepts such as cyclical cooling and ventilation on demand must be considered by coal mines with extreme caution as they may lead to ‘delayed cooling’ or creation of unplanned explosive atmospheres.

Application of wet head systems

The original concept of the wet head system on CMs was centred on water sprays being emitted from nozzles mounted on the rotating drum head and close to each bit. This differs from conventional external sprays as water is sprayed continuously to the bit and its vicinity as it cuts the coal. The water is strategically placed in the Pick Back Flushing Mode (PBF) position to reduce the probability of FIs. The wet head concept was developed in the 1970s by the former United States Bureau of Mines (USBM), the Bituminous Coal Re-search National Laboratory (BCRL) and various manufacturers (Merritt, 1987). The spray system on the wet head are integral with the tool holder and spray in the PBF mode known to be essential to reduce the incidence of incendive ignitions (Browning and Warwick, 1993; Courtney, 1987; Powell and Billinge, 1981).

It has been clearly established that FIs are caused by hot material that is ejected from behind the cutting tool and that water can be effective in preventing an ignition. It has also been established that less water is required when applied behind the cutting tool instead of in front (Powell and Billings, 1981). A
laboratory study at Pittsburgh Research Centre using cutter-drum-mounted water sprays indicated that FI during dry cutting of sandstone with a carefully designed water spray nozzle located behind the bit (PBF mode) substantially reduced the likelihood of FI (Courtney, 1987).

Sprays directed at the cutting area near picks ("wet heads") have been the subject of much development work, both for dust suppression as well as for FI mitigation. Several companies (Hydra Tools, Joy, Eimco, Sandvik (excluding ABM machines), and Kennametal) have conducted wet head development programs. Successful designs are commercially available for shearsers and road headers but are still in the prototype stage for CMs. This is largely due to difficulties in plumbing water (sealing related problems) to the rotating head and in maintaining clear pick jets.

Despite the interest in both the 1970s and the 1987 trials of the wet head on CMs, there is a lack of further literature indicating progress. This state-of-affairs is, in fact, explained by comments from the machinery manufacturers who have indicated that the high cost of producing and maintaining wet heads (including the cost of sophisticated seals), and, despite enquiries during the early 1990's, no wet heads were ordered until late in 1994 (Phillips, 1997). Further to this, first series of wet head tests using new seals were carried out in South Africa (Belle and Clapham, 2001) have indicated the benefits of wet heads in terms of dust and bit consumption. Furthermore, the CM operators indicated that improved visibility in the cutting zone.

**Active On-board explosion suppression system**

Active on-board explosion suppression systems are mounted on coal cutting machines and detect the presence of a methane ignition by means of light sensors. The electronic signals from the sensors trigger the suppression system which creates a barrier of flame-suppressing material, thus containing the flame in the immediate vicinity of the ignition and so preventing further development and propagation of an explosion.

Active suppression systems can therefore be used in conjunction with ‘traditional’ methods of explosion prevention. Several active suppression systems have been tested in the 20 m rectangular Kloppersbos explosion tunnel to determine whether a system or configuration is able to fulfil the acceptable criteria for various cutter head positions, methane concentrations and roadway heights (Belle and Du Plessis, 2000).

In the year 2000, in response to an approach from the French research institute INERIS, which required further precautionary measures for their collieries, an on-board active suppression system, "Explo-Stop®", was trialed in South Africa (Figure 7). This project involved four different companies - Centrocen, the CSIR, INERIS and HBCM - the French mine in which the system was to be installed subsequent to the success of the tests.

The results of the tests showed that the system managed to suppress a methane mixture volume of 180 m³ of a 9% methane concentration. This was achieved with a temperature rise of less than 100 ºC at the operator's cab and no flame was detected at the operator’s cab.

A South African thermal colliery further installed an active suppression system (in early 2000's) and its performance could not be verified (as there were no triggers) and further installation was not pursued. Similarly, due to closure of all mines in France, experiences on active suppression system are not available.

![Figure 7 - The Explo-Stop® on-board active suppression system on a Dosco 1300H](Image)
Frictional ignition limit awareness

Awareness of FI is quintessential in the management of it as it drives the consistent application of the prevention methodologies and adherence to procedures and standards.

Any ignition in an underground coal mine is an issue of great concern to all parties. There exist in some Queensland mines, rules relating to maximum in situ methane gas content of 5.75 m$^3$/t to reduce FI risk. This was based on past experiences of FI incidents in seams with gas levels above 6.5 m$^3$/t in the 1990’s. FI risk is ever present in a coal mine, where a recent case of FI occurred in Queensland in a similar mining seam after 12 years despite complying with the 5.75 m$^3$/t gas limit. For example, a study by Krog and Schatzel (2009) noted that 90% of all FIs occurred in u/g coal mines that liberated at least 0.39% of CH$_4$ through their mine ventilation system. Interestingly, US production data suggested that there is no relationship between FI and productivity.

Figure 8 shows the FI statistics between 1990 and 2010 in metallurgical coal (Australia) and thermal coal mines (South Africa). As can be seen, there is no relationship between gas content and number of FI incidents. A study by Krog and Schatzel (2009) noted that 90% of all FIs occurred in underground coal mines that liberated at least 0.39% of CH$_4$ through their mine ventilation system. However, South African statistics indicated that FI have occurred in coal and gold mines that liberated even with lower methane emissions between 0.02% and 0.05% through their mine VAM system (Belle, 2009). Therefore, it is important to communicate that FI risks are ever present in coal mines regardless of gas contents or gas emissions.

![Figure 8 - FI incidents in metallurgical (Australia) and thermal coal mines (South Africa)](image)

Ignition of methane is applicable to metal mines with known fatalities in gold and platinum mines of Witwatersrand with extremely low gas contents. As recent as previous decade, several lives have been lost due to ignition of methane related explosive gases. On a positive note, there are suitable standards and procedures and PHMPs related to FI. However, misunderstanding of FI limit such as 5.75 m$^3$/t be eliminated during early training and mine induction period.

CONCLUSIONS

The preceding discussions on causes of FI are not new. The FI and explosion statistic definitely provide valuable information to drive and guide proper safety strategies at the time of risk assessment, determining likelihood and consequences. However, some of the identified new technologies such as chilled water sprays on CMs and LW shearers, use of wet head systems, active suppression systems and improved air velocity and gas monitoring systems will further aid in alleviating FI risks. Chilled water use in coal face area will have the added benefit of reducing thermal stress hazards. Although there is no empirical evidence of a relationship between chilled water and FI incidents, the progressive and safety conscious nature of mining industry would witness the benefits of cooling picks and face area using chilled water sprays, if adopted in years to come. This paper was written in the hope that it will enhance cross-pollination of experiences and aid in the prevention of FIs in mines through transfer of established technologies (in gold and platinum mines in South Africa) such as chilled water on face cutting machines.

In conclusion, this paper has presented ideas for prevention of FIs in the mining industry. It is noted that benefits of chilled water sprays for prevention of FIs will only be reflected through future FI statistics.
Lastly, the slogan (Phillips, 2007), “the price of safety is eternal vigilance,” applies to FI and any tools to improve such vigilance must be embraced sooner rather than later.

ACKNOWLEDGEMENTS

Authors are indebted to various sources of knowledge that were developed in the past that have resulted in a better understanding of management of FI in mines.

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DUST CONTROL PRACTICES IN THE INDIAN MINING INDUSTRY

Jai Krishna Pandey

ABSTRACT: Mining is a dust prone occupation and almost every major process in mining contributes to the atmospheric load of suspended particulate matter. Prolonged exposure of this dust is known to cause various respiratory diseases including deadly pneumoconiosis among the miners. The Central Institute of Mining and Fuel Research, Dhanbad has contributed significantly in the area of dust assessment and control. It has developed a few simple yet effective techniques for controlling dust in drilling, crusher houses, transfer points, and haul roads. These techniques of dust control are gaining industry interest in recent years mainly for two reasons: (a) dust generation has increased significantly due to higher mechanisation and the introduction of mass production technologies to meet our growing production needs making application of dust control mechanism inevitable, (b) growing consciousness of environment and stricter environmental compliance mechanisms has put constant pressure on the mining industry for regular use of dust control practices. The present paper briefly describes the techniques/methodologies for controlling dust during different drilling practices and at crusher house and conveyor transfer points and haul roads in the mining industry.

INTRODUCTION

Coal is a prime source of energy for India and it will continue to maintain its lead for the foreseeable future. Dust is an accepted fact for almost every operation in coal mining. Many processes can be pinpointed as contributions to dust generation like drilling, blasting, haul roads, coal cutting by continuous miner, conveyor belts and crusher houses. With an increased level of mechanisation and the pressing demand to boost production for minimising the supply gap, generation of air borne respirable dust is increasing necessitating more effective dust control practices. Being a labour intensive industry, coal mining warrants extra efforts to mitigate dust pollution.

Prolonged exposure of coal mine dust is known to cause various respiratory diseases like pneumoconiosis, silicosis, bronchitis, asthma, fibrosis of lungs and tuberculosis (TB), depending upon the nature of the dust. Free silica/quartz present in the dust of mine air has been identified as a main cause of these health hazards to miners. Indian coals are considered to be of ‘drift’ origin and therefore contain high mineral matter intermixed with coal matter. Quartz is one of the major minerals present in coal and therefore miners are exposed to health risks arising from inhalation of quartz laden coal dust generated in the coal mines (Nair and Sinha, 1988). Dust with quartz content of up to 14.49% has also been reported in coal mines of Bhart Coking Coal Limited, Dhanbad (Pandey, et al., 2008). The health risk to the miners varies depending upon the nature of coal and its mineral content, condition of the mines, nature of job handled by the miner and finally the quality and the efficacy of the safety measures adopted by management.

Air-borne dust from mining activities spreads over nearby populated areas and crops causing harmful effects in many ways to the people, vegetation, forests, animals and water resources. The corrosive effect of the dust shortens the life of lubricants of Heavy Earth Moving Machinery (HEMM), increases maintenance costs and reduces its operating efficiency. The dust impedes visibility thereby reducing production capacity. It is also a potential safety hazard.

DUST CONTROL PRACTICES IN INDIAN COAL MINES

Dust control involves either dust consolidation or dust capture. Dust consolidation is normally practiced for settled dust which becomes airborne in favourable condition e.g. haul roads of open cast mines. Dust collection or capture is resorted to when it is airborne. In these cases dust is collected close to its source of generation for effective result and therefore the method is useful for controlling dust generated in localised spaces or point sources like drilling, blasting, crusher house and conveyor transfer points with the help of special types of dust capture arrangement. Dilution of dust is limited to small concentration of dust only. The Central Institute of Mining and Fuel Research (CIMFR) has made a significant contribution in this area and has demonstrated some very effective yet simple techniques for

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providing dust control in various situations which are described in the following sections. Few face operations including blasting do not permit efficient and cost effective dust control. Preventive steps like pattern of holes, quantity and strength of explosive and water stemming facilitate lower dust generation and are probably the better tools to overcome the problem.

**Dust control in drilling**

Drilling produces the largest quantity of respirable dust per unit weight during the shortest time. A study (Nair, *et al.*, 1999) reports up to 1.46 kg of respirable dust generation per meter of drilling by a 250 mm diameter drill in iron ore opencast mines. Table 1 (Sinha and Nair, 1982) presents the level of dust generation during drilling in coal, limestone and iron ore which reveals that air borne dust generated in drilling increases with drill diameter and rock hardness. Analyses of dust collected from these drill holes reveals that the bulk of the drill hole dust (up to 65%) was above 500 micron in size, up to 12.2% was under 53 micron and up to 1% was in the respirable range.

**Table 1 - Quantity of total dust generated from drill hole for coal, limestone and iron ore for different drill hole diameter**

<table>
<thead>
<tr>
<th>Diameter of drill hole (mm)</th>
<th>Amount of dust (kg) generated per meter of drilling</th>
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<tbody>
<tr>
<td></td>
<td>Coal</td>
</tr>
<tr>
<td>60</td>
<td>3.7</td>
</tr>
<tr>
<td>100</td>
<td>10.2</td>
</tr>
<tr>
<td>150</td>
<td>23.0</td>
</tr>
<tr>
<td>200</td>
<td>40.9</td>
</tr>
<tr>
<td>250</td>
<td>63.75</td>
</tr>
<tr>
<td>300</td>
<td>91.92</td>
</tr>
</tbody>
</table>

Drilling is essential in mining and large quantities of dust will be produced irrespective of the method of drilling. Therefore, various methods of dust suppression will have to be introduced to bring down the concentration of dust to safe limit. The principal dust suppression methods are wet drilling, suppression by fog and dry dust collection.

The wet drilling method is based on the introduction of water into the hole being drilled, through the centre of drill steel. But it has not yet been possible to establish the relationship between the dust collecting capacity and the shape and size of the bit or even the number of outlet holes for the water (Parmeggiani, 1983). However, water has poor efficiency for collection of respirable dust (Sen and Ghosh, 1985). Water adds to the risk of jamming of the drill bit inside the drill hole, and reduces the rate of drilling. In underground mines, water added for drilling creates foggy situations, which lead to poor visibility. Addition of small amount of soluble oil (0.1-3%) creates an emulsion which reduces surface tension of the water and may improve the performance and extends the life of the bit. Many studies have shown that the addition of wetting agents to the circulating water is expensive with relatively little effect on fine dust collection.

Besides economics and efficiency, the situation may also arise where wet drilling is undesirable either because the machines become clogged, or because the spray and fog released by the machine may affect nearby electrical installations or create undesirable conditions for the operator drilling vertical overhead holes. There may also be workplaces that are not equipped with a water supply. Dry dust collection is therefore more common. The conventional practice of dry dust collection is energy consuming and not effective. CIMFR has developed few dry dust collection systems for the prevalent form of drilling practices used in Indian mines. The advantages of the CIMFR designs are:

- The dust collectors are portable and easy to use;
- They do not use any source of energy for operation – no water, electricity or anything else;
- They do not affect the rate of drilling, rather than improve it by avoiding jamming and energy waste;
- Collection efficiency is very high;
- No moving parts, therefore, no wear and tear only some inexpensive washers need replacement;
- The dust collectors are easily fabricated by small scale units;
- Operators do not require any special training for their use. Even unskilled labour can use them;
- The dust collectors are inexpensive.

Dust extractor for jack hammer drilling

The dust extractor (Nair, et al., 1999; Nair and Sinha, 1984) is comprised of a hood with a cushion base, an elastic collar attachment to cope with intermittent hammer motion and an elastic collar grip on the drill rod to prevent air leakage to atmosphere. A collared base plate is added to the bottom of the hood above the cushion base. A long filter bag is attached to the funnel opening. A foot pedal ensures firm grip of the device to the ground during drilling. The arrangement of the dust extractor is shown schematically in Figure 1.

![Figure 1 - Dust extractor for jack hammer drill (not to Scale)](image)

The jack hammer drill rod passes through the elastic collar, and through the base plate collar. The elastic top collar provides a leak proof grip on the drill rod, and yet it lets the rod move up and down during hammer action. The flexible gasket too permits free hammer motion without leakage. The base collar does not allow dust to fall back to the drill hole. During drilling, the cuttings are transported to the hood by the scavenging compressed air, which eventually gets channelled through the filter bag. The bag lets the air go out and retains all the dust. This gives collection efficiency (Table 2) of about 90%. The dust collection process is dry. No dust is allowed to fall back to the drill hole. For this reason use of the device improves on the rate of drilling. The device is ideal for secondary drilling as well as horizontal drilling in underground mines.

<table>
<thead>
<tr>
<th>Operation</th>
<th>Drilling without extractor in ppcc*</th>
<th>Drilling with extractor in ppcc*</th>
<th>Dust collection efficiency %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Before drilling</td>
<td>200</td>
<td></td>
<td></td>
</tr>
<tr>
<td>During 1st hole drilling</td>
<td>2720</td>
<td>440</td>
<td>83.82%</td>
</tr>
<tr>
<td>During 2nd hole drilling</td>
<td>3973</td>
<td>436</td>
<td>89.03%</td>
</tr>
<tr>
<td>During 3rd hole drilling</td>
<td>5695</td>
<td>659</td>
<td>88.42%</td>
</tr>
</tbody>
</table>

*particles per cubic centimetre

Dust extractor for electric rotary drilling

Electric rotary drilling is very common in underground mine where several such holes are drilled every day. This generates about 42 g of respirable dust per minute (Sinha and Nair, 1982) which cannot viably be diluted to safe limits. The dust extractor (Nair, et al., 1999) designed for electric rotary drilling is shown schematically in Figure 2.

The extractor weighing about 1.25 kg is comprised of a hood and a wide cushioned base, a narrow collared mouth, and a funnel shaped dust outlet to the side of the hood. A long collector bag is
attached to the funnel. A handle holds the hood on its foam base to the vertical coal strata while the bag hangs to the ground from the funnel end. During drilling, the drill rod passes through the mouth of the hood, and through the base till it meets the coal to be drilled. The design of hood takes into account the lateral throw of the dust during drilling. Forward motion of the drill and the gravitational fall of dust down the bag creates negative pressure within the hood which prevents dust heat through its mouth. The cushion base firmly resists leakage between coal and hood. All these result in nearly 100% dust collection.

![Diagram of dust extractor](image1)

**Figure 2 - Dust extractor for electric rotary drill (not to Scale)**

**Dust collector for drilling in mine roof**

This is basically a light weight (less than 2 kg) portable dust collector (Nair, et al., 1999) explained schematically in Figure 3. It consists of a cushion base, collared disc with spring action handle. A long collector bag hangs from the collar. A port of entry for the drill rod is provided in the collector bag, the mouth of this opening is stiffened and is provided with ring gaskets. During drilling the cushioned disc is placed against the roof and supported by a handle. The drill rod is allowed through the hole in the bag, till it touches the roof. The bag is flexible enough to permit drilling for vertical holes or inclined holes. The hanging bag is positioned along the handle so as to avoid obstruction to the drilling process. The collector permits collection of about 90% the dust.

![Diagram of dust collector](image2)

**Figure 3 - Dust collector for drilling in mine roof (not to Scale)**
Dust arrester for large diameter deep hole drilling in open pit mines

Up to a few hundred kilograms of broken dust is generated from each drill hole. The dust is finer than what is formed in other mining operations and contains a significant proportion of respirable dust. Normally this dust is flushed out from the drill hole by a few cubic meters of compressed air, at high velocity. This dusty air loses its kinetic energy upon reaching the surface, and a dust cloud is formed around the drill. The energy which is wasted in polluting the environment is put to use in the CIMFR designed Dust Arrester (Nair and Sinha, 1987) to clean the air of its own dust.

The dust arrester is a rectangular box with a ring shaped foam cushion washer at the top, a collar base plate with a cushioned bottom, and a large opening on one side to which a specially shaped long and tough filter bag is attached. Figure 4 explains the design of the dust arrester schematically. For dust collection, drilling is carried out through this device. The box is placed on its cushion base at the selected site. The drill rod is introduced through the top ring cushion and through the base plate collar till the drill rod touches the ore body. The dust filter bag is fully stretched. The hole in the ring cushion is smaller than that of drill rod, but flexible enough to let the larger drill bit pass through. This ensures an airtight grip between the drill rod, and the ring cushion yet permit free rotation of the rod. During drilling, part of coarse dust gets deposited around the base collar and fines pass through the filter bag. The base cushion gets pressed against coal strata due to the weight of dust settling inside the box, thereby preventing leakage of air at its base.

Figure 4 - Dust arrester for large diameter deep hole drilling (not to Scale)

Thus, the cushion at the top surrounding the drill rod and the cushion at its base together make the extractor an air tight unit, and therefore, dusty air from drill hole moves on its own towards filter bag for dust collection. The large surface area of the filter bag permits slow filtration of dusty air through its pores. Table 3 and Table 4 present the performance study of the dust arrester in stone and iron ore. The current design is ideally suited for drills of diameter up to 120 mm. A modified version is under consideration for larger diameter drills. The total weight of the box is about 5 to 6 kg only and that of bag another 3 to 4 kg.

Table 3 - Air-borne dust concentration measured during drilling in stone with and without dust arrester using 150 mm drill master drilling machine

<table>
<thead>
<tr>
<th>Time measured from start of drilling</th>
<th>Drilling with dust arrester (ppcc)</th>
<th>Drilling without dust arrester (ppcc)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1 minute</td>
<td>160</td>
<td>1998</td>
</tr>
<tr>
<td>5 minute</td>
<td>177</td>
<td>3000</td>
</tr>
<tr>
<td>10 minute</td>
<td>220</td>
<td>4940</td>
</tr>
<tr>
<td>15 minute</td>
<td>230</td>
<td>8053</td>
</tr>
<tr>
<td>Average</td>
<td>198</td>
<td>4650</td>
</tr>
<tr>
<td>Reduction in respirable dust</td>
<td>95.7%</td>
<td></td>
</tr>
</tbody>
</table>

Dust control at crusher house and transfer points

A significant amount of dust is generated at crusher houses and transfer points in the coal mines. CIMFR, Dhanbad has given a concept of the Canopy Curtain Method (Pandey, et al., 2001) for collecting dust at dust generating point sources like coal unloading bunkers, crushers, screening points and conveyor transfer points with dust collection efficiency of more than 80%. The technique involves dust collection at start point. It essentially requires filter cloth to make an appropriate enclosure at the
dust generating point. Porosity of the cloth enables it to take advantage of the natural wind. The enclosure has to be of sufficiently large size so that the up thrust velocity of dust generated due to impact of coal diminishes out. The dust will then be permitted to stick to the walls of the enclosure or escape through the chimney by the action of the normal air current, diffusion or otherwise. The enclosure has to be kept wet by capillary action or by wetting the canvas with the water drops from the top or a combination of both for a better and more effective result. A wire net framework will be required to keep cloth enclosure in order. A small exhaust fan can be fitted in the chimney to further improve the dust collection efficiency. A laboratory experiment was designed to assess the efficacy of the techniques. Dust collection efficiency obtained in conducted a six set of experiments varied between 76-94%. It is more for silica or stone dust as compared to coal dust. It has been found that wetting the canopy significantly improves the dust collection efficiency.

Table 4 - Air-borne respirable dust (ARD) concentration measured during drilling in iron ore with and without dust arrester using 150 mm drill master drilling machine

<table>
<thead>
<tr>
<th>Time in minute measured from start of drilling</th>
<th>Drilling with dust arrester (ppcc)</th>
<th>Drilling without dust arrester (ppcc)</th>
<th>Drilling with dust arrester (ppcc)</th>
<th>Drilling without dust arrester (ppcc)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Before drilling started</td>
<td>80</td>
<td>53</td>
<td>95</td>
<td>105</td>
</tr>
<tr>
<td>1 minute</td>
<td>143</td>
<td>8 040</td>
<td>105</td>
<td>100 000</td>
</tr>
<tr>
<td>5 minute</td>
<td>143</td>
<td>15 860</td>
<td>240</td>
<td>113 000</td>
</tr>
<tr>
<td>10 minute</td>
<td>139</td>
<td>16 950</td>
<td>120</td>
<td>80 000</td>
</tr>
<tr>
<td>15 minute</td>
<td>120</td>
<td>16 950</td>
<td>120</td>
<td>80 000</td>
</tr>
<tr>
<td>20 minute</td>
<td>280</td>
<td>3 120</td>
<td>144</td>
<td>429 000</td>
</tr>
<tr>
<td>25 minute</td>
<td>240</td>
<td>5 990</td>
<td>265</td>
<td>71 770</td>
</tr>
<tr>
<td>30 minute</td>
<td>213</td>
<td>2 770</td>
<td>280</td>
<td>48 720</td>
</tr>
<tr>
<td>Average</td>
<td>183</td>
<td>9 442</td>
<td>188</td>
<td>71 327</td>
</tr>
</tbody>
</table>

Dust control in haul roads

Unpaved haul roads in coal mines are a veritable source of dust pollution supporting normally 10 to 15 mm of dust on its surface (Pandey, et al., 1999). Dust from haul roads gets lifted and floats in the air during movement of trucks and forms a dust cloud. With an increase in the weight of trucks, speed and frequency of traffic, the cloud may appear to be continuous causing delays and difficulties. During continuous dumper runs, dust loads of surrounding atmosphere builds up both vertically and horizontally. Application of water at frequent intervals, remained by far the most practical solution for the control of dust on haul roads. Water applied in the conventional way gets dried up fast, and its replenishment at frequent intervals (up to 15 times a shift) becomes necessary for effective dust control which adds significant cost. In a case of Block II, Open cast Project of Bharat Coking Coal Limited (BCCL), water spraying over a 3 km long, 20 m wide haul road cost Rs 25 000 over a period of one year (Pandey, et al., 1999). It becomes more cumbersome and costly where water is not easily available. Unfortunately water becomes a scare resource in summer when it is required most. Besides, water has a poor wetting ability for coal. Therefore instead of penetrating into the dust and consolidating it, it flows down the sloppy road and make it muddy. A number of techniques have been adopted addressing these issues which includes application of hygroscopic chemicals like calcium/magnesium to increase water retentivity. These chemicals require repeated application as they are re-dissolved in subsequent water sprays and tend to drain out to lower levels in the usually sloppy mines, adding to cost of treatment. Spray with oil-water emulsion also helps to consolidate dust but it does not penetrate deep and underlying dry dust layers gets airborne quickly during dumper movement. Some surface crustling agents like cohex have also been tried which may need weekly or daily application. CIMFR, Dhanbad has contributed the following environment friendly and techno-economically viable methodologies for controlling dust in unpaved mine haul roads:

- Wet encrustation using super absorbent chemicals which can absorb, retain and reabsorb water several times, its weight without getting dissolved. The methodology involves mixing of the chemical with road dust and application of the water. It helps in very effective water management by increasing the water retaining capacity of road dust and in the process consolidates the dust and conserves water. This chemical is not known to have any environmental ill effects. Poor chemical absorbed water is not squeezed out under the
compressive force of tyres, the bondage being at molecular level. Poor solubility of the chemical increases the effective span of the treatment cycle to several months. In the case study of Block II Opencast project of BCCL it could result in a saving of about 45% over the water spraying. (Pandey, et al., 1999)

- To avoid water wastage and improving the economic viability of water spraying, it has also been proposed to selectively wet the road surface close to the tyre/road surface contact plane, before tyre to surface contact occurs, which can effectively eliminate dust emission from haul roads with far less spray of water. This can be achieved by designing a system to spray an adequate quantity of water ahead of the front tyre in each dumper. The system should be wide enough to match back wheel width and should preferably be fitted with inwardly facing sprays (Nair, et al., 2001).

- Chemical dust suppressants are nowadays gaining more acceptance in the industry for controlling dust at haul roads probably due to the fact that the application methodology for these chemical dust suppressants fits well to the conventional water spraying. Keeping in view their industry acceptance, Director General of Mines Safety has also issued circulars for ensuring environmental safety and hazard issues. One such chemical, Dustron PC Coal has been developed by Syntron Industries, Ahmadabad in collaboration with CIMFR, Dhanbad. The product is non toxic, biodegradable, meets all the safety standards as per statutory requirement and has been proved to be very effective in controlling dust at haul road (Trivedi and Kumar, 2011).

- Dustron PC Coal is poured into the conventional water spraying container in recommended dilution and sprayed on the haul road surface in a conventional way. It improves water penetration, water retention, agglomeration of dust and reduces the water consumption of dust with improved dust control. Syntron Industries has conducted a number of studies for dust control with the help of Dustron PC used in mines haul road. The results of these studies conducted at various coal and metal mines vis-a-vis conventional water treatment with respect to various parameter on these haul roads are being summarised in the following six tables. These studies (Trivedi and Kumar, 2011) reveals the significant improvement in water conservation on mine haul roads, including:

  1. Water requirement decreases by more than 50% and a commensurate reduction in diesel consumption for running of water tanker. Moisture of haul roads are increased three fold in comparison to normal watering.

  2. Sieve test analysis of haul road dust with chemical and with water alone reveals that application of chemical improves agglomeration conditions as fines (size 0.5 mm or less) have been reduced after application of chemical by 80%.

CONCLUSION AND RECOMMENDATIONS

CIMFR, Dhanbad has made a number of significant contributions to fight the menace of dust in mining operations. Most of these are very simple and effective in controlling the dust generated due to various mining operations. But their potential has not been optimally utilised as dust control largely remained limited to convention as water spraying and wet drilling. In the wake of the mechanisation and developmental need of the country, the mining operation has made a quantum jump which has significantly increased the dust generation level. This, coupled with growing consciousness for environmental and health hazards, and stricter environmental compliance mechanism is virtually forcing mining operations to use dust control mechanisms with dust prone operations, the industry is now looking for techno-economically viable dust control solutions. Accordingly most of these have started gaining acceptance of the mining industry. Dust collectors described for various types of drilling operation provides inexpensive and easy to use methods with high performance efficiency. Dust suppressing chemical is a very convenient, cost effective solution for unpaved haul road dust consolidation. A canopy curtain method definitely deserves a trial for transfer points particularly crusher houses. Mass production technologies like longwall, highwall and continuous miners are also coming up in India which has a larger dust generation potential than conventional method of working. Adding dust suppressing chemicals in water jet spraying at cutting faces for improving wettability will significantly reduce the dust generation.
ACKNOWLEDGEMENTS

The author is thankful to Director, CSIR-CIMFR, Dhanbad for his kind permission to present the paper for the conference.

REFERENCES


A CRITICAL EVALUATION OF DUST SAMPLING METHODOLOGIES IN LONGWALL MINING IN AUSTRALIA AND THE USA

Brian Plush, Ting Ren and Naj Aziz

ABSTRACT: Questions relating to the validity and subsequent suitability of the current dust sampling methodologies utilised in Australia and the USA have recently come under scrutiny. The reason for this scrutiny is that there has been a significant increase in Coal Workers’ Pneumoconiosis in the USA over the last few years despite recorded conformance to exposure level legislation. The opinion by many in the underground coal mining operators in Australia is that the current testing regime tells them very little about the actual operational production of dust on the longwall face in relation to where it is produced, how much is produced or how efficient installed controls are at preventing this dust from entering the atmosphere. Evaluation of the current testing regimes in Australia and the USA are proposed, which identify limitations that are raising questions relating to its suitability to ensure worker health in the underground coal mining operators.

INTRODUCTION

Evaluation of a workplace is primarily undertaken to establish if the workplace environment is safe for employees to perform their normal duties. Occupational hygiene has been an integral part of the mining operators for centuries; however its importance has grown with developments in mechanisation and rising community expectations of better occupational health.

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. The US EPA describes inhalable dust as that size fraction of dust, which enters the body, but is trapped in the nose, throat and upper respiratory tract. The medium aerodynamic diameter of this dust is about 10 µm.

A report by the director of mine safety operations branch of Operators and Investment NSW has found that there is an increasing level of dust being ingested by coal miners in New South Wales, potentially leading to long-term health problems (Chief Inspector Safety Bulletins, NSW, 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 operators 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, stage loader/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.

Studies by NIOSH in the USA have shown that 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. The continued occurrence of CWP in underground coal mine workers and the magnitude of respirable and inhalable dust overexposures in longwall mining occupations illustrate the need for the mining operators to improve existing dust control technology on longwalls, not only in the USA, but in Australia as well, to prevent the incidence of lung diseases from occurring.
Dust sampling of employees in Australian coal mines is carried out by cyclone separation and collection of the sized particles for weighing, generally over the period of a full shift to measure personal exposure levels to airborne contaminants. This testing methodology is described in AS2985 Workplace Atmospheres - Method for sampling and gravimetric determination of respirable dust and AS3640.

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 (NSW Govt. Gazette).

Questions relating to the validity and subsequent suitability of the current dust sampling methodologies utilised in Australia and the USA have recently come under significant scrutiny. The reason for this scrutiny is that there has been a significant increase in Coal Workers’ Pneumoconiosis (CWP) in the USA over the last few years despite recorded conformance to exposure level legislation, and the opinion by the underground coal mining operators in Australia that the current testing regime tells them very little about the actual operational production of dust on the longwall face in relation to where it is produced or how to prevent this dust entering the atmosphere.

The current testing regime in Australia provides the mine tested with a single figure for respirable dust exposure levels for five samples taken over a minimum of four hours during a production shift. These figures only provide information relating to the exposure levels of the person sampled, relative to the 300 mm breathing zone described in AS2985, and does not provide any feedback on where the dust has come from or any other information that would allow the mine site to implement improvements in mitigation procedures should a non-compliance, or failure to statutory regulations occur.

For the testing regime in the USA the problem is more serious as a direct result of a known increase in CWP identifying 1000 new cases per year since 1984 and the recent findings of the Upper Big Branch disaster where autopsies revealed seventeen of the 24 victims’ autopsies (or 71%) had CWP. This compares with the national prevalence rate for CWP among active underground miners in the USA which is 3.2%, and the rate in West Virginia which is 7.6%.

Further, of the 17 UBB victims with CWP, five of them had less than ten years of experience as coal miners, while nine had more than 30 years of mining experience. At least four of the 17 worked almost exclusively at UBB. All but one of the 17 victims with CWP began working in the mines after the 2.0 mg coal mine dust limit was put in to effect in 1973. This was an exposure limit that was believed at the time sufficient to prevent black lung disease. This exposure limit has since been determined ineffective for protecting miners’ health (Davitt, et al., 2010).

CURRENT AUSTRALIAN DUST MONITORING PRACTICES

AS2985 and AS3640 clearly define the process to be used to determine personal exposure levels in coal mines.

According to NSW Coal Services Pty Ltd respirable dust testing analysis, there have been 18 900 respirable dust samples, including re sampling, taken in the period 1984-2007 (Mace 2008). Of these samples, it has been reported that there have been 1200 samples above the exposure limit for respirable dust in NSW, which represents less than 6.5% of total samples taken (Mace, 2008).

Coal Services background, regulations and testing methodology for NSW respirable dust sampling

Dust monitoring service

As reported by the NSW Coal Services Health (formerly the Joint Coal Board and JCB Health) Dust Monitoring Service (Cram, 2003) is quality accredited and has been the sole organization involved with personal dust monitoring in the NSW coal operators since the current regulations were gazetted in
March 1984. The service has the total support and acceptance of both management and unions. The specified limit for respirable dust other than quartz-containing dust is 3 mg of respirable dust/m$^3$ of air sampled. The specified limit for quartz-containing dust is 0.15 mg of respirable quartz/m$^3$ of air sampled. In NSW sample collection commences at the time of leaving the crib room at the start of the shift and ceases on arrival at the crib room at the end of the shift. The sampling period, if practicable should be not less than five hours. While it is the responsibility of mine management to meet the frequency of sampling required by the CMRA the Coal Services Health monitoring programs are structured in such a manner that management's obligations are fulfilled were possible.

The integrity of results is guaranteed by a Coal Services Health employee present in the workplace during the sampling shift recording such information as ventilation quantities, blocked sprays, operator location, water pressures or anything which may affect results. Results are used solely to identify problem areas which may exist and are not used at any time for punitive measures. Where areas of high dust concentrations are found to exist, efforts are directed to these areas in order to rectify the problems. These efforts in many cases involve management, union and coal services health initiatives. Results of the sampling are forwarded to the Colliery Manager, Senior Inspector of Coal Mines, United Mineworkers District Check Inspector and included in the coal services health dust database.

If the result of any sample exceeds the specified limit a re-sample must be taken within seven working days in similar circumstances to those existing when the sample was collected. If the resample still exceeds the specified limit the District Inspector of Coal Mines may, in writing, direct the Colliery Manager to carry out additional procedures to reduce the concentration of airborne dust.

The following information is extracted from the document titled “Airborne Dust in Coal Mines Respirable Dust and Quartz Inhalable Dust Coal Services Pty Ltd, 2008”:

**Sampling: method used to determine the respirable dust concentration of air in working places**

The approved sampling method adopted in the New South Wales coal operators is personal gravimetric sampling. In this method, respirable dust is collected from the breathing air very close to the nose and mouth of a mine worker and the amount of dust is then measured by weighing. The weight of fine dust drawn into the lungs gives the most accurate prediction of the likelihood of developing pneumoconiosis (being dusted). The samples are taken by means of a small battery powered pump worn by the mine worker as shown in Figure 1. The pump is connected with a piece of plastic hosing to a sampling unit (or cyclone) that is clipped to the individual's shirt. A steady stream of air is drawn through the sampling unit where the coarse dust is first removed and only the very fine respirable dust is collected on a filter and weighed.

**Purpose of dust sampling**

A comprehensive monitoring programme is continually being carried out to determine whether dust levels at every coal mine are kept below the approved limits and to protect the long term health of mine workers.
Working places sampled

As per the NSW Coal Mine Health and Safety Regulation (NSW CMHST) 2006, mine workers are sampled regularly. For longwall faces, sampling is carried out at intervals not exceeding six months on each producing shift. For continuous miner panels, sampling is carried out at intervals not exceeding 12 months on each producing shift. Other underground working places, open cuts, coal preparation plants, crusher and loading stations are all sampled at intervals not exceeding 12 months on only one production shift.

Dust results

Copies of all results are sent to the mine operator, inspector of coal mines and operators check inspector. Following a failed result, the mine manager informs the person who was sampled and there is an obligation under the (NSW CMHST) 2006 to take action to correct the situation. Coal Services, through the Standing Dust Committee (SDC), also maintains an overview of the results of the dust sampling programme in mines and where necessary advises the mine management on how to improve the situation. This SDC recommends the display of all results on the mine notice boards.

Limits for extended shifts

The current exposure limits for dust and quartz are based on a 40 h week (8 h shifts 5 d a week) over a 40 year working life. For working weeks greater than 40 h therefore the exposure limit needs to be lower. As a general rule the exposure limit can be adjusted by a factor calculated from the ratio of weekly exposure in a normal work cycle to the average weekly exposure in the extended cycle. The NSW coal services health and safety trust normally provide information on the extended shift exposure limit adjustment factors for coal mine dusts.

Results if a person is exposed to variable dust levels

The method of dust sampling is designed to give the average result for the duration of the shift taking into account periods of high and low exposure dust. The dilution effect of a worker being exposed to a non-contaminated atmosphere following a short but high exposure would therefore be beneficial to the worker such as job rotation during the shift. One of the key factors involved in the onset of lung dust disease is the total amount of coal dust or quartz that a person has inhaled during their working life. It is not based on whether the person has been exposed to a high level of dust in a single event on one part of a shift or due to a particular mining method.

Method used to determine the inhalable dust concentration

The gravimetric method used for respirable dust sampling is also used for inhalable dust sampling. The main difference is the sampling head which collects dust particles below 100 microns rather than only the very small respirable dust particles.

Figure 2 - Dust particle size comparison

Location and frequency of sampling inhalable dust

As per the NSW CMHST 2006 mine workers are sampled regularly. For longwall faces, sampling is carried out on each producing shift at intervals not exceeding 12 months. For continuous miner panels,
any part of a mine where cement products are being applied, other underground places including
crusher stations, open cuts and coal preparation plants are all sampled on one shift only at intervals not
exceeding 12 months.

**Respirable dust exposure limit in NSW coal mine**

The concentration of respirable dust should not exceed 2.5 mg/m$^3$ over the sampling period. The
concentration of respirable quartz dust should not exceed 0.12 mg/m$^3$ in underground coal mines and
not exceed 0.1 mg/m$^3$ in open cut coal mines and the surface parts of underground coal mines.

**Determination of limits**

The current coalmine exposure standard was determined after extensive research at a number of NSW
coalmines in the early 1980’s and these levels are constantly being reviewed in the light of new
research. There has been a steady decrease in dust disease patterns in NSW coalmines over the last
30 y and consequently the SDC considers that compliance with current exposure standards will provide
effective protection. The gravimetric measurement of respirable dust and quartz is the internationally
recognised technique for monitoring the dust exposure of coal mineworkers.

**Inhalable dust exposure limit in NSW coal mines**

Inhalable dust is the visible dust particles below 100 $\mu$m size. It is the concentration in milligrams of
inhalable dust per cubic metre (abbreviated to mg/m$^3$) of air, collected from the breathing zone (not
inside respirators or airstream helmets) of mine workers during their working shift. The concentration
of inhalable dust should not exceed 10 mg/m$^3$ in all coal mining operations.

**Background, regulations and testing methodology for Queensland respirable dust sampling
according to SIMTARS**

**Sampling Strategy**

Respirable dust samplers are distributed amongst a selection of personnel performing a range of
activities. Respirable dust monitoring involves workers wearing a personal sampling device consisting
of a constant flow sampling pump connected to a cyclone elutriator positioned within the breathing zone
(300 mm radius extending in front of the face and measured from the mid-point of a line joining the ears).
Operators are requested to wear these devices for the entire shift, or a period representative of their
normal duties.

**Sampling Techniques**

Techniques involved in the measurement of respirable dust followed those outlined in Australian
Standard 2985 - 2004 - Workplace Atmospheres - Method for sampling and gravimetric determination of
respirable dust and Simtars laboratory procedure LP00138. Respirable quartz (SiO2) concentrations
were subsequently determined using methods described in the NH and MRC (1984) document 9 on
quartz analysis and Simtars laboratory procedure LP001610. Constant flow sampling pumps
connected to a Simped’s cyclone elutriator containing a 37 mm 5 $\mu$m GLA-5000 filter are used to collect
respirable dust samples. A flow rate of 2.2 L/min is set prior to sampling and checked at the conclusion
of the sampling period using a calibrated flow meter.

Results derived using these methods represent time weighted average concentrations of respirable dust
encountered by operators during their normal working shift. With respect to respirable dust, a
time-weighted average implies a mass of respirable dust collected over a known time period (preferably
more than 4 h) from which an average mass/volume concentration is calculated. It is from time
weighted average concentrations that assessments are made with respect to acceptable health levels
and compliance with regulatory requirements.

**LIMITATIONS ASSOCIATED WITH THE CURRENT TESTING IN AUSTRALIA**

Calls from operators are advocating for a review of the current inhalable and respirable dust sampling
methods used in Australia and to investigate alternative sampling methodologies applicable to major
underground coal mining tasks, report on their validity within the codes, guidelines and standards and
propose a new testing methodology that better identifies atmospheric contamination caused by dust produced during the cutting cycle in longwall mining.

It has been suggested that with changes in the work routines of many Australian miners, the traditional way of sampling is no longer adequate. Further, operators believe that the current testing process is getting what are believed to be data errors arising from how sampling is being conducted not by over exposure to dust levels. Many samples are being contaminated leading to a failed result. The operators feel that rather than being recorded as a failure to the tested mines these should be deemed as invalid samples and quite rightly retested.

Mining operators have been investigating alternative ways of placing dust sampling units to eliminate contamination whilst still meeting the strict codes, guidance and standards applied to this area. Operators want to identify techniques that more accurately identify what specific work activities lead to specific results which will assist further in managing specific risks. Mining operators would like to include instantaneous or real time monitoring devices that may also assist with identification and eventual mitigation of airborne contaminant risks.

It has been suggested that there is a need to establish a database of best practice dust suppression techniques used by longwalls for the operators to peruse and use along with the management of sampling data. Currently the operators invest significant money in the sampling conducted by the regulatory regime but receive very little useful information on how to mitigate airborne contaminants. With the volume of data collected the operators should have a fairly accurate picture and understanding of the underground longwall work environment to help refine installed controls and measure their dust knockdown efficiency, but currently only receive single sample information with details recorded for a five sample batch not individual samples. The operators feel it would be better to have information on individual pieces of plant and equipment, tasks and activities and on the practices of crews or individuals. The operators would also like to see a review which will document standards of approach in the areas of dust control efficiencies to capture a definitive benchmark which will allow for a more scientific approach to the management of airborne contaminants.

Operators are suggesting a review of competency requirements for persons undertaking dust sampling, along with a review of the occupational exposure limits and identification of possible legislative shift adjustment criteria specifically designed to better reflect the continual changes in the mining environment.

**CURRENT USA DUST MONITORING PRACTICES**

According to the federal register, October 19 2010, Section 202(b) (2) of the federal mine safety and health act of 1977 requires each underground coal mine operator to continuously maintain the average concentration of respirable dust in the mine atmosphere during each shift to which each miner in the active workings is exposed at or below 2.0 μm of respirable dust per cubic meter of air. Section 205 required that when coal mine dust contains more than five percent quartz, the respirable coal mine dust standard must be reduced according to a formula prescribed by NIOSH.

The federal register further states that under United states of America Mines Safety and health Administration’s’ (MSHA’s) existing standards, mine operators are required to collect bimonthly respirable dust samples and submit them to MSHA for analysis to determine compliance with applicable respirable dust standards (compliance samples). If compliance samples do not meet the requirements of the applicable dust standard, MSHA issues a citation for a violation of the standard and the operator is required to take corrective action to lower the respirable dust concentration to meet the standard.

Additionally, according to the federal register, the operator must collect additional respirable dust samples during the time established in the citation for abatement of the hazard or violation (abatement sampling). Underground coal mine operators must collect and submit two types of samples during bimonthly sampling periods: (1) Designated Occupation (DO) samples taken for the occupations exposed to the greatest concentrations of respirable dust in each mechanised mining unit (DOs are specified in s.70.207); and (2) Designated Area (DA) samples collected at locations appropriate to best measure concentrations of respirable dust associated with dust generation sources in the active working of the mine (s.70.208). The operator’s approved ventilation system and methane and dust control plan, required in existing 30 CFR part 75, must show the specific locations in the mine designated for taking the DA samples. In addition, mine operators take respirable dust samples for part 90 miners (s.90.207
and s.90.208). Compliance determinations are based on the average concentration of respirable dust measured by five valid respirable dust samples taken by the operator during five consecutive normal production shifts or five normal production shifts worked on consecutive days (multiple-shift samples). Compliance determinations are also based on the average of multiple measurements taken by the MSHA inspector over a single shift (multiple, single-shift samples) or on the average of multiple measurements obtained for the same occupation on successive days (multiple-shift samples).

LIMITATIONS ASSOCIATED WITH THE CURRENT TESTING IN THE USA

According to the federal register, October 19 2010, exposure to respirable coal mine dust can cause lung diseases including coal workers’ pneumoconiosis (CWP), emphysema, silicosis, and chronic bronchitis, known collectively as black lung. These diseases are debilitating, incurable, and can result in disability, and premature death. While considerable progress has been made in reducing the respirable coal mine dust levels, miners continue to develop black lung.

Based on a recent draft report from the USA National Institute for Occupational and Health (NIOSH, 2010), the prevalence rate of black lung is increasing in the nation’s coal miners; even younger miners are showing evidence of advanced and seriously debilitating lung disease, as shown in Figure 3.

The report continues further details that in the last decade, death certificates list coal workers’ pneumoconiosis, commonly called black lung disease, as a cause in more than 10 000 deaths. Black lung disease is caused by inhaling coal mine dust. It results in scarring of the lungs, emphysema, shortness of breath, disability, and premature death. The prevalence of black lung disease decreased by about 90% from 1969 to 1995 after the enactment of the coal mine health and safety act. Unfortunately, since 1995, the prevalence of black lung among those who have participated in the coal workers’ health surveillance program and who have been coal miners for more than 20 years has more than doubled. Severe and advanced cases of lung disease occur currently in young underground miners as young as 39. Identification of advanced cases among miners under age 50 is of particular concern, as they were exposed to coal-mine dust in the years after the 1969 federal legislation had mandated disease-prevention measures. An increased risk of pneumoconiosis has been associated with work in certain mining jobs, in smaller mines, in several geographic areas, and among contract miners (CDC).

![Figure 3 - Percentage of miners examined in the NIOSH coal workers’ X-ray surveillance programme with Coal workers’ pneumoconiosis by tenure in mining (NIOSH, 2010)](image)

CONCLUSIONS

Both Australia and the USA have identified that the currently installed controls for the mitigation and removal of harmful coal dust from the underground mining environment have proven, in the first instance, to be hard to measure in terms of the success in mitigating airborne contaminants, and secondly, in the case of the USA, have failed to remove the risk of underground workers contracting CWP from their working environment.
In the case of the USA, The federal register, October 19, 2010 suggests that a reduction in the current exposure levels from 2 mg/m$^3$ to 1 mg/m$^3$ be implemented as the only practical solution to reducing the alarming increase in CWP amongst younger underground workers.

Along with the proposed reduction in exposure levels, several provisions in the proposed rule change, that is, basing noncompliance determinations on single shift sampling, sampling of extended work shifts to account for occupational exposures greater than 8 h per shift, and changing the definition of normal production shift, would singularly lower coal miners’ exposure to respirable dust. For example, MSHA’s Quantitative Risk Assessment (QRA) estimates the reduction in health risks when two provisions of the proposed rule are implemented-the proposed respirable dust limit and single shift sampling. The QRA shows that these two proposed provisions would significantly reduce the risks of CWP, severe emphysema, and death from non-malignant respiratory disease (NMRD). The proposed rule change would potentially create 50 fewer cases of severe emphysema and 15 fewer deaths due to NMRD per thousand exposed cutting machine operators. The other provisions in the proposed rule would further reduce health risks to miners. Cumulatively, the proposed provisions would reduce the continued risks that coal miners face from exposure to respirable coal mine dust and would further protect them from the debilitating effects of occupational respiratory disease.

In Australia, it has been suggested that the traditional way of sampling is no longer adequate. Operators members believe that the current testing process is getting sample failures due to reasons other than high exposure levels, for example, uneven distribution of dust on the filter paper and pumps not running a full shift, and rather than being recorded as a failure to the tested mines these should be deemed as invalid samples and quite rightly retested.

Mining operators also want to identify techniques that more accurately identify what specific work activities lead to specific results which will assist further in managing specific risks. Mining operators members would also like to look at instantaneous measuring devices that may also assist with identification and eventual mitigation of airborne contaminant risks.

There is a need to establish a database of best practice dust suppression techniques used by longwalls for the operators to peruse and use along with the management of sampling data. With the volume of data collected the operators should have a fairly accurate picture and understanding of the underground longwall work environment to help refine installed controls and measure their dust knockdown efficiency, but currently only receive single sample information with details recorded for a five sample batch not individual samples. The operators feel it would be better to have information on individual pieces of plant and equipment, tasks and activities and on the practises of crews or individuals. The operators would also like to see a review which will document standards of approach in the areas of dust control efficiencies to capture a definitive benchmark which will allow for a more scientific approach to the management of airborne contaminants.

Finally, it has been suggested that a review of competency requirements for persons undertaking dust sampling be undertaken and that a review of the occupational exposure limit is covered and suggested legislative shift adjustment criteria is recommended to better reflect the continual changes in the mining environment.

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EXPERIENCES OF THE INSTITUTE OF OCCUPATIONAL MEDICINE FOAM RESPIRABLE SAMPLER USE IN MINES

Bharath Belle

ABSTRACT: The mining industry worldwide spends a significant amount of human and financial resources in sampling of safety and health hazards for ensuring adequate control measures. Most mining countries carry out personal exposure monitoring for respirable dust. Unlike Australia, very few countries spend their resources in sampling of inhalable dust in mining industry. Over the years, size-selective sampling curve and instruments which replicate human inhalation have also changed. In addition, since its inception in 1920s, the recommended occupational exposure limits of a substance have varied significantly between mining countries worldwide. This paper discusses the experiences of introducing newly available monitoring instruments through laboratory and field evaluation. The institute of occupational medicine respirable foam sampler was evaluated in coal, diamond, gold and platinum mines. For comparison purposes, Higgins-Dewell type cyclone that conforms to the new size-selective curve with a D50 of four microns was used as a “true” reference sampler. For the laboratory study, the two samplers were exposed to two types of dust, viz. coal and sandstone briquette dust with a quartz content of 50.6%.

Based on the results of the laboratory study, the correlation coefficient (r) between the foam and reference sampler was found to be 0.79 and 40% underestimation in measured values by the foam sampler (p-value of 0.000). Field evaluations of side-by-side foam and reference samplers in coal, gold, platinum and diamond mines, showed a poor non-linear relationship (r = 0.67) for a wide range of dust levels. From the non-linear regression equation, on average, the foam respirable sampler underestimated the dust levels by approximately 48% for a compliance level of 2 mg/m³. For increased dust levels, the underestimation of the measured dust levels by the foam sampler also increased, which led to the sampler being unsuitable for use during engineering control purposes. In overall, the foam sampler failed to meet the NIOSH accuracy criteria and was not pursued further for use in South African mines. Study suggests sufficient and prior due-diligence of any new instruments or methodologies to industry wide applications. Any modifications to sampling methodology or introduction of new instruments must ensure that the collected exposure data is relevant for continued development of long term dose-response curves and understand potential level of risks.

INTRODUCTION

Monitoring of dust in mines is an important task and requires reliable instruments. There are various means of measuring dust, viz., personal sampling, area sampling and engineering sampling. Knowledge of routine dust exposure levels can help workers’ and industry focus on protection of workers respiratory health. Against this background, the search for an improved or alternative instrument or sampling methodology that will measure occupational exposure more accurately and more reliably is continuing. This paper shares experience of introducing a new instrument for the exposure monitoring that is relevant to similar industries worldwide.

Past studies have suggested that the personal sampling method is the most suitable method for assessing, and most representative of, the worker’s dust exposure (Leidel, et al., 1977; Kissell and Sacks, 2002). A decade ago, an effort was made to evaluate the newly available instruments for personal exposure assessment in the typically harsh conditions of South African underground mines.

Dust sampling is pursued in mines to understand the level of risk associated with exposure to hazards. Figure 1 provides a typical fraction of dust data in a British colliery taken up by exposed humans during breathing (Gibson, et al., 1987).

It was noted that the inspirable dust mass of 38.4 mg contained 6.6 mg of respirable dust, 3.7 mg of tracheobronchial dust, 13.5 mg of thoracic dust.

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The following conclusions were drawn from the review of available worldwide literature on newly developed gravimetric sampling instruments, and their evaluation by researchers:

- Some of the “new” gravimetric-type instruments, such as the Institute of Occupational Medicine (IOM) sampler, do not yield any additional information on respirable dust or help the mining industry as the existing gravimetric samplers are capable of collecting the necessary personal exposure data;
- The errors associated with personal sampling are usually the result of the worker’s body movements, instrument portability and other sampler-handling mistakes. Therefore, ultimately, an accurate sampling instrument that would be able to cope with the worker’s usual production demands is required for the harsh environment of the mining industry.

In order for the introduction of the new dust-monitoring instruments for personal sampling in underground mines to be accepted by the stakeholders, they were required to meet the basic requirements (criteria) as outlined below:

- They must be intrinsically safe for use in underground mines;
- They must sample according to the accepted size-selective criteria at specified flow rates;
- They must meet the ± 25 % NIOSH accuracy criterion;
- They should preferably use a different quick analysis procedure to the weighting method that is currently used;
- They must be robust enough to withstand the harsh conditions prevailing in mines;
- They must be compact and portable for personal sampling;
- They must offer the possibility of collecting dust samples for further quartz analysis.

INSTITUTE OF OCCUPATIONAL MEDICINE FOAM SAMPLER

The IOM foam sampler, as shown in Figure 2, was designed by Mark and Vincent (1986) and collects dust samples by the gravimetric method. It has a 15 mm diameter inlet orifice. Aerosol is aspirated into the IOM sampler at a flow rate of 2.0 L/min. Particles aspirated into the inlet are either collected by a 25 mm filter or deposited on the inside surfaces of an internal two-piece cassette. The original IOM foam sampler that was modified by Health and Safety Laboratory is already in widespread use above ground for sampling inhalable dust (MDHS 14/2, 1997). The cassette of the sampler has been modified to incorporate two size-selective foams in front of the usual filter; this means that the sampled inhalable dust is further subdivided into thoracic and respirable dust fractions, i.e. all three fractions are sampled simultaneously. The three dust fractions can be quantified by analysing the foams and filter separately. The respirable dust collected on the filter can be further analysed for quartz content.
The IOM sampling head weighs only 20 g and as in the case of the cyclone, an intrinsically safe pump is required. Historically, as no study had been carried out in South Africa, the respirable foam sampler was considered for evaluation as a personal sampler in South African mines. Unlike Australia, very few countries spend their resources in sampling of inhalable dust in the mining industry.

**Figure 2 - IOM foam sampler**

Area sampling performance of six inhalable aerosol samplers was studied using monodisperse, solid particles by Li et al. (2000). The study reported that the area sampling performance of the foam sampler is highly dependent on wind orientation, wind speed and particle size. When the measured sampling efficiency was compared with the inhalable convention, the IOM sampler over sampled the large particles (>20 µm).

**Laboratory evaluations**

The laboratory Polley dust duct of the National Institution of Occupational Health (NIOH) in Johannesburg is shown in Figure 3. The experimental design, the laboratory tests and the data analysis procedures are given elsewhere (Belle, 2002). For all laboratory comparison purposes, the Government Mining Engineer (GME) approved South African Higgins-Dewell type cyclone (GME#GE05) was used as a 'true' reference sampler. Figure 4 shows a typical side-by-side positioning of samplers in the laboratory test chamber. Tests were carried out with instrument pairs exposed to coal and sandstone briquette dust.

During both the laboratory and field trials, the foam respirable sampler operated at 2.0 L/min and the HD type cyclone operated at 2.2 L/min. They were positioned side by side inside the dust chamber and exposed to the coal and sandstone briquette dust.

**Figure 3 - Photo of the laboratory Polley dust duct**

**Figure 4 - Laboratory test table for samplers**

**Field evaluations - test mines and instrumentation**

The foam sampler was compared with HD type South African cyclones. It was assumed that the cyclone samplers gave negligible errors and a "true" measurement of personal dust concentration. In order to carry out the personal sampling in mines, a sampling harness was prepared and the dust monitors were worn in a specific position consistently in all the test mines (Figure 5).

The left lapel of the harness contained the reference sampler and the foam sampler. A summary of the sampled mines and individual sampling locations is given in Table 1. The sampled gold, platinum, coal and diamond mines are unique with regard to their extremely challenging environmental conditions. Some of the mines used diesel-operated equipment and machinery. The test procedure is described in the underground test protocol and was discussed by Belle (2002).
Establishing an accuracy criterion

For all comparison purposes, the dust level measured by the HD type cyclone sampler was considered the “true” concentration. Therefore, the concentration ratio of the “evaluation instrument” to the reference instrument (in this study, the HD sampler) was calculated. If the variability in the concentration ratio is small, then one can consider accepting the “evaluation instrument” for further use. The concentration ratio is analogous to the bias as described by Kennedy et al. (1995). The relative standard deviation (RSD) was calculated from the standard deviation and the mean concentration ratio.

Accuracy criteria of ±25% analogous to the NIOSH instrumentation accuracy criterion (Kennedy, et al., 1995) were used. For normally distributed data, 95% of the measurements fall within the range ±1.96s, where s is the standard deviation. For example, assuming that the mean is 100, for the criterion of ±25%, then 1.96s = 25 or s = 12.7. Because the mean is 100, the standard deviation divided by the mean (called RSD or CV) is 0.127. Thus, the ±25% accuracy criterion (NIOSH) is met at RSD = 0.127 or less.

RESULTS OF PAIR-WISE COMPARISON OF FOAM SAMPLER AND HIGGINS-DEWELL CYCLONES

Laboratory results

This section of the paper discusses the results of the laboratory evaluation of the foam sampler. The relationship between the measured values obtained from the side-by-side foam and HD reference samplers during the laboratory evaluation for both types of dust is shown in Figure 6.

The correlation coefficient (r) between the two samplers is 0.79. The plot shows a nominal linear relationship and there is a significant difference between the IOM and reference samplers and clearly...
indicates the underestimation by the foam sampler for various measured dust levels. For coal dust, the average measured levels using the reference and foam samplers are 7.23 mg/m\(^3\) and 3.09 mg/m\(^3\) respectively for the test conditions. Similarly, for sandstone dust, the average measured levels using the reference and foam samplers are 11.29 mg/m\(^3\) and 5.41 mg/m\(^3\) respectively. From the regression line it can be inferred that the foam sampler underestimates the respirable dust levels by about 36% at a compliance level of 2 mg/m\(^3\), but at greater dust levels the underestimation of measured dust levels is much higher.

Figure 6 - Laboratory relationship between side-by-side foam and HD reference samplers

Statistical analyses: Laboratory data

Table 2 shows summary statistics of respirable dust values obtained from the side-by-side comparison of foam and reference samplers when exposed to coal and sandstone briquette dust. From the summary statistics table (Table 2) it can be seen that there is no clear relationship between the accuracy of an instrument and its measured concentration levels. Overall, the CV of the ratio between the sampler dust levels failed to meet the NIOSH accuracy criteria.

Table 2 - Summary of the IOM correction factors

<table>
<thead>
<tr>
<th>Dust</th>
<th>SP</th>
<th>MRSC</th>
<th>MRC</th>
<th>NT</th>
<th>SD</th>
<th>RSD or CV (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>C, S</td>
<td>HD-HD</td>
<td>10.80</td>
<td>1.035</td>
<td>15</td>
<td>0.074</td>
<td>7.15</td>
</tr>
<tr>
<td>C</td>
<td>IOM-HD</td>
<td>7.232</td>
<td>0.442</td>
<td>8</td>
<td>0.133</td>
<td>30.09</td>
</tr>
<tr>
<td>S</td>
<td>IOM-HD</td>
<td>11.293</td>
<td>0.493</td>
<td>8</td>
<td>0.098</td>
<td>19.87</td>
</tr>
<tr>
<td>Overall</td>
<td>IOM-HD</td>
<td>9.263</td>
<td>0.468</td>
<td>16</td>
<td>0.116</td>
<td>24.78</td>
</tr>
</tbody>
</table>

C: Coal; S: Sandstone; SP: Sample Pair; MR: Mean Reference Sampler Concentration; MRC: Mean Ratio of Concentrations; NT: No. of Tests

A paired t-test (Table 3) was performed on the set of sample pair data to determine whether there was a statistical difference in the loge-transformed (normally distributed) concentration levels between the sampler pairs. A paired t-test of hypotheses was developed to compare the mean concentration levels measured with two sampling instruments (\(\mu A\) and \(\mu B\)). The null and alternative hypotheses for the sample pairs tested were:

\[
H_0: \mu A = \mu B \\
H_1: \mu A \neq \mu B
\]

In the paired t-test, hypothesis \(H_0\) states that the mean dust concentration levels from both samples (\(\mu A\) and \(\mu B\)) are equal. On the other hand, the alternative hypothesis states that the two samplers in fact measure different mean concentration levels. Hypothesis tests were carried out for the data set.

In this study, a cut-off p-value of 0.05 was used (95% confidence level). From the analysis table it was observed, with various degrees of freedom, the large p-value (>0.05) suggesting that the measured mean concentration levels are consistent with the null hypothesis, \(H_0: \mu A = \mu B\), that is, the dust concentration measured by foam sampler and reference sampler is not affected at the 95% level of confidence.

From Table 3, for both dust types, there was a significant difference in the measured dust levels between the reference HD sampler and the foam sampler.
Table 3 - Results of paired t - test (on transformed values)

<table>
<thead>
<tr>
<th>Dust</th>
<th>SP</th>
<th>NT</th>
<th>95 % LCL</th>
<th>95 % UCL</th>
<th>p-value</th>
</tr>
</thead>
<tbody>
<tr>
<td>C</td>
<td>IOM-HD</td>
<td>8</td>
<td>0.618</td>
<td>1.08</td>
<td>0.000</td>
</tr>
<tr>
<td>S</td>
<td>IOM-HD</td>
<td>8</td>
<td>0.566</td>
<td>0.880</td>
<td>0.000</td>
</tr>
<tr>
<td>Total</td>
<td>IOM-HD</td>
<td>16</td>
<td>0.660</td>
<td>0.915</td>
<td>0.000</td>
</tr>
</tbody>
</table>

C: Coal; S: Sandstone; SP: Sampler Pair; LCL: Lower Confidence Level; UCL: Upper Confidence Level; NT: No. of Tests

Underground results

The relationship between the measured values obtained from the side-by-side foam and reference samplers in coal mines is shown in Figure 7. The correlation coefficients (r) between the two samplers in coal mine A and coal mine B are 0.77 and 0.47 respectively. A combined plot of the two coal mine data sets (r = 0.52) and samplers show poor linearity when measured in coal mines, despite there being less scatter. The plot indicates that, on average, the foam sampler underestimates the respirable coal dust levels by more than 50%.

The relationship between the dust values obtained from the side-by-side foam and the reference sampler in two gold mines is shown in Figure 8. The correlation coefficients (r) between the two cyclones in gold mine A and gold mine B are 0.86 and 0.95 respectively. A combined plot of the two gold mine data sets (r = 0.76) and the two samplers show comparatively reasonable linearity when measured, with wide scatter. The plot indicates that, on average, the foam sampler underestimates the measured respirable dust level by approximately 35%.

Similarly, the relationship between the concentration values obtained from the side-by-side foam and reference samplers during the field trials in a platinum mine is shown in Figure 9. The correlation coefficient (r) between the two cyclones in the platinum mine is 0.58. The two samplers show poor linearity, with wide scatter. On average, the plot indicates that the foam sampler underestimates the measured respirable coal dust concentration by approximately 13% at low concentration levels.

The relationship between the concentration values obtained from the side-by-side foam sampler and reference samplers during the field trials in a diamond mine is shown in Figure 10. The correlation coefficient (r) between the two cyclones in the diamond mine is 0.83. The measured dust levels had a wide range and at compliance levels the foam sampler underestimates the measured respirable coal dust concentration by more than 60%. From the plot we observe that at higher dust concentrations, the foam sampler underestimates to a larger extent.

In order to determine the relationship between the dust values obtained from the side-by-side personal foam and reference samplers during the field trials in hard rock mines (gold, platinum and diamond), the relationship was plotted as shown in Figure 11. The correlation coefficient (r) between the two cyclones in all hard rock mines is 0.67, showing a poor non-linear relationship between the samplers. The combined scatter plot of all mine data (Figure 12) again shows a poor non-linear relationship (r=0.67) between the foam and SA samplers measured in various mine types with a wide range of measured dust levels.

From the non-linear regression equation we can deduce that, on average, the foam sampler underestimates the measured respirable dust levels by approximately 48% for a compliance level of...
2 mg/m³. As the dust levels increase, the underestimation of the measured dust levels by the foam sampler also increases, which makes the sampler unsuitable for use even for engineering control purposes. Also at low concentrations, the foam sampler measures higher than the reference sampler. From the plot we can observe that all underground measurement values included both compliance and non-compliance levels for the sampling period and that the scatter was wide for both low and high dust concentrations.

Figure 9 - Relationship between side-by-side personal foam and HD Reference sampler in a platinum mine

Figure 10 - Relationship between side-by-side personal foam and HD Reference sampler in a diamond mine

Figure 11 - Relationship between side-by-side personal foam and HD Reference sampler in non-coal mines (gold, platinum, diamond)

Figure 12 - Relationship between side-by-side personal foam and HD Reference sampler from all mines (gold, platinum, diamond and coal)

Statistical analyses - mine data

Table 4 shows summary statistics of the respirable dust concentration values obtained from the side-by-side comparison of the foam and reference samplers measured in coal, gold, platinum and diamond mines by three personnel. The CV is the ratio of standard deviation and mean value expressed as a percentage. From the summary table (Table 4) it is observed that there is no clear relationship between accuracy and the measured concentration levels. Overall, the foam sampler failed to meet the NIOSH accuracy criteria.

All the dust concentration data for each sample set were tested for Anderson-Darling normality and it is evident that the data do not follow a normal distribution. Preliminary data analysis indicated that loge–transformed data gave an improved fit of the normal distribution. Therefore, for the statistical analysis, loge(Ha) and loge(Hb) were compared (paired t-test). The subscripts, Ha (SA sampler) and Hb (test sampler), are the dust concentration values measured using the identified personal sampling instruments in the sample pair (random) at various test mines. Hypothesis tests were carried out at each of the mines to test the sampling environment (gold, diamond, platinum and coal). The null and alternative hypotheses for the tested sample pairs were:

\[
\begin{align*}
H_0: \mu_{diff} & = 0 \\
H_a: \mu_{diff} & \neq 0
\end{align*}
\]

In the paired t-test, hypothesis H0 states that the mean difference in concentration values (transformed values) between side-by-side personal instrument pairs is equal to zero. On the other hand, the alternative hypothesis states that the two personal dust-monitoring instruments positioned side by side in fact measured different mean concentration levels or the difference was not equal to zero. For this
research work, a standard 95% confidence level was chosen. The results of the paired t-test statistical analyses are given in Table 5.

**Table 4 - Summary of the correction factors for the foam and Reference samplers in all mines**

<table>
<thead>
<tr>
<th>Mine Type</th>
<th>Person</th>
<th>MRSC</th>
<th>Ratio of IOM/SA Conc</th>
<th>NT</th>
<th>SD</th>
<th>RSD or CV (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>C1</td>
<td>B</td>
<td>2.005</td>
<td>0.577</td>
<td>5</td>
<td>0.146</td>
<td>25.30</td>
</tr>
<tr>
<td></td>
<td>L</td>
<td>1.697</td>
<td>0.634</td>
<td>5</td>
<td>0.206</td>
<td>32.49</td>
</tr>
<tr>
<td></td>
<td>J</td>
<td>1.431</td>
<td>0.641</td>
<td>5</td>
<td>0.160</td>
<td>24.96</td>
</tr>
<tr>
<td></td>
<td>Total</td>
<td>1.711</td>
<td>0.617</td>
<td>15</td>
<td>0.162</td>
<td>26.26</td>
</tr>
<tr>
<td>C2</td>
<td>B</td>
<td>3.516</td>
<td>0.297</td>
<td>5</td>
<td>0.112</td>
<td>37.71</td>
</tr>
<tr>
<td></td>
<td>L</td>
<td>4.836</td>
<td>0.345</td>
<td>5</td>
<td>0.175</td>
<td>50.72</td>
</tr>
<tr>
<td></td>
<td>J</td>
<td>2.504</td>
<td>0.421</td>
<td>5</td>
<td>0.155</td>
<td>36.73</td>
</tr>
<tr>
<td></td>
<td>Total</td>
<td>3.618</td>
<td>0.354</td>
<td>15</td>
<td>0.149</td>
<td>42.09</td>
</tr>
<tr>
<td>Overall</td>
<td></td>
<td>2.665</td>
<td>0.486</td>
<td>30</td>
<td>0.203</td>
<td>41.77</td>
</tr>
<tr>
<td>G1</td>
<td>B</td>
<td>0.533</td>
<td>1.763</td>
<td>5</td>
<td>0.333</td>
<td>18.89</td>
</tr>
<tr>
<td></td>
<td>L</td>
<td>0.483</td>
<td>2.072</td>
<td>5</td>
<td>1.248</td>
<td>60.23</td>
</tr>
<tr>
<td></td>
<td>J</td>
<td>0.721</td>
<td>1.457</td>
<td>5</td>
<td>0.233</td>
<td>15.99</td>
</tr>
<tr>
<td></td>
<td>Total</td>
<td>0.502</td>
<td>1.764</td>
<td>15</td>
<td>0.748</td>
<td>42.40</td>
</tr>
<tr>
<td>G2</td>
<td>B</td>
<td>0.752</td>
<td>2.219</td>
<td>5</td>
<td>2.057</td>
<td>92.69</td>
</tr>
<tr>
<td></td>
<td>L</td>
<td>0.953</td>
<td>2.072</td>
<td>5</td>
<td>2.673</td>
<td>129.0</td>
</tr>
<tr>
<td></td>
<td>J</td>
<td>0.994</td>
<td>1.642</td>
<td>5</td>
<td>1.724</td>
<td>104.9</td>
</tr>
<tr>
<td></td>
<td>Total</td>
<td>0.899</td>
<td>1.978</td>
<td>15</td>
<td>2.041</td>
<td>103.2</td>
</tr>
<tr>
<td>Overall</td>
<td></td>
<td>0.701</td>
<td>1.870</td>
<td>30</td>
<td>1.514</td>
<td>80.96</td>
</tr>
<tr>
<td>P</td>
<td>B</td>
<td>0.450</td>
<td>2.221</td>
<td>8</td>
<td>0.886</td>
<td>39.89</td>
</tr>
<tr>
<td></td>
<td>L</td>
<td>0.399</td>
<td>1.877</td>
<td>6</td>
<td>0.489</td>
<td>26.05</td>
</tr>
<tr>
<td></td>
<td>J</td>
<td>0.434</td>
<td>1.876</td>
<td>7</td>
<td>0.692</td>
<td>36.88</td>
</tr>
<tr>
<td></td>
<td>Total</td>
<td>0.431</td>
<td>2.008</td>
<td>21</td>
<td>0.712</td>
<td>35.46</td>
</tr>
<tr>
<td>D</td>
<td>B</td>
<td>4.480</td>
<td>0.401</td>
<td>5</td>
<td>0.250</td>
<td>62.34</td>
</tr>
<tr>
<td></td>
<td>L</td>
<td>3.733</td>
<td>0.324</td>
<td>5</td>
<td>0.137</td>
<td>42.28</td>
</tr>
<tr>
<td></td>
<td>J</td>
<td>2.212</td>
<td>0.512</td>
<td>5</td>
<td>0.355</td>
<td>69.33</td>
</tr>
<tr>
<td></td>
<td>Total</td>
<td>3.475</td>
<td>0.412</td>
<td>15</td>
<td>0.256</td>
<td>62.13</td>
</tr>
<tr>
<td>MM</td>
<td>Total</td>
<td>1.245</td>
<td>1.583</td>
<td>66</td>
<td>1.267</td>
<td>80.03</td>
</tr>
<tr>
<td>AM</td>
<td>Total</td>
<td>1.689</td>
<td>0.936</td>
<td>96</td>
<td>1.172</td>
<td>125.2</td>
</tr>
</tbody>
</table>

C: Coal; G: Gold; P: Platinum; D: Diamond; MM: Metal Mines; AM: All Mines; MRSC: Mean Reference Sampler Concentration; NT: No. of Tests

From Table 5 it is observed that, for all test mines, there was a significant difference in measured dust levels between the reference and foam samplers. A paired t-test was performed on the combined data of two dust monitors to determine whether there was a statistical difference in the results obtained from the reference sampler and the other monitors tested. The foam sampler showed rejection of the hypothesis that the dust readings measured by the two samplers side by side are significantly affected at the 95% level of confidence.

The measured dust concentration ratios between the data from the test samplers (Reference Sampler and foam sampler) were used to perform an analysis of variance (ANOVA). A discussion of the ANOVA models and their underlying assumptions can be found in any of the standard books on statistics. From the results of the ANOVA, the following relevant conclusions for foam sampler can be deduced:

- The effect of mine (dust) type on the concentration ratio between the two side-by-side monitors positioned in the breathing zone of the workers is highly significant. Apart from the dust type encountered in the individual test mines, the environmental conditions (such as humidity and temperature and thus worker’s orientation to wind directions) and conditions such as continuous sweating and discomfort may have contributed to variations in the measured dust levels.

- The foam sampler's performance is not significantly affected by the sampling individual (p=0.323) as all of them were exposed to the same mine environmental conditions.
### Table 5 - Results of paired t-test (on transformed values)

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Mine Type</th>
<th>HSA-IOM</th>
</tr>
</thead>
<tbody>
<tr>
<td>95% LCL</td>
<td>Gold</td>
<td>-0.643</td>
</tr>
<tr>
<td></td>
<td>Platinum</td>
<td>-0.794</td>
</tr>
<tr>
<td></td>
<td>Diamond</td>
<td>0.736</td>
</tr>
<tr>
<td></td>
<td>Coal</td>
<td>0.643</td>
</tr>
<tr>
<td>95% UCL</td>
<td>Gold</td>
<td>-0.175</td>
</tr>
<tr>
<td></td>
<td>Platinum</td>
<td>-0.491</td>
</tr>
<tr>
<td></td>
<td>Diamond</td>
<td>1.33</td>
</tr>
<tr>
<td></td>
<td>Coal</td>
<td>0.994</td>
</tr>
<tr>
<td>t-statistic</td>
<td>Gold</td>
<td>-3.58</td>
</tr>
<tr>
<td></td>
<td>Platinum</td>
<td>-8.84</td>
</tr>
<tr>
<td></td>
<td>Diamond</td>
<td>7.49</td>
</tr>
<tr>
<td></td>
<td>Coal</td>
<td>9.53</td>
</tr>
<tr>
<td>P–value</td>
<td>Gold</td>
<td>0.001</td>
</tr>
<tr>
<td></td>
<td>Platinum</td>
<td>0.000</td>
</tr>
<tr>
<td></td>
<td>Diamond</td>
<td>0.000</td>
</tr>
<tr>
<td>Hypothesis (accept or reject)</td>
<td>Gold</td>
<td>Reject</td>
</tr>
<tr>
<td></td>
<td>Platinum</td>
<td>Reject</td>
</tr>
<tr>
<td></td>
<td>Diamond</td>
<td>Reject</td>
</tr>
<tr>
<td></td>
<td>Coal</td>
<td>Reject</td>
</tr>
<tr>
<td>Sample size</td>
<td>Gold</td>
<td>30</td>
</tr>
<tr>
<td></td>
<td>Platinum</td>
<td>21</td>
</tr>
<tr>
<td></td>
<td>Diamond</td>
<td>15</td>
</tr>
<tr>
<td></td>
<td>Coal</td>
<td>30</td>
</tr>
</tbody>
</table>

### CONCLUSIONS

An extensive laboratory and field evaluation of respirable foam sampler positioned side by side of a HD reference sampler. The HD type sampler was used as a ‘true’ reference sampler operated according to the CEN/ISO/ACGIH size-selective curve. Field evaluation of the instruments as personal dust-monitors, side by side in the breathing zone, was carried out in gold, platinum, coal and diamond mines of South Africa. The results of the evaluation are relevant to Australian mines in the context of practices of personal dust exposure monitoring.

Based on the results of the laboratory study, the correlation coefficient (r) between the foam and reference sampler was found to be 0.79 and 40% underestimation in measured values by the foam sampler (p-value of 0.000). Field evaluations of side-by-side personal foam and reference samplers in coal, gold, platinum and diamond mines, showed a poor non-linear relationship (r = 0.67) for a wide range of dust levels. From the non-linear regression equation, on average, the foam respirable sampler underestimated the dust levels by approximately 48% for a compliance level of 2 mg/m³. For increased dust levels, the underestimation of the measured dust levels by the foam sampler also increased, which led to the sampler being unsuitable for use during engineering control purposes. In overall, the foam sampler failed to meet the NIOSH accuracy criteria and was not pursued further for use in South African mines.

Mining industry worldwide spends significant amount of resources in sampling safety and health hazards for ensuring adequate control measures. Most mining countries sample for respirable dust, however sampling of inhalable dust in mining industry is carried out in very few countries like Australia. Over the years, size-selective sampling curve and instruments which replicate human inhalation have also
changed. In addition, the recommended compliance limits of a substance have varied significantly between various mining countries worldwide.

This evaluation experience suggests sufficient and prior evaluation of any new instruments for industry wide applications. Any modifications to sampling methodology or introduction of new instruments must ensure that the exposure data collected is relevant for continued development of long term dose-response curves and to understand potential level of risks.

Over the years, exposure limits of substances have changed and exposure assessment or compliance determination is becoming more confusing and complex due to terminologies used (for example, Indicative OELVs), instrument used, exposure period, work status. Finally, what is quintessential is the consistent approach to sampling, instruments used, availability of measurement relationships between past and new instruments that will be readily available for correcting systematic biases in sampling which in the longer term assists in exposure determination and for continued formulation of dose-response relationships.

ACKNOWLEDGEMENTS

The author hopes that the knowledge sharing of relevant findings presented in this paper will enhance complex issues of dust monitoring, challenges of introducing a new dust monitoring instrument and need for data for continued development of dose-response relationships so as to improve safety and health of workers. Various inputs of all relevant parties are clearly acknowledged.

REFERENCES


Kissell, F N and Sacks, H K, 2002. Inaccuracy of area sampling for measuring the dust exposure of mining machine operations in coal mines. SME, USA.


INVESTIGATION OF SPONTANEOUS HEATING ZONES AND PROACTIVE INERTISATION OF LONGWALL GOAF IN FENGHUANGSHAN MINE

Ting Ren¹, Zhongwei Wang¹, Jan Nemcik¹, Naj Aziz¹ and Jianming Wu²

ABSTRACT: To understand the spatial distribution of spontaneous combustion zones under a Y ventilation scheme, field tests and numerical modelling studies were carried out on a longwall face in Fenghuangshan mine. Computational fluid dynamics models were developed and base model results validated using tube bundle gas monitoring data. A three dimensional high oxygen concentration zone where spontaneous combustion was most likely to occur was predicted behind the longwall face. Parametric studies were conducted to develop proactive goaf inertisation strategies to minimise the spontaneous combustion zones. Results indicated that effective goaf inertisation can be achieved by injecting inert gas on the belt road side at least 100 m behind the face, and, if underground access becomes prohibitive, injection can be carried out via surface wells/boreholes. Injection behind the retaining wall is only effective for localised heating(s) around the injection point(s), as much of the injected inert gas will be dispersed by air leakage along the unconsolidated goaf boundary.

INTRODUCTION

Spontaneous heating of coal has always been an issue of mine safety concern for mines operating seams liable to heating. There has been persistent effort for engineers and researchers to understand the mechanism of self-heating of coal and the prevention of its further development into spontaneous combustion or open fire. Experimental studies were carried out by Beamish (2005) and Beamish et al. (2005a; 2005b) to investigate the relationship between R70 and coal rank (together with the ash content and moisture) aiming at better evaluating the risk of spontaneous combustion. Yuan and Smith (2007; 2008; 2009) conducted numerical studies on the self-heating of coal in longwall gobs where the impact of bleeder and bleederless ventilation system was investigated. Rapid inertisation strategies were developed by Ren et al. (2005) and Ren and Balusu, (2009) by inert gas injection in longwall goafs assisted by extensive Computational Fluid Dynamics (CFD) modelling, and field application demonstrated the success in converting goaf into an inert environment; Whilst in China, the three-phase foam (composed of mud, nitrogen, and water), slurry-grouting, gel and inert gas are typical strategies carried out in field for goaf inertisation and fire control (Zhou, et al., 2006; Liang and Luo, 2008).

Like many other coal production countries, a large number of Chinese coal mines are also under the threat of spontaneous combustion (Liang and Luo, 2008). Fenghuangshan is a Chinese coal mine famous for its good quality coal (Anthracite/hard coal). Although the coal is identified as spontaneous combustion prone, Y type ventilation was adopted for 154307 longwall face to minimise the number of coal pillars and increase the coal extraction rate simultaneously. This paper presents the development of 3D CFD models as a tool to investigate oxygen ingress distribution patterns particularly potential spontaneous oxidation zones behind the face and the optimisation of inertisation strategies under the mining condition of Fenghuangshan mine.

OXYGEN INGRESS INTO LONGWALL GOAF

It is acknowledged that Y type ventilation can lead to higher levels of oxygen penetration into the goaf than the U type ventilation (Smith, et al., 1994). For the effective control of a possible goaf fire development in a spontaneous combustion prone coal seam, the air ingress patterns must be well understood and the gas compositions in the goaf carefully monitored to understand the changes of goaf atmosphere as the longwall retreats from its start-up line.

Figure 1 shows the layout and ventilation system of the 154307 longwall face, in which both belt road and rail transport road are used to bring fresh air to the face. As the face retreats from the start-up line,
Caved goaf is formed immediately behind face, and the rail (track) roadway behind the face (which would collapse normally) is maintained by building a retaining wall close to the longwall goaf, and used as a bleeder road for air return in this case, thus forming the Y type ventilation. A tube bundle system was employed on site along the retaining wall at intervals of 30 to 50 m (refer to the red points in Figure 1) to monitor changes of goaf gas composition (distribution) inside the retaining wall along the goaf edges, especially the levels of carbon monoxide (CO) which has been widely used as an indicator gas for spontaneous heating. Gas samples were collected every two days and analysed by dedicated gas chromatograph (GC) on the surface. Figure 2 provides a snapshot of oxygen distribution in the goaf along the retaining wall when the longwall face had retreated some 850 m from the start-up line. Gas monitoring data showed that there was significant ingress of fresh air on the goaf rib side, with high oxygen levels at more than 18% even 700 m behind the face. This high level oxygen leakage could lead to the development of self heating of residual coal in the goaf, particularly around the edge of unconsolidated goaf inside the retaining wall.

![Figure 1 - Longwall face layout and ventilation system](image1)

![Figure 2 - Oxygen concentration changes along the goaf](image2)

**BASE CFD MODEL RESULTS AND VALIDATION**

A three dimensional CFD model was developed to represent the longwall face which was 780 m in length, 176.4 m in width and 80 m in height to cover the caved and fractured zones in the goaf. Figure 3 shows the geometry of the CFD model, boundary conditions and computational grid.

As an integrated part of the CFD numerical modelling, validation of base model results was carried out using the field gas monitoring data. Figure 4 shows a comparison of the predicted oxygen concentration levels and field monitoring data inside the retaining wall along the goaf. CFD modelling results show that the oxygen ingress inside the retaining wall remains as high as 20% even up to 700 m behind the longwall, indicating that significant air leakage between the retaining wall and boundary of consolidated goaf. A good agreement can be observed between the base CFD modelling result and field gas monitoring data. The base model was then used to investigate the spatial distribution of spontaneous heating zones and proactive goaf inertisation strategies.
Figure 3 - Longwall CFD model geometry and computational grid

(a) CFD base model results - oxygen levels along the goaf edge inside the retaining wall
Figure 4 - Base CFD model validation - model results vs. field monitoring data

Figure 5 shows the CFD predicted oxygen distribution and the spatial distribution of spontaneous combustion zones in the goaf. CFD results indicate that fresh air from the belt road (inlet 1) leaks into the goaf along the ribs to a depth up to 300 m, the leaked air partly travels further towards face start-up line, and partly penetrates across the unconsolidated goaf, but mostly turns back along the unconsolidated goaf edge to join the main air stream behind the chocks within 50 m in the goaf; it then merges with air leakage from the face and rail road (inlet 2), and travels along the goaf boundary strip inside the retaining wall until the start-up line, before reporting to the return roadway. Figure 5 (b) shows the 3D Iso-Surfaces of oxygen concentration at 7% and 18%, between which is the oxidation zone where the heating of residual coal is mostly to occur. As can be observed from the plot, this area is spatially distributed in the goaf on the belt road side up to 200 m, 50 m behind the face, and along the retaining wall side about 60 m into the goaf. Beyond this area are the cooling zones (oxygen level is above 18%) where excessive air leakage will dissipate any oxidation heat to support the self-heating process, and the choking zone (oxygen level is below 7%) where the oxygen level would be insufficient for coal to undergo active spontaneous heating.

CFD model results show that the use of Y type ventilation can induce serious air leakage spatially into the goaf with a wide range of oxidation zones that are conductive to spontaneous heating. The most likely areas for spontaneous combustion to occur in the goaf (at mining level) are the goaf edge on the belt road side some 200 m behind the face and 60 into the goaf; within 50 m behind the chocks, and the unconsolidated areas some 60 m inside the retaining wall along the bleeder (return) road till the start-up line. Goaf gas composition along the retaining wall must be carefully monitored to detect the onset of any heating and avoid delayed control actions against the occurrence of a possible heating in these areas. Obviously any proactive measures such as goaf inertisation should be targeted towards these areas.
(b) Spatial distribution of potential spontaneous heating zones

Figure 5 - Oxygen distribution and spatial distribution of spontaneous heating zones in the goaf

PARAMETRIC STUDY OF GOAF INERTISATION STRATEGIES

An effective method to prevent the development of self-heating in longwall goafs is to implement proactive inertisation by pumping inert gas such as nitrogen or carbon dioxide into the goaf to minimize the area of potential oxidation zones. To develop the optimum inertisation strategies, a set of parametric studies were conducted using the CFD model to identify the optimum injection point behind the longwall face.

Inertisation from belt road side

Figure 6 shows a comparison of oxygen distribution in the goaf at the mining level after inert gas (pure nitrogen) injection at 30 m, 50 m, 100 m and 200 m behind the face on the belt road side at a rate of 0.25 m³/s. It can be seen that the oxidation zones (refer to Figure 5a) on the belt road side and behind the chocks has been eliminated by injecting inert gas at more than 50 m behind face while the inertisation has limited effect inside the retaining wall side, unless the injection is conducted at some 200 m behind face. It also indicates that inert gas injection point should not be too close (i.e., less than 50 m) to the face where the goaf is not completely compacted and air leakage is relatively high, the inert gas will be easily dispersed by air leakage as soon as it is injected.
Inertisation from retaining wall

Figure 7 shows the effect of goaf inertisation when injection is carried out on the retaining wall side. CFD modelling results show that injection on the retaining wall side only has a limited effect on areas around the injection point, as much of the inert gas will be flushed away by air leakage inside the retaining wall. Again, this demonstrates that good inertisation effect cannot be obtained if injection is carried out at high air leakage area. There is almost no inertisation effect in oxidation zones on the belt road side and immediately behind the chocks. Consequently, goaf inertisation should not be conducted on the retraining wall side, unless it is needed to deal with localised heating in combination with other control measures such as multiple injection points (Figure 7.c and d), foaming or temporary stoppings to minimise air leakage into these areas.
Inertisation using surface goaf wells or boreholes

Under certain circumstances when underground access becomes prohibitive (e.g. mine evacuation), inertisation can be conducted using existing wells or boreholes which are usually used for gas drainage. In this case, it is assumed that the surface wells are located at 60 m away from belt road side at an interval of 200 m, and the first well is 130 m behind face. Figure 8 shows the goaf inertisation effect of inert gas injection from a single surface well/borehole and a combination of two or four goaf wells. It can be seen that the inertisation of high oxygen areas in the goaf can be more effectively achieved by injecting inert gas via surface goaf wells/boreholes. Modelling results show that a combination of surface wells should be used to inject inert gas for goaf inertisation when surface access can be obtained, rather than delivering inert gas underground. At least, the practice of using a single borehole for inertisation at about 100 m following the face, as shown in Figure 8.a, should be adopted for longwall systems using Y ventilation (or bleeder road).

![Diagram of inert gas injection at 100 m behind face](image1)

![Diagram of inert gas injection simultaneously at 50, 100 and 200 m behind face](image2)

![Diagram of inert gas injection at 100, 200 and 300 m behind face on the retained wall side](image3)

![Diagram of oxygen distribution](image4)

Figure 7 - Goaf inertisation on oxygen distribution - injection from retaining wall

(a) Inert gas injection from the first surface well

(b) Inert gas injection at 100 m behind face

(c) Inert gas injection simultaneously at 50, 100 and 200 m behind face

(d) Inert gas injection at 100, 200 and 300 m behind face on the retained wall side
CONCLUSIONS

Three dimensional CFD modelling has been developed to investigate the distribution of spontaneous combustion zones for a Chinese longwall face using Y type ventilation. Modelling results show that the most likely areas (7–18% oxygen) for spontaneous combustion to occur spatially in the goaf are the goaf edge on the belt road side some 200 m behind the face and 60 m into the goaf; within 50 m behind the chocks, and the unconsolidated areas about 60 m inside the retaining wall along the bleeder (return) road till the start-up line. Air leakage is most excessive in the unconsolidated goaf boundary inside the retaining wall along the bleeder road.

Both goaf gas monitoring and proactive inertisation are recommended to minimise the occurrence of spontaneous goaf heating. Optimum goaf inertisation can be achieved by pumping inert gas (nitrogen at a rate of 0.25 m$^3$/s) at least 100 m behind the face on belt road side, or ideally, if surface access permitting, via surface goaf hole(s). Goaf inertisation on the retraining wall side will only be effective for localised heating, and should be used in combination with other control measures to minimise air dilution in these areas.
REFERENCES


SELECTION OF OPTIMUM COMBINATION OF FANS FOR BORD AND PILLAR COAL MINES - A CASE STUDY

Ramakrishna Morla\textsuperscript{1}, Krishna Tanguturi\textsuperscript{1} and A. Manohar Rao\textsuperscript{2}

\textbf{ABSTRACT:} A good design of ventilation system for a mine should supply adequate air flow for all workings. One of the objectives of the ventilation planning is to select optimum operating points for the fan and its combinations to achieve the required air flow rate. It will improve the safety conditions and minimise the total air power consumption. The objective of this study is to increase ventilation quantity with low operating pressure and effective utilisation of the main mechanical ventilator. Ventilation simulator and CFD modelling studies were conducted with different combinations of fans. Operating parallel fans with the same capacity and blade angle at the return air shaft is the best possible solution for achieving this objective. A detailed case study of the pressure survey of the ventilation network and the simulation results is presented.

\textbf{INTRODUCTION}

In India, Singareni Collieries Company Limited (SCCL) is one of the biggest coal producing company with 36 underground and 14 opencast mines in operation. To increase the production capacity in the currently operating underground mines demands full utilisation of main mechanical ventilator(s). At high fan pressures there is a chance of air leakage through fire seals, ventilation stopping's and barriers, which will lead to fires. Due to differential pressures in multi seam workings there can be air breathing into different seams, which lead to spontaneous combustion and fires.

The total production capacity including underground and opencast mining methods of SCCL is approximately 52 Mtpa. Of this, 12 Mtpa of coal is produced from underground mining operations using methods like bord and pillar, longwall, continuous miner and blasting gallery. Underground coal mining in SCCL is currently operating at a maximum depth of 500 m and most of the operations are carried at 200 to 400 m depth.

In SCCL most of the main fans are axial flow fans rated between 150 to 225 kW. Out of 36 underground mines highest pressure delivered by the main fan is 860 Pa and highest quantity delivered is 200 m\textsuperscript{3}/s. The average resistance of the mines is 0.0365 \textit{Ns}^2\textit{m}^8, air flow quantity is 133 m\textsuperscript{3}/s and pressure developed by main fans is 540 Pa. The company is spending US$2.84 million per annum for main fans power consumption. The average power cost of the main fan for SCCL operated mines is US$136 per annum per pascal.

The SCCL has proposed to increase a production capacity from various existing underground mines by effectively utilising the available fan facilities and building additional infrastructure wherever needed. A detailed case study of the GDK 11 incline was carried out to assess the necessary action taken.

\textbf{GDK 11 INCLINE}

GDK 11 incline is situated in RG-I area of SCCL which is located at a distance of 18 km from Ramagundam Railway Station. The mine has four workable Seams (I, II, III and IV) and the gradient of the seams is 1 in 8 and the gassiness of the seams classified as degree-I. All the seams were extensively developed by bord and pillar methods. Reserves in Seam-I are extracted by Continuous Miner Technology at a maximum depth of 350 m. Reserves in Seam-II are being developed and extracted by semi-mechanization with Load Haul Dumpers (LHD). Reserves in Seam-III are extracted by the Blasting Gallery (BG) method. Reserves in Seam-IV are extracted by hydraulic stowing with LHDs.

\textsuperscript{1} CSIRO Earth Science and Resource Engineering, Pullenvale-4069, QLD, Australia
\textsuperscript{2} The Singareni Collieries Company Limited, Kothagudem, AP -507101, India
Details of existing ventilation system

GDK 11 incline has four intakes ie. main incline dip (MID), manway dip (MWD), 3rd entry and an 80 m depth intake shaft. All these four intakes were connected to seam I. This mine has a 190 m depth return shaft, which is connected to seam I, II and III. The diameters of the return and intake shafts are 6 m. The mine has one Voltas made 225 kW fan with a maximum blade angle of 32.5°. The total air flow quantity delivered by the main fan at 650 Pa pressure is 205 m$^3$/s. Table 1 shows the details of the existing ventilation system.

Table 1 - Details of existing ventilation system

<table>
<thead>
<tr>
<th>Sl.No</th>
<th>Parameter</th>
<th>Details</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Seam-I quantity (m$^3$/s)</td>
<td>106.10</td>
</tr>
<tr>
<td>2</td>
<td>Seam-II quantity (m$^3$/s)</td>
<td>26.11</td>
</tr>
<tr>
<td>3</td>
<td>Seam-III quantity (m$^3$/s)</td>
<td>44.85</td>
</tr>
<tr>
<td>4</td>
<td>Seam-IV quantity (m$^3$/s)</td>
<td>27.94</td>
</tr>
<tr>
<td>5</td>
<td>Total air quantity (m$^3$/s)</td>
<td>205.00</td>
</tr>
<tr>
<td>6</td>
<td>Pressure drop (Pa)</td>
<td>650</td>
</tr>
<tr>
<td>7</td>
<td>Total Air power (kW)</td>
<td>133.35</td>
</tr>
<tr>
<td>8</td>
<td>Power consumption per year @ 60% Efficiency</td>
<td>1945 450</td>
</tr>
<tr>
<td>9</td>
<td>Power cost per year @ US$0.0678/kWh</td>
<td>131 901</td>
</tr>
</tbody>
</table>

The overall resistance of the mine is 0.15 Ns$^2$/m$^8$ and the area of equivalent orifice of the mine is 9.7 m$^2$. Pressure drop at the return air shaft is 94 Pa (14%) and at the intake entries is 82 Pa (12.6%). All intakes and returns have parallel paths.

Objective

GDK 11 incline has planned to improve its production targets from 0.8 Mtpa to 1 Mtpa. For achieving this production target the main mechanical ventilator(s) should satisfy the following conditions:

- The overall required quantity for the mine should be 235 m$^3$/s but with the existing 225 kW single fan it is only possible to get up to 205 m$^3$/s;
- A new ventilation system should be used with only one return shaft;
- The fan(s) should operate at the lowest operating pressure (<900 Pa).

To meet the above specified objectives for the GDK 11 incline, the following schemes for upgrading can be considered for the existing ventilation and to meet the projected capacity:

- Effective utilisation of existing entries and provision of additional new entries;
- Re-organization of existing ventilation systems;
- Optimum utilisation of existing capacity of main fans;
- Installation of high capacity main fan;
- Usage of booster fans;
- Introduction of an optimum combination of parallel fans.

For the use of high capacity fans, it requires high initial investment and construction of new fan house. These fans operate at high pressure which leads to fire. Inclusion of new entries is a time taking process and may take three to four years of time. With booster fans it is possible to get additional quantity but its usage in bord and pillar method of SCCL mines is not practised due to pillar fires. Fixing of parallel fans is more convenient and better suitable for delivering high quantity of air at low operating pressure.
Simplified ventilation network diagram

The simplified ventilation network diagram (Manohar and Morla, 2011) of GDK-11 incline has 53 nodes, 79 branches and 11 tunnels. Figures 1 to 4 shows the nodes, branch locations, intakes and return paths of the simplified ventilation network diagrams of the mine.

Figure 1 - Simplified ventilation network diagram of seam I

Figure 2 - Simplified ventilation network diagram of seam-II

Figure 3 - Simplified ventilation network diagram of seams-III

Figure 4 - Simplified ventilation network diagram of seam-III Top Section and seam-IV
Branches details

Field study was conducted to measure air quantity and pressure drops between the nodes. Using the equation \( R = \frac{P}{Q^2} \), the resistances of all branches were calculated. Table 2 shows the starting node and ending node of the branches and their resistance values.

<table>
<thead>
<tr>
<th>S.N</th>
<th>E.N</th>
<th>R</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>2</td>
<td>0.0078</td>
</tr>
<tr>
<td>1</td>
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<td>9</td>
<td>0.1022</td>
</tr>
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<td>3</td>
<td>4</td>
<td>0.0010</td>
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<td>3</td>
<td>46</td>
<td>68</td>
</tr>
<tr>
<td>4</td>
<td>5</td>
<td>0.0005</td>
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<td>48</td>
<td>69.92</td>
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<tr>
<td>5</td>
<td>6</td>
<td>0.0019</td>
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<td>5</td>
<td>45</td>
<td>13.509</td>
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<td>6</td>
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<td>0.0011</td>
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<td>0.0095</td>
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<tr>
<td>10</td>
<td>26</td>
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<td>11</td>
<td>13</td>
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<table>
<thead>
<tr>
<th>S.N</th>
<th>E.N</th>
<th>R</th>
</tr>
</thead>
<tbody>
<tr>
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<td>15</td>
<td>9.36</td>
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<td>12</td>
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<td>42</td>
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</tr>
<tr>
<td>25</td>
<td>38</td>
<td>0.01021</td>
</tr>
</tbody>
</table>

Table 2 - Branches with resistances values

Parallel fans

In a parallel fan arrangement, three main fans installed at top of the upcast shaft are set to operate at uniform capacity and blade angle. Any two fans will be in continuous operation and the other fan is kept as standby. All of these parallel fans are electrically inter-locked in such a way that shutting off one of the fan will automatically shut the other fan. Figure 5 shows the arrangement of parallel fans.

Details of simulations

For achieving the specified objective, simulation studies were conducted with ventilation simulator (Sastry and James, 1985) for different cases. The details of ventilation simulations with different cases are outlined in Table 3.
Table 3 - Details of simulations

<table>
<thead>
<tr>
<th>Case</th>
<th>No. of Intakes &amp; Details</th>
<th>No. of Returns &amp; Details</th>
<th>No. of fans &amp; Capacity</th>
<th>Fan Blade Angle</th>
<th>Fan constants</th>
</tr>
</thead>
<tbody>
<tr>
<td>Existing</td>
<td>4 (MID, MWD, 3rd entry and 80 m shaft)</td>
<td>1 (190m shaft)</td>
<td>1 (225 kW)</td>
<td>32.5°</td>
<td>A=-0.02398 B=1701</td>
</tr>
<tr>
<td>Case 1</td>
<td>4 (MID, MWD, 3rd entry and 80 m shaft)</td>
<td>1 (190m shaft)</td>
<td>1 (375 kW)</td>
<td>15°</td>
<td>A=0 B=1000</td>
</tr>
<tr>
<td>Case 2</td>
<td>3 (MID, MWD and 3rd entry)</td>
<td>2 (80 &amp;190m shafts)</td>
<td>2 (225 kW)</td>
<td>17.5° each</td>
<td>A=-0.04412 B=1484.62</td>
</tr>
<tr>
<td>Case 3</td>
<td>4 (MID, MWD, 3rd entry and 80 m shaft)</td>
<td>1 (190m shaft)</td>
<td>2 parallel fans (225 kW each)</td>
<td>17.5° each</td>
<td>A=-0.04412 B=1484.62</td>
</tr>
<tr>
<td>Case 4</td>
<td>4 (MID, MWD, 3rd entry and 80 m shaft)</td>
<td>1 (190m shaft)</td>
<td>2 parallel fans (225 kW each)</td>
<td>15° &amp; 17.5°</td>
<td>A=-0.04953 B=1319.20 (For 15°)</td>
</tr>
<tr>
<td>Case 5</td>
<td>4 (MID, MWD, 3rd entry and 80 m shaft)</td>
<td>1 (190m shaft)</td>
<td>2 parallel fans (225 kW each)</td>
<td>17.5° &amp; 20°</td>
<td>A=-0.03372 B=1489.94 (For 20°)</td>
</tr>
<tr>
<td>Case 6</td>
<td>4 (MID, MWD, 3rd entry and 80 m shaft)</td>
<td>1 (190m shaft)</td>
<td>2 parallel fans (150&amp;225 kW)</td>
<td>17.5° each</td>
<td>A=-0.0362 B=760 (150kW fan)</td>
</tr>
</tbody>
</table>

Fan constants (A and B values) calculated from fan characteristic curve with the least square approximation method. All cases had four intake air ways and one return air way but case 2 has three intakes and two returns. Case 1 had a 375 kW fan, case 6 had a combination of 150 kW and 225 kW fans and all other cases have 225 kW fan(s). First three cases have single fan and all other cases have parallel fans. All intakes are connected to one seam and air travels through tunnels to other seams. The return air shaft is connected to seam I, II and III. Figure 6 and 7 shows the details of intake and return air ways for all cases.

**SIMULATION RESULTS AND DISCUSSIONS**

In case 1, the ventilation system operates with a 375 kW fan at 15° blade angle with four intakes and one return air way. The 375 kW fan is able to supply 246 m³/s of air flow quantity which is sufficient for all the seams but the pressure developed by this fan is 1 000 Pa, which is very high, and there is a chance of fire in BG panels and at other sealed off workings. The other issues for SCCL for not operating 375 kW fan is it requires construction of new fan house and purchase of fan structures. Installation of new 375 kW fan at existing return air shaft will take two to three years of time. Also, in this case total air power is 246.7 kW and power cost per year is US$ 244 203 which is very high when compared with the existing system.
In case 2, the ventilation system operates with two 225 kW fans at 17.5° blade angle at two different shafts with three intakes and two return air ways. The overall resistance of the mine is increased when the intake shaft is used as a return air shaft. The main fan pressure for 80 m depth shaft is 946 Pa, which is in stall zone. Air delivered by these two fans is 216 m³/s which is less than the recommended quantity of 235 m³/s. The total air power is 196.86 kW and the power cost per year is US$ 194,867 which is very high when compared with the existing system. The cost for maintenance of two fans at different locations is expensive.

In case 3, the ventilation system operates with two 225 kW fans at 17.5° blade angle with four intakes and one return air shaft. The air flow quantity delivered by these combinations of fans is 238 m³/s at a pressure of 850 Pa which satisfies the recommended quantity. The total air power is 202 kW and the power cost per year is US$200,252 which is less than that of 375 kW fan. For this case an additional third fan is required as a standby fan.

In case 4, the ventilation system operates with two 225 kW fans at 15° and 17.5° blade angles with four intakes and one return air shaft. The total air quantity delivered by these combinations of fans is 226 m³/s at a pressure of 790 Pa which is less than the minimum requirement of the mine. The total air power is 178.5 kW and the power consumption per year is US$176,733 which is very high when compared with the existing system.

In case 5, the ventilation system operates with two 225 kW fans at 17.5° and 20° blade angle with four intakes and one return air shaft. The combination of fans able to supply 242 m³/s of air flow quantity which is sufficient for all the seams but the pressure developed by this fan is 908 Pa which is high. The total air power 219.7 kW and the power consumption per year is US$ 217,512 which is very high when compared with the existing system.

In case 6, the ventilation system operates with 150 kW and 225 kW fans at 17.5° blade angle at two different shafts with four intakes and one return air way. The total air quantity delivered by this combination of fans is 200 m³/s at 615 Pa pressure which is less than the minimum requirement of the existing system. The total air power is 123.7 kW and the power consumption per annum is US$ 122,448 which is much less when compared with the existing system. The air flow quantity delivered by 150 kW fan is 62 m³/s at an operating pressure of 615 Pa which may damage the fan due to its high pressure.

Table 4 and 5 shows the brief simulation results of all the cases.

<table>
<thead>
<tr>
<th>Sl.No</th>
<th>Parameter</th>
<th>Existing</th>
<th>Case-1</th>
<th>Case-2</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Seam-I quantity (m³/s)</td>
<td>106.10</td>
<td>127.67</td>
<td>108.47</td>
</tr>
<tr>
<td>2</td>
<td>Seam-II quantity (m³/s)</td>
<td>26.11</td>
<td>31.51</td>
<td>27.80</td>
</tr>
<tr>
<td>3</td>
<td>Seam-III quantity (m³/s)</td>
<td>44.85</td>
<td>53.93</td>
<td>49.13</td>
</tr>
<tr>
<td>4</td>
<td>Seam-IV quantity (m³/s)</td>
<td>27.94</td>
<td>33.59</td>
<td>30.60</td>
</tr>
<tr>
<td>5</td>
<td>Total air quantity (m³/s)</td>
<td>205.00</td>
<td>246.70</td>
<td>115+101 = 216.00</td>
</tr>
<tr>
<td>6</td>
<td>Pressure drop(Pa)</td>
<td>650</td>
<td>1000</td>
<td>881&amp;946</td>
</tr>
<tr>
<td>7</td>
<td>Total Air power (kW)</td>
<td>133.35</td>
<td>246.70</td>
<td>196.86</td>
</tr>
<tr>
<td>8</td>
<td>Power consumption per year @ 60% Efficiency</td>
<td>1945 450</td>
<td>3 601 820</td>
<td>2 874 156</td>
</tr>
<tr>
<td>9</td>
<td>Power cost per year @ $0.0678/kWh</td>
<td>131 901</td>
<td>244 203</td>
<td>194 867</td>
</tr>
<tr>
<td>10</td>
<td>Remarks</td>
<td>One 225 kW Fan at 32.5° Blade Angle</td>
<td>One 375 kW Fan</td>
<td>Two fans at two shafts, both with 17.5° blade angle</td>
</tr>
</tbody>
</table>

**Table 4 - Simulation results of existing, case 1 and 2**

**CFD models**

Preliminary CFD modelling simulations were carried out for single fan, parallel fans with same capacity and blade angle and parallel fans with different capacities.

CFD modelling studies were conducted with single and/or double fans operating at different pressures. The equivalent orifice area of the main shaft inlet is about 9.7 m². Figure 9 indicates velocity and pressure distribution pattern for a single fan of 225 kW at 32.5° blade angle. The pressure developed by the fan is about 650 Pa which was introduced as boundary condition in the CFD model.
results indicated that air flow quantity at the fan outlet is about 206 m$^3$/s. The results of the ventilation simulator and the CFD model for existing condition are more or less the same.

**Table 5 - Simulation results of case 3, 4, 5 and 6**

<table>
<thead>
<tr>
<th>Sl.No</th>
<th>Parameter</th>
<th>Case-3</th>
<th>Case-4</th>
<th>Case-5</th>
<th>Case-6</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Seam-I quantity (m$^3$/s)</td>
<td>123.99</td>
<td>116.80</td>
<td>125.00</td>
<td>101.00</td>
</tr>
<tr>
<td>2</td>
<td>Seam-II quantity (m$^3$/s)</td>
<td>30.11</td>
<td>29.00</td>
<td>31.00</td>
<td>27.00</td>
</tr>
<tr>
<td>3</td>
<td>Seam-III quantity (m$^3$/s)</td>
<td>51.54</td>
<td>49.40</td>
<td>53.00</td>
<td>44.00</td>
</tr>
<tr>
<td>4</td>
<td>Seam-IV quantity (m$^3$/s)</td>
<td>32.36</td>
<td>30.80</td>
<td>33.00</td>
<td>28.00</td>
</tr>
<tr>
<td>5</td>
<td>Total air quantity (m$^3$/s)</td>
<td>119 + 119 = 238.00</td>
<td>124+102 = 226.00</td>
<td>113+129 = 242.00</td>
<td>138+62 = 200.00</td>
</tr>
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<td>6</td>
<td>Pressure drop (Pa)</td>
<td>850</td>
<td>790</td>
<td>908</td>
<td>615</td>
</tr>
<tr>
<td>7</td>
<td>Total Air power (kW)</td>
<td>202.30</td>
<td>178.54</td>
<td>219.736</td>
<td>123.7</td>
</tr>
<tr>
<td>8</td>
<td>Power consumption per year @ 60% Efficiency</td>
<td>2 953 580</td>
<td>2 606 684</td>
<td>3 208 145</td>
<td>1 806 020</td>
</tr>
<tr>
<td>9</td>
<td>Power cost per year @ $0.0678 / kWh</td>
<td>200 252.7</td>
<td>176 733</td>
<td>217 512</td>
<td>122 448</td>
</tr>
<tr>
<td>10</td>
<td>Remarks</td>
<td>Two 225kW parallel fans at 17.5° Blade Angle</td>
<td>Two 225kW parallel fans at 15° &amp; 17.5° Blade Angle</td>
<td>Two 225kW parallel fans at 17.5° &amp; 20° Blade Angle</td>
<td>One 150kW and one 225kW parallel fans</td>
</tr>
</tbody>
</table>

Figure 10 indicates the velocity and pressure distribution pattern for double 225 kW fans with the same blade angle of 17.5°. The pressure developed by the fans is about 850pa which was introduced as the boundary condition in the CFD model. Modelling results indicated that air flow quantity at each fan outlet is about 120 m$^3$/s.

Figure 11 indicates the velocity and pressure distribution pattern for double fans of 150 kW and 225 kW capacity. The pressure developed by these fans is about 340 Pa and 650 Pa which was introduced as the boundary condition in the CFD model. Modelling results indicated that air flow quantity at the fans outlets are about 63 m$^3$/s (revers flow) and 249 m$^3$/s. The combination of low and high capacity fans will lead to reverse flow at low capacity fan region which may damage the fan.
CONCLUSIONS

In bord and pillar mines, alteration of the intake air shaft into a return air shaft will increase overall resistance of the mine. Installation of high capacity fans leads to fire in BG panels and other sealed off area. Installation and operation of parallel fans at same capacity and blade angle more useful for achieving required quantity of air at low operating pressure. It is easy to install parallel fans at same capacity and blade angle with utilisation of the existing low capacity fans and fan houses. Parallel fans at same capacity and blade angle will give better results than different blade angles. Usage of parallel fans with different power capacities may damages the low capacity fan.

REFERENCES


A Manohar Rao and Ramakrishna Morla, 2011. Selection of parallel fans for effective Ventilation in existing ventilation systems of underground coal mines of SCCL. In proceedings International Mining Conference, (NIT Rourkela, India).
DESIGN AND CONSTRUCTION OF WATER HOLDING BULKHEADS AT XSTRATACOAL’S OAKY NO 1 MINE

Verne Mutton¹, Michael Salu², Mark Johnston³ and Christian Mans⁴

ABSTRACT: A systematic approach is required for the design of bulkheads including consideration of the longevity of building materials, quality control during construction and methods to monitor performance of the retention system. Bulkhead site preparation and construction method using wet mix concrete is described with particular reference to anchoring the bulkheads to the strata with keys and steel bolts and post-construction resin injection of the concrete/strata interface.

INTRODUCTION

In 2011 over 40 mines on the East Coast of Queensland were impacted by flooding caused by severe weather events and in many cases mines were shut down causing production loss and financial impact. Potential hazards exist where there are accumulations of any material that flows when wet such as waste rock in an underground cut-through, water storage dams, tailings and waste dumps. At some operations difficulties have been experienced in pumping out excessive water because of limits on environmental approvals. Principal hazard management plans should provide for adequate mechanisms to warn of potential flooding and guidance on the appropriate actions to take depending on the likelihood and severity of such flooding, which may include a system of evacuation or moving people to a place of safety and ensuring the site and equipment is properly prepared to minimize risk.

Because of coal seam contour shape it is sometimes necessary to store high heads and volumes of water in areas of the mine that are at a higher contour level than the working longwall or development panels inbye of these areas. The location and head requirements of hard barriers such as bulkheads is critical and as far is practical potential conduits, for example boreholes, joint sets, partings and shear zones should be sealed in the zone effected by the impoundment. The erosion capacity of water driven by a permanent and substantial pressure head is strong and constant with potential for scouring joints and cracks with increased site permeability. For this reason organic resin injection of bulkhead sites was chosen for blocking leakage paths at bulkhead sites and increasing strength of strata confining the bulkheads. Additional secondary support can be particularly effective in containing relaxed incumbent strata.

OAKY CREEK

Oaky Creek Coal (OCC) is located between the mining towns of Tieri and Middlemount in Central Queensland. It extracts the German Creek Seam to produce one of the most sought after Bowen Basin coking coals. The Oaky Creek Complex consists of two underground Longwall mines and a coal handling and preparation plant. Oaky Creek No.1 has been operating for over two decades, while Oaky North has been in operation since 1995 with its first Longwall coal in 1999. Development at Oaky No.1 started in 1989 with first Longwall coal in 1990. Initial development included driveage of the main dips area which allowed mining of Longwalls 1 to 12. This was followed by the North East Mains which allowed extraction of blocks 14 to 20. The South Mains were driven while retreat of longwall blocks 21 to 25 commenced. Development of the Sandy Creek East Mains (SCE Mains) was also started during this time. Operations at the mine gradually progressed to the SCE area by late 2006. Currently Oaky Creek 1 (OC1) runs three full time development panels and is about to commission a new Longwall. Production for 2010 was reported as 5.57 M t ROM, reaching the mine record for annual tonnes produced. With up to nine gateroad developments, face-line drives and longwall blocks remaining, production is expected to continue until mid 2016.

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GEOLOGY

OC1 has relatively uniform regional lithology, the primary units making up the immediate roof are described in the OCC model as three sandstone units known as A10, A20 and A30 which are overlain by the Sandy Creek Siltstone (SCST) (Esterle, et al., 2003). The three sandstone units are separated by weaker siltstone units. A generalised lithology of OC1 is shown in Figure 3. The SCST has a geophysical response indicating a unit which is more porous, higher in potassium content and relatively weaker than the surrounding units.

The current classification system defines the immediate roof lithology above the German Creek Working Seam (GCWS) as comprising two competent sandstone units (RU2 and RU4) with two weaker interburden units which separate the GCWS, RU2 and RU4. These units have been labelled RU1 and RU3.
Sandstone is the predominant lithology from the SCST upwards to the Corvus 1 seam (C1), however, minor units of laminated to interbedded sandstone / siltstone and laminated siltstone may be present.

**Seam geology**

The GCWS varies in thickness from 2.9 m in the NW to 1.7 m towards the SE of the OC1 lease (Mans, *et al.*, 2011). The thickest part of the seam is at the inbye ends of Longwalls 34 to 36. The seam gradually thins SE down to 2.0 m by MG30, and 1.7 m by the inbye end of MG33.

The face cleat trend is consistent across the mine workings with most readings falling between 140° and 160°. The butt cleat trends NE, however swings orientation within 45° from region to region. Major structures can alter cleat directions in both the horizontal and vertical planes.

**In situ stress measurements**

Thirteen surface over-core stress measurements have been taken across OCC. The current over coring database consists of 34 samples taken from these 13 boreholes, three from OC1 and 10 from Oaky Creek North (OCN), analysis of the data shows a mean principal horizontal stress direction of 12° (with 1 standard deviation = 17°).

The $\sigma_{H}/\sigma_{V}$ ratio and $\sigma_{H}/\sigma_{Q}$ ratio have been determined using Young’s Modulus (E) and depth of cover according to the methodology of Colwell and Frith (2006). The rock strength characteristics database for OCC has been utilised to correlate E with sonic derived UCS values (Table1) ($y=0.149x + 3.9443$, R$^2 = 0.3452$). This equation should be updated regularly as testing information becomes available. The E for the bolting horizon has been calculated using the average sonic derived UCS. Stress Values used for this report are summarised in Table 2.

**Table 1 - Summary of rock mass properties for generalised lithology of longwalls 34 and 36**

(Note: All values in blue have been estimated from general material information due to lack of available site specific data)

**Table 2 - Summary of sonic derived UCS data for OC1**

(Note that no borehole data points are currently available for the nominated horizons and the values listed are based solely on the contour plans)
NEED FOR BULKHEAD

Parsons Brinckerhoff (PB) worked with Minova to develop a reliable engineering design for 20 m and 30 m bulkheads for Oaky No.1 mine. The mine requested a factor of safety of at least 4, resulting in extremely high design pressures when compared with short-term explosion or blast pressures that seals are typically designed for. As the bulkhead thickness increases, its structural behavior changes from primarily flexural to a combination of flexural and arching action. The proportion of load carried by each mechanism depends on the stiffness of the surrounding strata and in particular whether the roof and floor are coal or rock. By carrying out an engineering sensitivity analysis and keeping the practicalities of underground construction in mind, PB and Minova were able to provide a safe and cost-effective solution for high-head bulkheads.

Strata conditions at bulkhead site - Maingate 20a, C heading 2-3 c/t

The longwall block adjacent to the seal site was extracted in 2006. Chain pillar dimensions are 100 m x 22 m centres x 2.8 m height with a cover depth of 170 m. Under a single abutment loading scenario it is expected that chain pillar convergence or compression would be about 4 mm. Compression occurs over time, and it would be safe to assume that virtually all compression has taken place at the time of bulkhead construction.

Some long term deterioration of the immediate mine roof (<100 mm) is expected over time due to the high humidity of the air within the mine. This can readily be observed as skin failure between straps in the Main Dips and East Mains (>10 years old). In fully meshed roof it forms minor bagging of the skinned material between bolts. This does not represent any significant deterioration or cause stress increases which will lead to long-term roadway instability.

Roof bore-scope examination in “normal conditions” typically shows very minor and infrequent fracturing up to 4 m, however, the majority of fracturing is limited to the first 0.5 m of roof. The ribs are likely to be fractured (or softened) up to 1 m, forming yield zones. A Polyurethane (PUR) injection program has been designed to infill the yield zones that surround the bulkhead sites. To further define the bulkhead site geology an examination of the MG20a Geo / Geotechnical Hazard Plan reveals at the bulkhead location a Sandstone roof and floor. The closest minor fault (< 0.5 m down throw) is located at 6 cut-through. Within the first 0.5 m from the roof and floor the rock strength ranges from 15 to 55 MPa generally increasing away from the coal seam. The relatively sparse occurrence of guttering and delamination (Oaky No 1 Mine-Geological hazard management plan, 2006) in the adjacent roadways indicates a low stress environment.

![Bulkhead Seals](image)

**Figure 4 - Location of Bulkhead Seals**

DEVELOPMENT OF RAIN EVENT TRIGGER ACTION RESPONSE PLAN

The Oaky No.1 Rain Event Trigger Action Response Plan (TARP) was initially developed to handle environmental discharge issues and surface water management actions to prevent water entering the
mine portals. The mine workings have over its life been getting deeper and as such the older working areas are outbye of the current workings and at a higher Relative Level (RL) meaning that if water was to build up in these areas and a seal failed the potential to block or flood some areas of the mine outbye of the working faces exists. The original strategy was to prevent water accumulation by pumping water out of the sealed areas via borehole pumps. Mechanical failure of these pumps and surface environmental restrictions on discharging water from the site has led to a review of the mine’s water management strategy. Part of this strategy is to increase the underground storage areas, provide longer periods to be able to discharge water from these areas. This allows several ways in which the water can be removed from these areas. The current strategy allows the water from LW1 to 8 and 9 to 20A to be gravity discharged into LW21 to LW25 sealed area which has the largest storage capacity. Three borehole pumps have been installed to discharge a maximum 280 l/s to surface open pits. In order to manage discharges and the underground surface open pit water balance the Rain event TARP added seal-head into the Rain Event TARP. The TARP now has normal water head on the seals set at up to 40% of the seals head capacity. A level 1 trigger is between 40-60% of Seal head Capacity and Level 2 Trigger 60-80% and Level 3 Trigger is 80%-100%. Once the head reaches 100% of any seals design capacity, mine workers are withdrawn to the outbye side of the seal area. It must be noted that the design factor of safety for the rated seal is 4, therefore at 30 m head rating theoretically it would fail at 120 m head.

The response to each trigger level increases from ensuring pumps are in service and operating and increased inspection frequency to level 3 formation of an Incident Management Team as was the case in April 2011.

RATIONALE FOR BULKHEAD LOCATION AND DESIGN CAPACITY

Oaky No.1 Mine has unique seam contours consisting of anticlines and synclines. The mine plan has been developed in distinct areas due to faults and the geological nature of the mine. These distinct areas require each seal to be independently designed for water head. In broad terms the strategy of Longwall seals is broken into Life of Panel seals and Life of Mine Final Seals (LMFSs). Life of panel seals are 0.14 MPa (20 psi) explosion rated seals with a bulkhead rating in addition to the rated overpressure design. This is based on the water head requirement and also the time the seals are exposed to the mine workings before becoming part of the goaf. U-Tubes are installed on each seal and depending on the location, are left open as the Longwall passes and the seal becomes part of the goaf. This ensures water balance between each adjoining goaf and prevents in-goaf seal failures which could create a sudden rise in water head on LMFSs. If seal sites require greater than 10 m water head they are automatically upgraded to a 0.35 MPa (50 psi) Type D seal with the required bulkhead water head rating determines the structures thickness within given roadway cross-section dimensions. All Final Life of Mine Seals are 0.35 MPa (50 psi) rated with the additional water head rating based on a factor of safety of 1:4. The use of borehole pumps in some goaf areas assists in controlling water buildup in the goaf and on the final seals. The management of the borehole pumps is also connected with the rain event TARP to ensure water head on the seals remains in the normal operating range of 0% to 40% water head capacity.

REINFORCEMENT OF STRATA THROUGH PUR INJECTION

The following quotation is relevant when considering the treatment of the strata surrounding the bulkhead site.

When (Harteis and Dolinar, 2008) a bulkhead has failed, leakage has generally been through the surrounding strata or along the bulkhead/strata interface, with the failure potential along the interface increasing with hydraulic head.

Leakage paths will be along faults, fracture networks, coal cleat and weak partings. Gas drainage also drains moisture from the seam increasing porosity. Grout injection lengths leakage paths around the bulkhead more than a keyway will do alone. The preferred practice is to construct bulkheads within a roadway which will not be affected by changes in vertical stress. Longwall 20A had been mined ten years previously and it is expected that there will be no further change in abutment load at the bulkhead sites.

All bulkheads with design pressure heads of 10 m or greater are pre-injected with PUR before keying and construction. Pre-injection of the MG 20A 3 cut-through bulkhead site was undertaken after the full
width floor, rib and roof keys were excavated. The injection pattern of holes was extended between 1.8 m in the floor to 3 m into the surrounding roof strata to completely capture any potential yield or relaxation zones. Figure 1 shows the injection sequence for each ring. It consisted of three rings of twelve holes at a ring spacing of 1.5 m with the centre ring located in the middle of the bulkhead key. Each ring of holes was injected in a predetermined sequence with the quantity of Polyurethane (PUR) recorded for each hole. WF grade PUR was chosen as it will only foam and expand in the presence of water, otherwise remaining as a solid resin. PUR has high rock bond strength and greater ability to penetrate than cement based grouts. Roof above the coal seam is commonly carbonaceous mudstone interbedded with very fine coaly bands. Injection within the keyway is shown in Figure 6 and Figure 7 shows PUR leaking from the upper interbedded coal and mudstone layers.

![Figure 5 - Cross-section showing hole pattern and PUR injection sequence](image5.png)

**Figure 5 - Cross-section showing hole pattern and PUR injection sequence**

![Figure 6 - Bulkhead 300 mm depth key with Tensar mesh removed and PUR injection of a central ring of holes](image6.png)

**Figure 6 - Bulkhead 300 mm depth key with Tensar mesh removed and PUR injection of a central ring of holes**

![Figure 7 - PUR injection of upper rib Mudstone interbedded with coal lenses](image7.png)

**Figure 7 - PUR injection of upper rib Mudstone interbedded with coal lenses**

**PUR injection quantities**

An injection campaign of five bulkhead sites each using 36 injection holes have required PUR quantities ranging from 450 to 2610 kg per site, averaging 1453 kg with MG 20A 3 cut-through site requiring 720 kg. With injection pressures up to 10 MPa at this site the roof injection holes required only between 5-10 kg and rib coal 40 to 60 kg per hole. With one ring it was not possible to inject PUR into the floor; however within an adjacent ring of holes, three lower left hand corner holes required a total of 320 kg of PUR. In other sites coal rib holes each required up to 180 kg of PUR indicating the possibility of gas and water leakage potential through the ribs.

**Post polyurethane injection**

Experiments were undertaken (Martino and Dixon, 2006) on a cured cast concrete bulkhead that was pressured to 300 kPa and showed high flow rates. Once the concrete-rock interface was grouted
subsequent pressurization showed substantially reduced flow along the interface. Cracking in concrete can be the result of one or a combination of factors such as drying shrinkage, thermal contraction, restraint (external or internal) to shortening, sub-grade settlement, and applied loads. Due to drier conditions at the boundary rock interface there will be the possibility of micro-cracking during curing.

The S50 (50 MPa strength) shotcrete for bulkhead construction is batched at a low water powder ratio of $\approx 0.38$ which helps reduce drying shrinkage. Bulkheads were cast in high humidity conditions underground between a 150 mm thickness shotcrete stopping and a pre-existing 0.35 MPa (50 psi) Meshblock concrete seal, helping reduce moisture and heat loss during curing. This would tend to reduce drying shrinkage. For these reasons it is necessary to consider PUR injection of the boundary contact, as a precaution, once the bulkhead is cured. This can be undertaken with cast-in purpose designed injection hoses placed within the bulkhead on the boundary contact or by casting pipes into the bulkhead periphery for subsequent injection. Figure 8 shows a cross-section of 25 mm fiberglass (pressure rated at 200 MPa) injection dowels cast into the outer formwork shotcrete wall. The injection dowel ends are located within 100 mm of the concrete/strata interface. Note that the V-shaped roof key is designed to ensure that the roof contact is completely sealed during concrete pumping where concrete can be injected at the highest point.

Figure 8 - Bulkhead roof cross-section showing V shaped roof key and the position of GRP injection dowels in relation to the roof contact

Keying of bulkhead sites

The first bulkhead keys (450 man-hours) required hand work with pneumatic jack picks, working off scaffolding, which raised the potential of occupational health and safety issues. There has been much discussion on the design of equipment that could be used to mechanically remove the strata in roof, ribs and floor to provide accurate keyways. This equipment will be powered using the hydraulics of a load-haul dump machine, being mobilized and operated while using the quick detach (QDS) system shown in Figure 9. The challenge is to design a boom mounted attachment that can excavate all configurations. This attachment may also require a rock breaker to negotiate hard stone and difficult to reach areas and could be used in a variety of applications such as rib trimming, sump formation, concrete removal, niches for fire stations and pumps and drainage ditches etc.

Figure 9 - QDS attached cutter head

BULKHEAD CONSTRUCTION USING WET MIX SHOTCRETE

Construction of the first of the six proposed bulkheads commenced in September 2011 in Maingate 20A in three cut-through designed as a 30 m head capacity structure with a core thickness of 1.25 m. A design certification provided by Parsons Brinckerhoff required a 1.25 m minimum thickness concrete core with a concrete compressive strength of 40 MPa at 28 days. There was a pre-existing 50 psi Type
D Meshblock concrete seal that was rated for a water head of 10 m. Existing 150 mm Victaulic drainage pipes were extended through the new bulkhead with the 0.35 (50 psi) rated seal acting as a back wall for the 30 m bulkhead.

Skeleton bolts provide additional shear resistance to the keys reinforcing the concrete bulkhead boundary contact with the strata. It is important to consider the depth of floor keys, the floor potentially being damaged from heavy machinery movement and pillar punching. When laminated, hard, floor strata was encountered the keyway was extended to 200 mm depth as there is often minimal cohesion between plies. Key depths are extended where the floor is sloping. After keying was completed peripheral 24 mm skeleton bolts were installed within the keys forming a double row (rows at 500 mm spacing) of bolts roof and floor. Bolts are fully encapsulated as voids could provide interconnectivity with strata plies forming leakage paths. The skeleton bolt layout is shown in Figure 10.

![Figure 10 - Front view of bulkhead showing peripheral steel bolt layout](image)

In order to contain the S50 concrete a 150 mm thickness shotcrete stopping was constructed to be able to form a 1.25 m core thickness, having sufficient strength to contain the hydrostatic head from pouring the concrete. The concrete was poured in a continuous manner using 2.1 m³ kibbles and an air-driven Jacon S42 concrete pump which has a maximum delivery capacity of 6 m³/h. This type of construction is aided by the use of a surface access slick line that enables concrete delivery closer to the site. However in order to prevent quality loss due to concrete segregation caused in free fall a re-mixer at the base of the slick line is advantageous. Concrete delivery ports were located half way up the bulkhead and at the top to help control the flow and distribution of concrete within the formwork. Air-driven vibrators can be used during pouring to ensure that all pipe inclusions, steel bolts and cast-in steel hatches are fully encased.

**STRUCTURAL DESIGN ASPECTS OF BULKHEADS**

The structural engineering design of bulkheads for underground coal mining applications varies significantly from the more common design of Ventilation Control Devices (VCDs), which are designed to resist a series of short, sharp pressure pulses rather than sustained pressure over a period of time. However, for both underground bulkheads and VCDs the consequences of failure can be catastrophic and therefore a high level of risk is associated with the design and construction in each case.

Structural engineering design is only one component of a successful and “fit for purpose” bulkhead, others include:

- Correct location with respect to the geology of the roadway;
• Correct location with respect to the layout of the roads and each cut-through (proximity to intersections);
• Quality of construction materials;
• Quality of workmanship;
• Regular inspection/monitoring and maintenance.

As a chain is only as strong as its weakest link, all of the above are equally as important as the bulkhead structural design. Despite what some Mine Managers would like to believe, it is not possible to use additional engineering “safety factors” to compensate for lack of proper consideration of the dot points above.

When considering the structural design of a water bulkhead, there are a number of key differences in the design compared with VCDs and these include:

1. Bulkheads must be designed for a long-term sustained pressure load rather than a short-duration shock or transient load;
2. Potential for softening and other structural changes in the surrounding strata must be considered such as those caused by increasing abutment or vertical load due to coal extraction;
3. Potential for leakage around the perimeter of the bulkhead must be considered;
4. Pressure loading on a bulkhead will typically be trapezoidal, varying from a maximum at the base, rather than uniform;
5. Consideration of the effects of roof falls in a flooded goaf must be made including the effect of a sudden pressure wave on the bulkhead.

Other design factors to be considered include:

1. Doors (hatches) are not usually required in bulkheads;
2. Cast-in pipes must all be fitted with puddle flanges to minimize risk of leakage through the bulkhead;
3. Numbers, sizes and location of pipes through a bulkhead are similar to those for VCDs;
4. Materials and methods of construction should be similar to those used for VCDs, to simplify underground logistics and reduce special training requirements.

Many underground coal mines specify large factors of safety, such as 4, to take account of uncertainties in design, construction and the nature of the surrounding strata. PB recommends a minimum a factor of safety of two on water pressure for bulkheads, with separate considerations to be made for roof falls or explosions in the goaf.

The methodology used to design a bulkhead is somewhat dependent on the load required to be resisted. For lower loads, a simple plate bending model may be sufficient as minimum structural dimensions are likely to govern the design. When higher loads are specified, such as the 80 m and 120 m head design pressures at Oaky No.1, then more sophisticated design tools are required otherwise the design will quickly escalate to an unrealistic and uneconomic plug thickness design. A very useful numerical tool for structural engineering analysis and design of bulkheads is “Strand 7”, an Australian designed and developed 3-D finite element software package. Strand 7 has the advantage that a variety of models can be developed ranging from simple “plate” models initially through progressively more complex and more realistic “brick” models to ultimately highly detailed 3-D representations of the bulkhead, stone floor and/or roof and coal ribs/roof/floor as applicable. If an even higher level of sophistication is required, Minova have in the past, commissioned “LS Dyna” numerical models which can incorporate the effects of progressive damage to a VCD or bulkhead as well as all of the factors mentioned above (Mutton and Remennikov, 2011).

One of the consequences of the high design pressures is that bulkheads are often much thicker than comparable size VCDs and this changes their structural behaviour. The increased stiffness of a thick bulkhead leads to a changes from a flexural (bending) response to more of an arching-type action to
resist and transfer loads to the ribs, roof and floor. As a consequence, more of the load is carried by compression forces in a “flat arch” configuration rather than relying on tensile and compressive strength as when a plate bends under pressure.

In order to ensure effective transfer of these high compressive forces into the surrounding strata, it is good practice to key the bulkhead into the roof, ribs and floor removing distressed material as a result of roadway relaxation. This also provides an improved seal around the perimeter of a bulkhead to reduce water leakage. Bulkheads should be keyed in all around, with a minimum keyway width of 300 mm and key depth of 200 mm into coal or soft rock including thinly laminated strata in which there is often little cohesion between plies. Keying into hard rock is less critical and a nominal 25 mm key or “heavy scabbling” is considered sufficient in most cases.

**Key aspects of 30 metre concrete water bulkhead design**

Practical engineering design always seems to be more complicated than the relatively straightforward design process outlined above and this was the case for the Oaky No.1 mine 20 m and 30 m bulkheads.

In order to allow for variations in the roadway size due to normal construction tolerances and to provide an additional margin for error, a design roadway size of 3.6 m high x 6.0 m wide was adopted. Based on previous experience, a preliminary design thickness of 1250 mm was selected as being suitable for a design water pressure of 120 m (approx. 1.200 kPa). The design pressure included an estimated actual water head of 30 m together with a safety factor of 4 as stipulated by the mine.

The Strand 7 plot in Figure 11 shows a vertical cross-section through the 3.6 m high bulkhead. The goaf is to the left and the arching action of the bulkhead is clearly visible as the green coloured stress plot. Also visible is the high stress at the “base” of the arch, which indicates that these are the critical locations for design strength of the bulkhead and clearly show the importance of having sound strata surrounding each bulkhead. The maximum brick stress shown in this example, 15 MPa, is less than the 16 MPa limit for 40 MPa concrete as provided in AS3600.

**Figure 11 - Stresses in vertical wall section**

Figure 12 shows an outbye face view of the bulkhead, and the peak stresses can be seen to be concentrated in the middle top and bottom edges. Lower stresses are evident at the ribs and this is partially due to the rectangular shape of the bulkhead and partially to the lower stiffness of the coal ribs.

Regular inspection of seals and bulkheads is a very important part of their successful long-term performance and this plot shows the key areas of this particular bulkhead that should be inspected for early signs of distress. It also shows why it is considered good practice to keep cast in pipes at least 600 mm from the edges of a bulkhead to avoid high stress zones.
It is of interest to note that the corners of the bulkhead are not highly stressed. Sometimes bands of stress can be seen at 45° "shortcutting" around the corners.

Figure 12 - Stresses in outbye face view of bulkhead (load on opposing face)

The view shown in Figure 13 is a horizontal section taken looking down on the bulkhead, with the goaf at the bottom of the view. Although the highest stresses occur in the 3.6 m high direction, this plot also shows arching action occurring horizontally. So the bulkhead is trying to form a "dome" shape and this can also be seen from plots for the deflection of the bulkhead, although deflections are typically very small. The neat, simple computer models depicted are only an approximation of the behaviour of real bulkheads. One obvious omission in the figures is the lack of any coving or fillet at the corners where the bulkhead contacts the roadway, as is usually when the surrounds of the bulkhead are finally shotcrete lined. This coving provides extra strength and in practical terms will reduce the high edge stresses predicted by the computer modelling. Since the computer model is therefore conservative, that inaccuracy is accepted as it makes modelling quicker and easier and will produce a Safe design.

Figure 13 - Bulkhead stresses looking down on bulkhead from the roof

CONCLUSIONS

Previously in longwall blocks 1 to 8, water storage and control was achieved using 10 m head capacity bulkheads and borehole submersible pumps, relying heavily on the serviceability of this equipment. In the vicinity of Maingate 20A the strategy has been to increase the bulkhead capacity up to a maximum of 30 m reducing reliance on submersible pumps. In the event of a major rain event the construction of these bulkheads has formed a much larger underground water reservoir capacity, giving increased time to remove excess water as part of the mine water management plan. This gives the operation greater flexibility given potential environmental restrictions on water discharge quantities and quality into natural waterways.

Because of the potential for relaxation of strata due to the presence of coal cleat and joints sets at bulkhead sites, two key construction techniques have been employed; keying and strata injection. As
organic resins have many advantages, a PUR injection program was implemented prior to and post bulkhead construction. Site injection quantities up to 2.6 t (2.15 m$^3$) have indicated the available void space at sites with potential leakage paths being sealed including the concrete/strata interface. Keying would be aided by the use of mechanical excavation equipment.

Bulkheads were designed to a maximum water head of 1200 kPa with a required safety factor of 4. Numerical analysis using Strand 7 software has shown that no part of the bulkhead concrete has superimposed stresses that would result in failed material. Steel skeleton bolts provide an additional key and resistance against shear or sliding failure of each bulkhead.

Continual monitoring of bulkheads and implementation of the rain event TARP will ensure the safety of the mine operation.

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HOT ENVIRONMENT- ESTIMATION OF THERMAL COMFORT IN DEEP UNDERGROUND MINES

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ABSTRACT: Underground mines are a special type of environment which requires significant attention since they directly relate to miners’ productivity, health and safety. With coal or ore exploitation into deep underground mine environments, high temperature and humidity creates risks because of large amounts of heat generated from geothermal heat, mining machines, groundwater and mine water. There have been many assessment methods for underground mine environment, such as dry-bulb temperature and wet-bulb temperature. However, none of the current methods can comprehensively evaluate the underground mine environment since most methods consider only one or a few defined factors and neglect others. In order to evaluate the human body’s thermal status in terms of both personal and environmental viewpoints, this investigation firstly discusses the new upper limits of dry-bulb temperature with the support of some simulated results of indicators from ISO7933, and then adopts both mean skin temperature and dry-bulb temperature to establish a zone diagram. The calculation of mean skin temperature is based on the heat balance of the human body with the ambient environment. Then existent regulations on underground mine environments and some typical condition parameters in underground mines are applied to circumscribe different thermal zones. The established thermal zone diagram defines the preferable condition zone, acceptable condition zone, and prohibited zone. This method considers parameters from both the human body and the surrounding environment and hence offers a more comprehensive estimation.

INTRODUCTION

In comparison with other built environments in terms of the quantification of key parameters affecting air quality, underground mines have seemingly paid more attention to traumatic injury, noise, radon daughter exposure, infra-red exposures in pyrometallurgical processes (Donoghue, 2004) but less attention to human thermal comfort. Hot and humid environments are encountered in deep underground mines, where the virgin rock and air temperatures increase with depth due principally to the geothermal gradient, increasing air pressure from auto-compression of the air column, and groundwater. Mine water transfers heat and vapour to the air by evaporation, and the air receives the heat liberated by mining machinery and equipment as well as less important sources of heat including oxidation processes, human metabolism, explosive blasting, rock movement, and pipelines (Hartman, et al., 1997).

In China the average temperature of the original rock in state-owned coal mines ranges between 35.9 and 36.8°C at the production level while the average mining depth was about 650 m by the year 2000, that would result in dry bulb temperatures exceeding 30°C and humidity of 95% to 100% in working faces (He, 2009). However, in Australian coal mines increasing strata temperatures at relatively shallow depths in combination with high surface ambient temperatures, e.g., in Queensland, have led to uncomfortably high ventilation temperatures on longwall faces. Wet bulb temperatures exceeding 30°C and humidity of 95% to 100% on longwall faces were reached due to the added heat from broken coal and rock on the face and in the goaf together with the heat dissipated by higher capacity longwall equipment (Mitchell, 2003). In South Africa, as the ore reserves for gold are now only to be found in deeper deposits, the current research contemplates depths of up to 5 km and beyond, and heat loads caused by increasing mining depths and the geothermal gradient. Webber-Youngman (2007) states that optimisation of energy resources through active control and predictive simulation modelling is possible, which could lead to establishing a safe, healthy and productive working environment in hot underground mines in South Africa.

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Currently widely-used evaluation indices for the human body’s thermal conditions in hot and humid environment, include, but are not limited to dry and wet-bulb temperature, Effective Temperature (ET) (ASHRAE, 2005), Wet-bulb Globe Temperature (WBGT) (ISO7243, 1986), Heat Stress Index (HSI), work and recovery heart rate, body temperature (NIOSH, 1986). However, some of these indices have not been improved for a long period and are lack of theoretical basis. For example, the coal mine safety regulations of China, provide that the ambient dry-bulb temperature of working face should not be higher than 28°C (SAWS and SACMS, 2006), which is determined from practical working experience and have existed for tens of years. Furthermore, none of the above parameters can comprehensively reflect the thermal status of miners. For instance, dry or wet-bulb temperatures, ET, WBGT, and HSI tend to focus on the environmental conditions without considering a subject’s actual thermal status, the rhythm of the heart or body temperature only shows the conditions of a human body without directly considering the interaction of human skin or respiration with the environment (Epstein and Moran, 2006).

For wet bulb temperatures exceeding 30ºC and humidity nearly saturated in working environments underground, it is necessary to improve and update the existing indices, like dry-bulb temperature, to determine what conditions can be regarded as acceptable, as well as the criteria and limits that should be adopted in assessing them. More importantly, an evaluation method that considers parameters from both the human body and the ambient environment is highly needed.

This paper aims to update the upper limit of dry-bulb temperature of working faces underground and develop a new method to comprehensively evaluate the thermal status of a miner in the underground environment by adopting the miner’s metabolic rate, mean skin temperature and dry bulb temperature.

**CALCULATION OF THE MEAN SKIN TEMPERATURE**

When a human body is at heat balance with the ambient, the metabolic heat generation holds the following expression,

\[ M = C + R + B + E + K + W, \text{ W/m}^2 \]  

(1)

Where:
- \( M \) is the metabolic heat generation rate;
- \( C \) is convective heat exchange;
- \( R \) is radiative heat exchange;
- \( B \) is heat loss by respiration;
- \( E \) is evaporative heat loss;
- \( K \) is heat exchange by conduction; and
- \( W \) is the mechanical work.

According to field measurement and analytical studies, conduction heat loss and mechanical work attribute a relatively small portion to the underground mine environment (McPherson, 1992), so \( K \) and \( W \) become negative. Therefore Equation (1) may be simplified as,

\[ M = C + R + B + E, \text{ W/m}^2 \]  

(2)

Equation (2) can be expanded term by term (McPherson, 1992) into:

\[ M = \frac{t_{sk} - t_a}{R_\alpha + 1/(f_c h_i)} + f_{eff} \varepsilon_{sk} h_i \left[ t_{sk} - \frac{R_\alpha (t_{sk} - t_a)}{R_\alpha + 1/(f_c h_i)} - t_a \right] + 1.7 \times 10^{-8} \times M (S_{out} - S_{in}) + \omega h_i f_{ec} (p_{sk} - p_a) \]  

(3)

Where; \( t_{sk} \) is the mean skin temperature, \( t_a \) is dry-bulb temperature, \( R_\alpha \) is clothing thermal resistance, \( f_c \) is factor of clothing area, \( h_i \) is convective heat transfer coefficient, \( f_{eff} \) is factor of effective radiation area; \( \varepsilon_{sk} \) is skin emissivity, \( h_i \) is the linearised radiative heat exchange coefficient, \( t_i \) is the mean radiant temperature, \( S_{out} - S_{in} \) is the difference of sigma-heat between exhaled and inhaled air, \( \omega \) is skin wetness coefficient, \( f_{ec} \) is clothing permeability factor for vapour transfer, \( p_{sk}, p_a \) are the saturated partial vapour pressure at the skin and ambient, respectively (Waclawik and Branny, 2004).
Equation (3) is an implicit equation with respect to $t_{sk}$ since $h$, $\omega$, and $e_{sk}$ all rely on $t_{sk}$. An iterative guess-correction numerical method may be applied to solve Eq. (3). Because a guessed $t_{sk}$ may not satisfy Equation (3), the difference between metabolic heat generation and total heat loss is:

$$
\Delta Q = M - (B + C + R + E) = M - \left[1.7 \times 10^{-4} M (S_{aw} - S_{sk}) + \frac{t_{sk} - t}{R_3 + \frac{1}{f_{pa}}} + f_{sh} \left(\frac{t_{sk} - t}{R_3 + \frac{1}{f_{pa}}} - \frac{R_3(t_{sk} - t) - t}{R_3 + \frac{1}{f_{pa}}}ight) + \alpha h f_{sh}(p_{aw} - p_d)\right]
$$

(4)

The correction temperature may be based on the linearised change rate of $\Delta Q$ with respect to $t_{sk}$ as,

$$
\Delta t_{sk} = \frac{\partial \Delta Q}{\partial \Delta t_{sk}}
$$

(5)

With an appropriate guess of $t_{sk}$, Equation (3) can be solved by updating the mean skin temperature with Equation (5) until $\Delta Q$ approaches zero.

For the working faces of underground mines that hot and humid, the air humidity sometimes can be close to saturation. This paper selects 90% and 100% as the relative humidity values when calculating thermal comfort indexes of miners. Miners work underground under moderate or severe physical labor intensity, so 180, 200, 220, 240, 260, 280, 300 W/m$^2$ and excessive physical strength 320 and 340 W/m$^2$ are selected as the amount of miners' metabolic values. The miners wear thin pants and sleeved T-shirt, so $R_3 = 0.093$, $f_{sh} = 1.18$, and taking $v = 1$ m/s, $t_a = t_s$.

Taking all the conditions above into Equation (1) to Equation (5), the curves of mean skin temperatures under the condition of RH = 90% and RH = 100% are generated as shown in Figure 1.

![Figure 1](image)

**Figure 1 - Simulated results of mean skin temperature in the working face, (a) RH=90%, (b) RH=100%**

**A NEW INDICATOR OF DRY-BULB TEMPERATURE AND ITS VERIFICATION**

From Figure 1, it can be seen that, when the ambient air relative humidity is 90% and the miners' metabolism is within the normal range, as long as the dry-bulb temperature does not exceed 28°C, the mean skin temperatures are below 36°C, which fully meets the thermal comfort requirements. When the relative humidity reaches saturation and the metabolism is lower than 260 W/m$^2$, as long as dry-bulb temperature is under 28°C, it is can be seen that the mean skin temperature is below 36°C. A new upper limit of dry-bulb temperature is taken as 28°C.

ISO7933 provides the limit values of subjective heat stress indexes which can distinguish the miners who are acclimatised to the working environment or not, such as sweating rate, skin wetness, and upper limit of working time (ISO7933, 1989). When the indicators are below the warning line, a healthy worker may work without any danger. So this study selects sweating rate $SW$, skin wetness $\omega$, and the upper limit of working time $T_{up}$ as the subjective indicators to verify the new upper limit of air dry-bulb temperature, which gives:

$$
SW = E/\eta
$$

(6)

Where $SW$ is sweating rate, g/h; $\eta$ is sweat evaporation efficiency.
\[ \eta = 1 - 0.5 \exp[-6.6 \times (1 - \omega)] \]  
\[ \omega = \frac{E_{\text{req}}}{E_{\text{max}}} \]  

Where \( E_{\text{max}} \) is the maximum evaporative heat loss, equivalent to the heat flux from all wet human skin; \( E_{\text{req}} \) is the evaporative heat dissipation for the human body. Because the breathing heat shares a very small proportion, so it is always be neglected, \( E_{\text{req}} = M \times (C + R) \).

\[ T_{\text{up}} = \frac{60D_{\text{max}}}{SW} \]  

Where \( D_{\text{max}} \) is the maximum dehydration capacity of a miner for one day, \( D_{\text{max}} = 3900 \text{ g} \) for a miner who has adapted to the environment.

Table 1 lists the values of heat stress indexes of ISO7933 which are used to identify if workers are adapted to the environment (ISO7933, 1989).

<table>
<thead>
<tr>
<th>Criteria</th>
<th>Non-acclimatised</th>
<th>Acclimatised</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>warning</td>
<td>danger</td>
</tr>
<tr>
<td>Maximum sweat secretion (g/h)</td>
<td>520</td>
<td>650</td>
</tr>
<tr>
<td>Dehydration (g)</td>
<td>2600</td>
<td>3250</td>
</tr>
<tr>
<td>Critical skin wetness</td>
<td>0.85</td>
<td>1</td>
</tr>
</tbody>
</table>

Taking different conditions of temperature and humidity into Equation (6) to Equation (9) with different amounts of metabolism of miners, the curves of sweating rate, skin wetness, and upper limits of working time can be obtained, as shown respectively in Figure 2 to Figure 4.

**Figure 2 - Simulated results of sweating rate in the working face, (a) RH= 90%, (b) RH=100%**

**Figure 3 - Simulated results of skin wetness in the working face, (a) RH= 90%, (b) RH=100%**

It can be seen from Figure 2 that when the relative humidity of the work environment is 90%, and the amount of metabolism generation by the miners is no more than 320 W/m², if the dry-bulb temperatures are below 28°C, the sweating rates of miners are below the warning lines of ISO7933 for the miners who...
are non-acclimatised to the environment, 520 g/h;  When the relative humidity reaches saturation, the working strength is less than 280W/m², if the dry-bulb temperature is below 28°C, the indicator can also be met.

Figure 3 shows that when the relative humidity of the environment is 90%, and the amount of metabolism generation of miners is no more than 300 W/m², if the dry-bulb temperatures is below 28°C, the skin wetness of miners is below the warning and danger lines of ISO7933 for the miners who are non-acclimatised to the environment, 0.85;  When the relative humidity reaches saturation, the working strength is less than 260 W/m², if the dry-bulb temperature is below 28°C, the indicator can also be met.

Figure 4 shows that when the relative humidity of environment is 90%, the amount of metabolism generation of miners stay at less than 280 W/m², if the dry-bulb temperatures is below 28°C, the working time of miners underground can meet the rule of six hours for one working day, especially, miners’ working time underground with metabolism of 300 W/m² might reach five hours;  When the relative humidity reaches saturation, and the working load is less than 260 W/m², if the dry-bulb temperature is below 28°C, the indicator can also be met.

To sum up, when the relative humidity is 90%, and the amount of the miners’ metabolism generation is approximately less than 300 W/m², if the dry bulb temperature does not exceed 28°C, these three indicators of heat stress mentioned above can meet the thermal comfort conditions;  When the relative humidity reaches saturation, and the working strength is no more than 260 W/m², and the dry-bulb temperature does not exceed 28°C, the three indicators mentioned above can also be satisfied.  As a result, they verify the correctness of the new upper limit of dry bulb temperature that can be extended to 28°C in China.

**CIRCUMSCRIPTION OF THERMAL ZONES**

According to Fanger's theory (Fanger, 1972) on human body, skin temperature is closely related to thermal sensation and hence it can be a key parameter to indicate a human body's thermal status.  There are seven parameters that can exert impacts on a person's heat balance, therefore on skin temperature.  They are metabolism (M), work (W), clothes thermal resistance (R_c), air temperature (t_a), mean radiant temperature (t_r), air velocity (v), water vapour pressure (p_v), where the former three parameters reflect a human's personal condition, whereas the latter four or any of their combinations indicate the surrounding environmental condition.  It seems possible to apply the skin temperature to describe a person's thermal status.

A person’s mean skin temperature is dependent on the complicated heat transfer governing equation that involves the aforementioned seven parameters in implicit format.  However, such implicit heat transfer equation is very hard to be applied to tell the miners’ thermal status.  Diagrams can provide a convenient method of assessing human body’s thermal comfort.  Figure 5 illustrates the concept of a thermal zone diagram, where the horizontal and vertical coordinates represent parameter 1 and 2, respectively, while curve 1 represents the effect of the rest parameters (concentrated to parameter 3) to the correlation of parameter 1 and 2.  An acceptable zone can be defined with the circumscription of line 1, line 2 and curve 1, which means lines 1 and 2 are the upper limits of the acceptable conditions,
respectively, and curve 1 is the lower limit. If a miner and the underground environment hold conditions within the acceptable zone, it implies this miner can work without inducing thermal symptoms.

Figure 5 - The concept of a thermal condition zone diagram

The following summarises the development of a thermal zone diagram in underground mine environment using dry-bulb temperature as parameter 1, mean skin temperature as parameter 2, and other parameters concentrating to parameter 3. From Equation (3), it can be seen the mean skin temperature is dependent on $M$, $t_a$, $R_{cl}$, $t_r$, $v$, and $p_a$. It is noted that air velocity, $v$, determines convective heat transfer coefficient, whereas water vapour pressure $p_a$, affects evaporative heat loss. These six parameters are usually in certain ranges in typical underground mine environment, so they can be applied to circumscribe thermal zones in terms of $t_{sk}$. The six parameters are discussed briefly as follows.

Metabolic production $M$

Underground manual work is usually moderate or heavy, with average metabolic rates normalised at 245 W/m$^2$ or 340 W/m$^2$ respectively (Mcpherson, 1992). In this paper, the metabolic rate level of 300 W/m$^2$ may be considered as the upper limit.

Dry-bulb temperature $t_a$

Article 102 of the Chinese Coal Mine Safety Regulations sets the upper limit of airflow temperature at 26°C in working faces. If the air temperature in working faces exceeds 30°C miners should cease work (SAWS and SACMS, 2009). According to the former analysis, the average temperature of 28°C is set as the upper limit of air dry-bulb temperature.

Clothes thermal resistance $R_{cl}$

Thin trousers and long-sleeved shirt which are commonly worn in underground workings are selected to assess the effective thermal resistances of clothing ensembles. This thermal resistance is 0.093 (m$^2$K)/W and its typical corresponding area factor is 1.18 (Mcpherson, 1992).

Mean radiant temperature $t_r$

The mean radiant temperature depends on the actual thermal status of rocks, heat release of mining machines and airflow condition. In underground mine, the mean radiant temperature will not differentiate much from the ambient air temperature due to strong flow condition and thus extensive heat transfer between hot surfaces and the ambient air in typical mines. Therefore one may set $t_r=t_a$ for simplicity.

Air velocity $v$

In order to condition underground mine air temperature, and also to dilute methane, carbon dioxide and other harmful gases, air velocity underground is usually maintained in the range of 0.25 m/s to 4 m/s (SAWS and SACMS, 2009). This paper has set 0.25 m/s as the lower limit and 4 m/s as the upper limit for airflow velocity to be considered.
Water vapour pressure $p_a$

Water vapour pressure affects latent heat transfer. Air in working faces is nearly saturated, with relative humidity commonly ranging from 90% to 100%. Water vapour fraction pressure can be calculated from relative humidity.

Evaporative heat loss $E$

Latent heat transfer in underground mines is mainly in the form of sweat evaporation. To prevent dehydration and thermal fatigue, ISO7933 requires the maximum allowed amount of dehydration for manual workers $D_{\text{max}} = 3900$ g (equivalent to sweating heat release of 1.5 kWh/m$^2$ (Waclawik and Branny, 2004) for a working day (6-8 hours) (ISO7933, 1989). Hence in this paper, we chose $E=1.5$ kWh/8h=187.5 W/m$^2$ as the upper limit.

Table 2 summarises all thermal condition parameters discussed above to circumscribe thermal zones.

<table>
<thead>
<tr>
<th>Parameters</th>
<th>$M$/Wm$^{-2}$</th>
<th>$t_a = t_r$/°C</th>
<th>$R_c$/m$^2$K/W</th>
<th>$t_u$</th>
<th>$\nu$/m s$^{-1}$</th>
<th>$E$/Wm$^{-2}$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Values</td>
<td>≤300</td>
<td>≤28</td>
<td>0.093</td>
<td>1.18</td>
<td>0.25-4.0</td>
<td>≤187.5</td>
</tr>
</tbody>
</table>

RESULTS AND DISCUSSIONS

Figure 6 illustrates the plotted thermal zone diagram, where the abscissa shows the dry-bulb temperature and the vertical coordinates the corresponding mean skin temperature. The entire region is divided into five sub-zones, i.e., zone I, II, III, IV, V separated by curve 1, 2, 3, 4 and SW. Zone I is the region below curve 1 and a part of SW, which represents the most favorable condition in underground mines, because at the condition portrayed by curve 1 ($\nu=4$ m/s and RH=90%) it favors heat release from the human body, whereas curve SW represents the mean skin temperature under the maximum sweating heat release of 187.5 W/m$^2$ to avoid excessive dehydration. Therefore, zone I represents that if the dry-bulb temperature is within 28° C, and the mean skin temperature is lower than the prescribed temperature on curve 1 and curve SW, there should be no heat stress risks.

Figure 6 illustrates the plotted thermal zone diagram in underground mines (plotted under metabolism production of 300 W/m$^2$).

Above zone I is Zone II, which is circumscribed by curve 1, 2 and a part of curve SW. Curve 2 represents the condition when the ambient air reaches saturation at 4 m/s. Zone III is confined by curve 2, 3 and a part of SW, where curve 3 is at the lower limit of air velocity at 0.25 m/s, but at a relative humidity of 90%. Similarly, Zone IV is confined by curve 3 and a part of SW. Curve 4 represents the most adverse condition, because the air movement was very slow with saturation state on this curve. Curve 4 is above curve SW, although the mean skin temperature in actual sites may possibly fall between curve SW and 4, it has violated the regulation of the maximum allowable dehydration prescribed on curve SW. Zone V represents the prohibited status for miners, and under any circumstances, the mean skin temperature should not fall in zone V for a long time to prevent miners from any heat stress.

In summary, from the viewpoint of human thermal condition, zone I represents the most preferable working condition for miners, where heat stress symptoms should not emerge; From zone II to zone IV, the mean skin temperature increases, therefore the risks for heat stress symptoms also increase although they are still in the acceptable zone; Under any circumstance, the mean skin temperature should not be in Zone V for safety. As Figure 6 was plotted under the maximum allowable metabolic
rate, the evaluation of a miner’s thermal condition under other smaller metabolic rates with Figure 6 will be conservative.

CONCLUSIONS

This paper analyses and summarizes the thermal stress evaluation indexes of ISO7933, from the simulated results of mean skin temperatures of different environmental humidity and labour intensity, it can be recommended that the upper limit of dry-bulb temperature of working faces in China Coal Safety Regulations could be extended from 26°C to 28°C under certain environmental conditions. The accuracy of the new indicator is verified by simulation results of subjective evaluation indicators, including sweating rate, skin wetness and the upper limit of working time.

In order to evaluate the human body’s thermal status from both personal and environmental conditions, this investigation has adopted both mean skin temperature and dry-bulb temperature to establish a zone diagram. Existing regulations on mine environments and some typical condition parameters in underground mines are applied to develop a diagram of thermal condition zones, which define the preferable condition zone, acceptable condition zone, and prohibited zone for workers working in underground mines. The thermal zone diagram method may also be used for evaluating building environments.

ACKNOWLEDGEMENTS

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MINING GASY COALS

Ian Gray

ABSTRACT: This paper reviews the basic factors and practice of mining gassy coals worldwide. It then suggests how changes need to be made to Australian mining methods to deal with the challenges of mining deeper and gassier coals.

THE BASICS

Gassy coals are coals that generally contain methane generated in the coalification process but may contain carbon dioxide. Other gases may also be present in low concentrations. Carbon dioxide is normally introduced to the coal through igneous events. Many coals are wet and have a reservoir fluid pressure that is higher than the pressure at which gas would be released from the coal, referred to as the sorption pressure. This pressure is akin to the bubble point of a conventional oil reservoir which contains solution gas. The gases in the coals are principally stored by a process of multilayer adsorption. In the water saturated state the water in the coal contains solution gas. Some coals are, however, not water saturated and has free gas in their pore space. From a reservoir viewpoint the free pore space in coals is that which is interconnected in a cleat network.

Cleating

The cleat in coals is generally well recognised in Australian and US coals and usually takes the form of a fracture network which is near perpendicular to the seam. There is generally a major or face cleat and a butt cleat which is approximately orthogonal and less well developed. The cleats may be open or filled with clays, carbonates or other minerals depending on the history of the coal. This cleat pattern is however not a universal trend. Many coal seams in Russia and China have a cleat pattern that is not developed to any extent, or where present, is quite irregular. The presence or otherwise of a cleat pattern has a major influence on the mechanical properties of coal and in particular its drainage characteristics.

What is a gassy coal?

The definition of a gassy coal is a relative term. An Australian bituminous coal used for coking purposes might be regarded as gassy if its gas content is 10 m$^3$/t. If this gas content existed in a semi anthracite from Shaanxi Province in China it would not be considered to be particularly gassy. The reason for this is, in part, the sorption isotherm characteristic of the coal. This describes the volume of gas stored in a coal as a function of pressure at the seam temperature. Frequently, this gas storage is approximated by the Langmuir Equation (1).

\[ V = \frac{V_L P}{P + P_L} \]  

Where:
- \( V \) = Volume of gas stored in isotherm, m$^3$/t;
- \( V_L \) = Langmuir Volume, m$^3$/t;
- \( P \) = Pressure, MPa;
- \( P_L \) = Langmuir Pressure, MPa.

More gas is stored in coals of higher rank for a given pressure. The higher the moisture level of the coal the lower the gas storage capacity. This may be seen as being due to the water and methane competing for storage on the coal surface. Sorption isotherms are generally regarded as being reversible and can be generated by measuring the uptake or release of gas from coal over a range of pressures. While Mavor (1990) suggests that the moisture content should be normalised to a level controlled by humidity there seems little justification for doing this. Rather, the process of testing at as close to an in situ moisture content as possible is needed. This is generally obtained by testing core as received. When mixed gases exist there is a further level of competition between the gas types and...

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Clarkson and Bustin (2000) describe a process for arriving at mixed gas isotherms using the individual isotherms of the component gases.

It would seem that the order in which the gases came to exist in the coal may have an influence on the storage. The question may well be asked what has really happened to a coal that has generated methane under wet conditions; has lost most of that gas due to groundwater movement; has then had a carbon dioxide rich hydrothermal sweep move through the seam replacing the methane and leading to carbonate filling of the cleats and a raised temperature; this raised temperature then leads to the generation of more methane. This is a complex but real scenario for many coals.

There is, however, always a level of uncertainty in knowing the true nature of the sorption isotherm as recreated in the laboratory, due to a lack of knowledge of the precise moisture content and due to a lack of knowledge of the history of the coal. The measurement of what is termed the native sorption isotherm is therefore advocated. This is arrived at by placing coal in a sealed pressure vessel, either as part of the coring process or by the use of a sealed core barrel, though the latter is not commonly available. In the former case the pressure vessel is normally a close fit on the core and the small excess volume is taken up with water. Once sealed in a vessel, a pressure equilibrium is reached. Once this has occurred a quantity of gas is released and the pressure is once again allowed to reach equilibrium. This process may be repeated with measurement of the gas and water release and of the equilibrium pressure. As with all sorption isotherm or desorption measurements, this should be undertaken at the reservoir temperature. The correct knowledge of the sorption isotherm is vital in the assessment of whether coal is gassy or not.

Diffusion characteristics

Knowledge of the diffusion characteristics of coals is also important. Coals release gas at quite different rates. This release of gas from coals is generally thought of as Fickian diffusion down a concentration gradient though other types of diffusion such as Knudsen diffusion also may exist. In terms of Fickian diffusion, the rate of gas release from a coal lump is determined by the diffusion coefficient, the concentration of gas (gas content) and its size and geometry. Because coals are always to some degree heterogeneous and may contain fractures, the direct use of the theoretical equations of Fickian diffusion (Crank, 1975) seldom accurately describe the rate of desorption from a lump of coal or a core. They do however, seem to quite accurately describe the diffusion of small particles of coal (Gray, 2011a), when the effects of fractures and heterogeneity are removed.

Characteristically, the diffusion behaviour of lump coals show a more rapid initial gas release rate than would be expected if the coal followed Fickian behaviour. This means that an estimate of the diffusion coefficient made from the rate of early desorption will be of a higher value than one made at a later stage of desorption.

The use of Fickian diffusion equations are, however, the basis for determining the lost gas component of core that is retrieved during wireline coring operations. In this, the gas release is approximated by the first order term of the equation that describes the initial diffusion from a solid cylinder shown in Equation (2).

\[
\frac{V_t}{V_\infty} = \frac{4}{\sqrt{\pi}} \left( \frac{Dt}{a^2} \right)^{1/2} - \frac{1}{3\sqrt{\pi}} \left( \frac{Dt}{a^2} \right)^{3/2} + \ldots
\]

(2)

Where:
- \(V_t\) = Cumulative volume of gas released at time \(t\) (consistent units);
- \(V_\infty\) = Total volume of gas release (consistent units);
- \(a\) = Core Radius, m;
- \(D\) = Diffusion Coefficient, \(m^2/s\);
- \(t\) = Time, s.

The lost gas is typically derived by plotting the cumulative gas release versus the square root of time and projecting the straight line slope back to cover the time over which this gas loss occurs. While this period is usually poorly approximated as half the period it takes to retrieve a wireline core barrel (AS3980) the slope is in itself a very useful measurement. Using the slope and the final total gas volume it is possible to determine the apparent initial diffusion coefficient. This is given in field units in Equation (3).
The apparent initial diffusion coefficient would be a precise measure if the core were a cylindrical solid of radius $a$ with a uniform diffusion coefficient. If the core is substantially free of fractures this is an approximation to the early real value of the diffusion coefficient. However, if the core contains fractures that are significantly more closely spaced than the core diameter then a more useful measure is that of the slope of the plot of initial desorption versus square root of time, divided by the total gas volume and mass of core.

As an alternative to using a diffusion equation, Airey (1968) used an empirical equation to describe the rate of gas release from broken coals. This is shown in Equation (4).

$$\frac{v}{v_0} = 1 - e^{-\left(\frac{t}{t_0}\right)^b}$$

(4)

Where:

- $t_0$ = Characteristic desorption time (consistent units with time, $t$);
- $b$ = Shape factor (dimensionless)

The diffusion equations provide a basis for determining how coal will release gas in the broken state. This is important in determining gas make at the face as coal is cut and fragmented. It is also important in determining the propensity of the coal to outbursts. In either case, the gas release rate is determined by the particle distribution size, total gas content and the diffusion coefficient. In the case of outburst the ability of the coal to deliver gas at pressure into the spaces between fragmenting coal is a significant contributor of energy to the outbursting process.

**Permeability**

Coal permeabilities measured in Australia have been shown to vary over six orders of magnitude, from a few microdarcies to several Darcies. This variation is essentially a function of the frequency, interconnectivity and openness of the cleat network. The coal permeability varies spatially and with effective stress. The state of effective stress is affected by the fluid pressure and state of desorption of the coal as the coal shrinks as it releases gas or dries. The coal's response to shrinkage is highly dependent on the coal's modulus which tends to rise significantly with confining stress. These processes are described by Gray (2011 b). The variability of coals means that their gas conditions are quite different and therefore the methods to mine them must be varied to suit the conditions.

**PROBLEMS WITH MINING GASSY COALS**

The problems posed by gas in coal are several fold and include outbursts, gas make from ribsides into the roadways, gas make from coal when it is being cut, and the production of gas from coal seams that are in the relaxed zone of mining, whether the seams are relatively intact or substantially broken in the goaf. In some cases the gas released into coal mines may come from sources other than the coal itself. Contributors may be carbonaceous shales or porous sandstones.

**Outbursts**

Outbursts are the expulsion of gas and coal from the working face. The violence of the outburst is dependent on the energy release that occurs with it. This is a function of the size of the coal mass affected and energy release per unit volume of that coal. The energy release is derived from the strain energy stored in the coal and rock, the energy of gas stored in free pore space (cleats) and the gas that diffuses from the coal. Some of this energy will be absorbed by fragmentation of the coal. Thus, coals that are already broken such as fine fault gouge material pose a particular problem in that the energy required to fragment them is negligible while their ability to desorb gas quickly is high. This process is explained in more detail by Gray (2006).
The main way to prevent outbursting is to drain gas thus relieving this element of stored energy. The degree to which drainage must be achieved to avoid outbursts is dependent on the gas type, the initial diffusion coefficient, the gas content/pressure as related by the sorption isotherm, and the potential for the coal to fragment. These are the factors that should be taken into account in determining outburst risk. Drainage does not necessarily relieve a highly stressed coal of strain energy release when it breaks. It should also be appreciated that weak coals which are confined may be highly stressed behind a solid face and if that face suddenly fails, this energy may be suddenly released. Nevertheless, despite all these additional factors which should be taken into account, particularly in more stressed coals, the main factor used in determining outburst risk is gas content.

The value of the gas content at which it is safe to mine is dependent on the gas and coal. It may be that it is safe to mine an anthracite coal at 13 m$^3$/t, a value which would be likely to lead to a significant risk of outbursting in a sheared bituminous coal. The reason for this is principally the difference in the sorption pressure and coal toughness. Permeable coals are not generally considered to be a risk from an outburst viewpoint because the gas has drained from them ahead of the mining face.

Gas make into roadways

Gas make from the ribsides into underground roadways is a function of the reservoir characteristics of the coal seam. Many coal seams are wet and the water pressure must be lowered to below the gas desorption pressure before the gas is released from the coal. Once gas is produced in the cleat network, a two phase Darcy flow regime will exist in the cleats. The rate of Darcy flow is determined by the potential gradient and the permeability to either the gas or water phase. The potential gradient contains both pressure and gravitational terms. In many cases the gas release rate is determined by Darcy flow, however in a few coals the gas release rate is limited by the rate of diffusion from the coal solid into the cleat network. This is particularly the case where the cleating is widely spaced and the permeability high. Therefore, unless the rate of gas release is governed by diffusion, the coals that pose the most problems with gas make into roadways are those with high permeability and an adequately high sorption pressure to drive the gas through the cleat network. Gas drainage by in seam drilling is particularly effective in dealing with such situations. In some cases it is necessary to lower the water pressure significantly to achieve gas drainage and care needs to be taken to design the gas drainage system with adequate peripheral drainage to cut off water recharge to the block being drained. It is then possible to lower the seam reservoir pressure to a level that permits gas to be released and drained. Generally, dealing with high permeability coals is a comparatively easy process that can be accomplished not only by in-seam drilling, but also in some cases, by the use of vertical wells.

Coals may also be quite dry and behave as single phase gas reservoirs from initial drainage. This situation has been found in highly permeable coals of the Surat Basin in Queensland where it would appear that the free gas is essentially a gas cap in the traditional petroleum sense in what are otherwise water saturated coal seams. Essentially dry coal seams have also been found to exist in the very tight coals such as those of the Karaganda basin in Kazakhstan.

Gas make on cutting

As coals become less permeable the gas release from the ribsides becomes less and the problem of gas make shifts from the ribsides to that from the coal being cut. The limits of gas content to minimise outburst risk are frequently far lower than those that would be required to permit high production mining. The concern with gas make at the face, whether it is on development or on a producing longwall, is the risk of developing an ignitable mixture. The factors that control the rate of gas production are: the rate of cutting, the coal's gas content, its diffusion rate and the particle size distribution of the cut coal. The latter is determined by the cutting means and the nature of the coal. Some coals will break to produce a fine product no matter what the pick lacing of the cutting head. The measurement of coal core desorption rates is a very important basis for determining what the gas release rate on mining will be.

Experience in Russia would indicate that coals with higher moisture content desorb gas less rapidly. Thus, the use of water infusion may serve a far more useful purpose than dust control. This is worthy of investigation in the context of varying coal types as it may prove to be very important for face gas control elsewhere.
Gas make from relaxed zone

The drainage of the relaxed zone around a mined seam may be the key to being able to continue mining safely at economic rates as very large gas releases may occur from broken strata. If these releases occur continuously they are easier to deal with than where the ground breaks suddenly leading to a large release of gas which may expose the face to an explosive mixture. The general purpose of relaxed zone drainage is to draw the gas into boreholes or drainage galleries under the use of vacuum so as to reduce the face concentration. Care must be taken to not draw in excessive air thus causing the mixture that is being drained becoming explosive. Too much vacuum may also lead to spontaneous combustion if air is drawn through coal. How relaxed zone drainage is achieved is dependent on its exact purpose and the conditions existing in the mine.

One of the important uses of relaxed zone drainage is in draining the longwall block of gas prior to mining. In this situation the longwall block has usually been drilled from the gateroads prior to mining. These holes might serve a function of some pre-drainage. If however, the coal is highly impermeable then the gas quantity obtained in pre-drainage may be minimal. When the longwall approaches it causes a raised abutment stress with subsequent de-stressing as the coal passes its peak strength and yields. Under these circumstances the coal in front of the face may yield the bulk of the gas it contains to the in seam holes. These holes should be operated at a modest vacuum.

Such holes need not just be drilled in-seam. The practise of drilling above or below the seam and waiting for the stress changes to cause relaxation of the strata around the seam to permit gas release is fairly commonly used. The complications with the use of such holes is in keeping them open and in ensuring that they have an adequate gas carrying capacity as the volumes of gas liberated may be very large. In some situations the use of a drainage gallery above the seam being mined is used. This may be used on its own or with drainage holes drilled from it. In the case of either borehole or gallery their success is dependent on how well they stay open and what vacuum may be applied to draw gas away from the workings. To try and achieve the maximum level of openness of hole or drainage gallery these need to draw gas away from the broken ground toward solid ground. Under these circumstances, if the hole or gallery is sheared at the goaf end, then the remainder of it continues to operate.

In some mines goaf drainage can be achieved simply by the use of vertical holes which are usually cased and have vacuum applied to them. Such holes are usually drilled from surface to the location of the goaf for a seam which has not had mining take place above it. Where overmining has been used, drilling of surface gas drainage holes becomes much more difficult as it must pass through old goaf areas. The vacuum level applied to such surface holes is usually automatically controlled so that the gas composition does not fall below a prescribed level. There is no reason why such automatic control should not be applied to underground boreholes other than cost. The technology exists to implement it. The benefits of using such a system to minimise face gas make and minimise the risk of spontaneous combustion caused by drawing air into the relaxed area by excessive vacuum are considerable.

LOW PERMABILITY COALS

Gassy coals that have very low permeabilities pose particular challenges to economic mining. This state of low permeability may be caused by the infill of cleats with mineral matter or may be brought about by the high stress state of the coal. In either case, the only way to drain the coal is to bring about an increase in its permeability. There are essentially only two means to achieve this.

Shrinkage induced stress relief

If effective stress in the coal decreases readily with gas drainage due to shrinkage then inducing initial drainage will lead to improved permeability. This effect can sometimes be quite dramatic as at Leichhardt Colliery in Queensland (Gray, 1983 and 2011b) when the permeability appeared to increase by several orders of magnitude. Such cases are brought about by desorption pressures that are close to reservoir pressure, high shrinkage behaviour, stiff coals and low initial stresses. Where carbon dioxide is the seam gas the shrinkage with gas release may be greater than that with methane. Dartbrook Colliery in New South Wales epitomised this situation. It had a lightly stressed stiff coal with high levels of shrinkage associated with the carbon dioxide seam gas. Here in some locations, where dawsonite (a carbonate) filled the cleat system, the only way to bring about initial drainage was to hydrofracture the coal from in-seam holes. Once drainage was initiated, shrinkage brought about great improvement in permeability and drainage. The coal could then be mined with manageable gas levels.
Mining induced stress relief

Where the factors such as higher fluid pressure in the coal, reduced shrinkage, less stiff coal and sorption pressures that are significantly below reservoir pressures change, the situation may be reversed and the natural trend is for the coal to reduce permeability with drainage. Under such circumstances the method required to decrease effective stress is to remove material.

The traditional practice used in countries with tight coals is to mine one seam to cause relaxation of the adjacent seams thus reducing the stress in adjacent seams. Mining may be accomplished from the top down or from the bottom up. This sort of operation is, or has been, practised in China, Germany, Kazakhstan, Poland, Russia, Ukraine, and the UK. The extent of the zone relaxed by mining the first seam is dependent on the geomechanics of the situation but may typically extend 50 m from the seam being mined. The first seam to be mined is chosen because of its suitability to being worked first. Of prime consideration in this case is a low risk to outbursting brought about by the tough nature of the coal, its lack of geological structure, lower gas content and other related factors. It may also be chosen for its comparatively low working height so that excessive disruption of the adjacent seam to be mined next does not occur, especially if there is little interburden.

Drilling induced stress relief

Alternatives to mining an entire seam for removing material so as to facilitate de-stressing have come in a variety of forms. The most commonly practised has been the drilling of relief holes in a face. These have been drilled anywhere between 0.1 m diameter to the 1.3 m used at the Japanese mine of Akabira in the 1980's. Such holes serve only to drain gas unless conditions exist so that failure occurs leading to fracturing which propagates from the borehole. To do this the stresses must be sufficient that they cause significant failure of the coal around the boreholes. The practice of washing outburst prone material from gouge zones has been tried in what was the Soviet Union but without the protection of some well control system. Drilling holes without some form of well control has led to uncontrolled emissions and outbursts.

In Australia, the process of water jet drilling followed by slot cutting to relieve stress has been tried. This has apparently not been successful because gas content remained high in the environment in which it was tried. There is however, no reason to reject the technique simply because it has not been successful in a single application, there needs to be an assessment of the reasons why the system did not work under the specific conditions that existed where the trial took place.

WHAT NEXT?

Coal mining has taken place in deep, highly stressed and gassy conditions for many years in a number of countries and methods have been developed for working in such conditions. Australia has developed underground mining in conditions that are, by comparison, relatively shallow and not as gassy as many other operations when considered on a world scale. The standard Australian approach has been to drain gas by in-seam drilling to lower the gas content to avoid outbursts or to de-gas a block prior to longwall mining. This has been permitted by permeabilities that are generally high enough (>1 millidarcy) that such techniques have been successful. The permeability has furthermore been seen to increase with drainage in many of the shallower coals. These conditions have allowed some gassy Australian coal mines to be highly productive.

Mines are now being planned to be worked at greater depth than is current practice. This means that instead of 600 m in New South Wales and 400 m in Queensland consideration is being given to working at 800 m and deeper. These depths bring with them challenges in terms of high stresses, higher fluid pressures and in some cases much lower permeabilities. Australian coal mining has seen glimpses of what may await it as it goes deeper through zones in existing mines that have had either extremely low permeability or borehole collapse, or both. There have also been situations where the permeability declines with drainage as fluid pressure is lowered and effective stress increases. It should be pointed out however that while there is a general trend to declining permeability with depth there are many coals that do not follow this trend as evidenced by experience in the testing of deep coals for commercial gas production.
Current Australian approaches to dealing with high gas and low permeability conditions have been to drill many holes through the zones that will not readily drain, to attempt hydrofracture, to remotely mine or to use permitted explosives to shotfire through difficult coals.

The drilling of multiple holes has been found in some cases to be completely ineffectual, either because the holes close or because the coal is too tight to permit any significant drainage. The consequence is large delays in production while the mine re-adjusts its plan to negotiate around the difficult zone. The problem here is as much a failure to take into account the real risk factors for outbursting as a lack of technology to deal with the drainage problem.

The use of hydrofracture from in-seam holes was applied successfully to Dartbrook mine in the Hunter Valley of NSW. This was however a shallow mine with strong coal and a high shrinkage that led to enhanced permeability. Attempts at hydrofracture in the Bulli seam have been unsuccessful because the coal has inadequate strength to support a packer and because casing in-seam is impractical both from the viewpoint of the difficulty in doing so and because leaving steel in seam is highly undesirable for future mining.

The use of remote mining has some benefits as it removes personnel from the immediate zone of outburst risk. However the possibility of the occurrence an outburst that will send a slug of explosive gas through the mine is a real possibility. The same comment applies to shotfiring through outburst prone coals except that there is the additional risk of an outburst catching unaware crew returning to the face after the shot. In either case, the potential exists for an outburst to come from the ribs behind the face as occurred behind a road header at Pervomayskaya mine in the Kuzbass in 2005.

Dealing with the outburst challenge

Australian mining has used gas content alone as an indicator of outburst proneness. While this approach has served the industry well in terms of there being no outburst fatalities for many years, it should be recognised that it is a simplistic approach developed empirically for the Bulli seam. There are many other factors apart from mere gas content that contribute to the occurrence of an outburst. To prevent outbursts occurring under changing conditions and indeed to save the industry the large cost due to unnecessarily lost production it is very important that new approaches be taken to determine the true risk of an outburst occurring. A system based on the total net potential energy available within a coal mass as being a far better indicator is advocated. This will take into account strain energy, adiabatically expanding gas derived from desorption or pore space, and the toughness of the coal. It is considered that this approach can be made sufficiently straightforward that errors due to its increased complexity can be avoided.

Safely mining an initial tight gassy seam

The problems associated with mining a single gassy seam or the first of multiple seams remain essentially the same. The seam must be mined safely and economically. In the case where the seam being mined is the first of multiple seams it may be considered that the de-stressing being brought about by mining it is the key to mining the adjacent seams and therefore the economics of mining this seam should be considered in the context of the entire mine economics. In this case, the first seam to be mined may be considered as being an essential part of the mine degassing and mining process.

Pre-drainage for development

The first consideration in mining is to be able to develop in safety. This means that outbursts should be prevented. Potential energy stored within the coal needs therefore to be brought to a safe level. A key component in this is reducing the gas level. If the coal has a very low permeability the drainage will be very slow and some process will be required to stimulate flows. If in addition, boreholes will not stay open because the stresses in the coal are sufficiently high, and the coal strength inadequate, then borehole collapse will occur. Under these circumstances the drilling of in seam holes for gas drainage is a pointless exercise. If however, conditions exist whereby large scale failure is brought about by drilling then, provided there is adequate control over gas and material discharge from that hole, systems such as large augers or water jet erosion may be considered as a useful tool for de-stressing and degassing, albeit on a local scale. Such localised techniques have great potential to permit mining to negotiate gouge zones that would be particularly outburst prone.
In the more general context of a highly impermeable coal seam that cannot be drilled because of hole collapse and is highly gassy, alternative means to drain the seam are needed. Hydrofracturing has great promise because the fracture is held open by the use of a propant, usually sand. The problem of hole collapse may, under the right circumstances, be avoided by not drilling in the seam the hole from which the hydrofracturing process is conducted. Rather the hole may be drilled in rock adjacent to the seam provided that the rock is of adequate strength to withstand the stresses therein. Doing this enables the hole to be cased and cemented so as to maintain its integrity. The casing can then be perforated and hydrofractured in multiple stages. By this process hydrofractures can be created at intervals that might be as close as 6 m. For the process to be successful the hydrofracture needs to propagate into the coal seam and transsect the seam.

The fracture needs to propagate from the borehole to the seam in a direction that is usually vertical. If it were to extend horizontally then it would not extend into a horizontal seam. For it to propagate vertically from the borehole the minimum effective stress needs to be horizontal. This is a function of the local stress regime which is in turn dependent on the weight of the strata, the Poisson's ratio and Young's modulus of the rock and the tectonic strains that exist (Gray, 2011b). If the tectonic strains are sufficiently high in the horizontal direction and the rock is stiff then there is an increasing tendency for the horizontal stresses to be high. Rocks with high Poisson's ratios also tend to have higher horizontal stresses.

The development of the fracture from the borehole is dependent on the stress regime in the rock and nature of stress concentration around the borehole or the perforation. Good perforating techniques lead to reduced fracture initiation pressures and the rapid development of a fracture that is orthogonal to the minimum principal stress. Such a fracture will extend evenly outwards from the borehole until it reaches a lower stressed rock such as a coal in which it will thereafter preferentially propagate.

The techniques to permit such an operation have substantially been developed for the tight gas and shale gas industries. Their use in tight coals will however require some adaptation as it is envisaged that the fracture spacings are likely to be much closer than in the above industries. However, the need to extend the fractures is likely to be much less than those required in shale gas industries as the prime requirement is likely to be the degassing of development roads. It is envisaged that surface to in-seam drilling could be used with such a hydrofracture system for initial development but thereafter the process would be handled more cost effectively by drilling and casing from underground, quite possibly from roadways created in rock specifically for the purpose of drainage. In this case, the cement for casing, perforating jet sand fluid and hydrofracture sand fluid mixtures could be pumped at the surface and reticulated to the underground via a system of cased holes and high pressure pipework.

Whether the coal permeability decreases or increases with drainage merely affects the choice of the spacing of the hydraulic fractures along the borehole length so as to achieve drainage within a specific period. Too close a spacing may however lead to fractures coalescing and not adequately extending laterally.

Pre-drainage of the production block

Once development may be undertaken in safety then the problems of gas make on cutting need to be addressed. While it would be possible to employ the hydrofracturing techniques discussed to de-gas an entire longwall block this may not be economic. If boreholes, which stay open, can be drilled in the coal and abutment stress changes create a de-stressed permeable zone in advance of the face then it may be possible to utilise this technique to drain the block with vacuum ahead of the face. This technique is not, however, applicable to the start of the longwall extraction when the abutment stresses have not been developed. Some other approach is required in this case. This could be by hydrofracture.

As the key to increasing the permeability of coal is to de-stress it, then the approach to achieving such de-stressing is to remove material. This might be done by cutting the coal to relieve stress, which could be accomplished in a variety of ways including the use of water jets. However, a simple chain cutter run between gate roads would achieve this function with minimal equipment cost. What is envisaged is a toothed chain pulled around drive wheels located in the gate roads with a leading and following chain. This would cut a slot of about 150 mm width parallel with the seam, thus de-stressing it. The gas quantity produced is likely to be significant and some means to collect this gas would be needed. This could be by pre-drilled holes, if these would remain open, or by a shroud arrangement drawing gas from the slot. The slot could be expected to close to some degree a few metres behind the passage of the chain unless it was filled with a propant. The question as to whether the chain would become trapped
by the slot closure needs to be considered in some detail, taking into account the force that could be exerted on the chain. The pumping of proppant filled slurry behind the chain may be able to be used as a form of permeable stowing that would permit drainage.

**Drainage of the relaxed zone**

The production of gas from coal and rocks in the relaxed zone or goaf may be the single most important factor in determining whether production can take place at economic rates. While vertical goaf drainage holes may serve to drain the goaf in a single seam shallow mining operation, they are unlikely to be suitable for most multilevel seam mining. The reasons for this are the difficulty in maintaining a hole through multiple goaf formations and the extreme difficulty in drilling through old goaf areas.

Drilling holes into the roof or floor strata from longwall gate roads is a difficult and time consuming process. These holes often have to be drilled at close spacings to have adequate effectiveness as they frequently become blocked by rock movement adjacent to the gate road. In some cases, casing has to be installed for approximately the first 60 metres to avoid the hole closing. In addition, the volume of gas that is produced requires a significant number of holes or large hole diameters. Having to repeat the drilling process for each longwall block is a time consuming and expensive process.

One alternative is to create long boreholes in rock between seams. These can be drilled either along the longwall block so that the rear portion of the hole is being absorbed into the goaf, or they can be drilled across multiple longwall blocks in which case entire hole ends would disappear into the goaf as the face advances. In the latter case, the holes would need to be drilled at a reasonably close spacing to pick up the gas being emitted. The drilling of holes of 3 km length in rock requires significant underground drilling capacity, but is technically feasible.

The volume of gas that may be expected in some situations could require a hole of 0.5 m diameter to ensure that a vacuum is maintained to draw gas from the end of the hole. In some countries, such holes would be mined as drainage adits, a process that permits full support to be used. In the Australian context, the cost of this may be high and consideration should be given to the use of remotely controlled tunnel boring machines for this purpose. The prime challenge that these would face is one of ventilation, or more promisingly, intertwinisation of the hole, and in making the equipment flameproof. As holes become larger the problems of support become more complex with joint related failure becoming increasingly important.

**CONCLUSIONS**

While some gassy coal seams will continue to be able to be drained for mining using various permutations of current practices of surface to in-seam or underground drilling, others will not. This particularly applies to the deeper, more highly stressed and impermeable coals. Under these situations, drilling in-seam can be extremely difficult due to hole failure which becomes more of a problem as the fluid pressure is reduced. In addition, the permeability may drop to a few microdarcies. In these conditions in-seam drilling is not an option.

Under these circumstances drilling needs to be conducted in more competent strata adjacent to the seam. These holes can then be cased, cemented, perforated and hydrofractured through to the seam so as to initiate drainage over a far greater area than that available to a borehole. This can be conducted through surface to in-seam holes or from underground. In the latter case, a system for the reticulation of hydrofracture fluid through the mine development would be useful. The hydrofracture spacing may be reduced to several metres to cope with low permeability coals. Whether the coal drains after fracturing will depend to some degree on the stress changes that the coal undergoes during drainage. These hydrofracture techniques are likely to be suitable to lower gas contents so that mine roadway development may take place. In this case, the hydrofracture borehole is probably best situated in the floor beneath the roadway or gate roads so as to avoid damage to the roof.

The main block of coal will also need to be drained before production takes place. This could be accomplished by the same technique as for development but is probably better handled by using the traditional approach of using abutment loading to break and de-stress the coal and thus promote drainage. An alternative approach of slot cutting the longwall block to de-stress it and promote drainage may be an option.
The use of mining to relax and de-stress adjacent seams so as to promote gas flow and permit the subsequent mining of that seam is considered to be a very important technology that will need to be utilised in Australia. Indeed, in some cases, its use is the only way in which mining may economically take place. In Europe, Russia and China the mining of an initial sacrificial seam at sub economic rates is often the only way to improve the permeability of other seams so that they may be drained and mined without risk of outbursts and gas outs of the face.

The traditional practices, of cross hole drilling from underground to collect the gas from the relaxed zone are however thought to be inefficient. Rather it is suggested that alternative technologies that permit the drilling of long, large diameter holes along longwall blocks or across multiple blocks could be beneficially used. Technology to drill such holes is available or may be readily developed.

The overall concept is presented in Figure 1 where drilling is accomplished from drainage adits driven in rock. Such adits could be fitted with pipe work for gas removal, or the entire adit be used for gas removal.

The total drainage process lends itself to an approach where a large block containing several seams and a number of longwall blocks are prepared for mining by drilling for pre and post drainage. In this respect the concept of the process is one of pre-conditioning prior to mining such as might be adopted in a block caving mine. This keeps production and gas drainage separate.

Finally Australian practice in determining outburst risk needs some fairly drastic overhaul to accommodate all of the real factors that influence outbursts. A potential energy based approach is advocated.

Aspects of the technologies described in this paper are patented or are the subject of patent applications.

REFERENCES


DEVELOPMENT AND MINING OF HIGH GAS LOW PERMEABILITY COAL SEAMS IN THE KARAGANDA COAL FIELD-KAZAKHSTAN

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ABSTRACT: The seams being mined in the Karaganda coalfield in Kazakhstan are very gassy and have low permeability, so called tight coals. In addition, the coalfield is dry, which means that “traditional” predrainage, through a two phase gas liberation process, by first dewatering, is not possible. Outbursts in development particularly in the D₆ seam are still a major concern and technologies to allow better prediction of geological anomalies are being tried. Managing gas levels during the longwall extraction calls for special applications to be adopted. These have resulted in improved volumetric underground gas capture and purity, which allows the resultant drained gas to be utilised for local power generation. The methods developed and those foreseen in the future to achieve more effective predrainage and more effective operational gas management are described.

INTRODUCTION

The Karaganda coalfield, in Kazakhstan, was first developed during the early part of last century, with major mine exploration and new mine developments during the post-war period. Mining currently takes place from some 15 mines, eight of which are underground and belong to “ArcelorMittal”, Coal Division.

Most operating underground mines date from the second half of last century and these are more or less of standard Soviet design, with a nominal annual shaft hoisting capacity of 1.5-2 Mt (ROM) and using full caving longwall extraction.

Over the last five years the accident rate within mines has improved significantly, through a focussed programme of investment, modernisation and operational mind set changes. Many technical challenges still remain to be fully mastered, high gas and low permeability coal being one of them.

GEOLOGICAL SETTING

The Karaganda basin is located in the area of the same name near central Kazakhstan. Within the Karaganda region, the coal bearing areas occupy some 2000 km² at a total thickness of 4000 m. The basin is characterised by three main synclinal structures. The productive coal mines go down to depths 700-1200 m, the working series contain up to 30 coal seams, varying from <1 m to 7 m in thickness, consisting of high quality energy coal and prime metallurgical coal. Total coal resources of the Karaganda basin up to the depth of 1800 m have been estimated at 41.3 bt with a total coal thickness of over 40 m.

The main target seams for extraction are the K and D seams. Both seams are outburst prone at depth and liable to spontaneous combustion. Dips in the coalfield vary between 8 to 35 plus degrees. Near verticality is observed at the sub-crops. Minor faulting and associated dip changes are not infrequent.

The D₆ seam is a thick high quality coking coal, with a very distinct shear zone (0.2-1.2 m) in the bottom section. The D₆ seam has been found to be extremely outburst prone, particularly the bottom section. The permeability of the seam is extremely low, and wet drilling through the bottom section zone has to date been found to be most challenging, probably because of the fine coal within the shear zone and swelling clays within the seam itself. The diffusion coefficient of the bottom section has also been found to be several orders of magnitude greater than the top section of the seam.

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GAS ENVIRONMENT

The Karaganda basin is considered a high gas content resource area. The gas content intensity of the seams increases from the beginning of the methane zone at 400-500 m to gas levels of 15-20 m$^3$/t at current working depths and stabilising at 800-1200 m depth to limits of 22-27 m$^3$/t. The depth of the methane free zone from the surface varies over the range of 60-250 m and depends on local geologic and structural features.

Operational challenges

As the underground workings get deeper the gas content within the main seams increases and the permeability of the seams decreases, as illustrated in Table 1:

| Table 1 - Variations in depth and permeability (Baimukhametov, 2006; Sigra, 2009) |
|---|---|---|
| Seam series | Depth / m | Permeability /$10^{-2}$ mD |
| $K_{10^{-2}}K_{12}$ | 400 | 1.51-2.77 |
|  | 600 | 0.19-0.35 |
|  | 800 | 0.05-0.09 |
| $D_1-D_6$ | 400 | 5.85-3.89 |
|  | 600 | 0.75-0.50 |
|  | 800 | 0.19-0.13 |
| $D_6$ (measured 2009) | 600 | 0.3 |

In response to containing mining costs, and benefitting from the higher production rates of modern mining equipment, increased productivity is being sought.

All three factors potentially lead to a contradictory mix.

- Increased gas contents require more effective pre drainage, or more time to drain;
- Lower permeability relies on more effective permeability enhancement, and/or requires more drainage time;
- Higher productivity demands for increased development and extraction rates, which is only possible in "safely" drained areas of extraction.

GAS

The efficiency of coal seam degassing has relied on increasing the gas permeability of the coal bed by creation of induced fractures and by allowing for an increase of the degassing period.

In order to achieve an acceptable gas content reduction the following actions have been taken:

- Permeability enhancement has been adopted from vertical surface predrainage wells which have been subjected to in-seam water fracking, with possible reworking at a later stage;
- In the underground situation, undermining (or overmining) the target seam, thereby allowing it to be extracted in the “de-stressed” zone. So called “protection” extraction has proven quite successful. Increasingly the potential “protection seam” is outside the distress zone, or not viable (too thin) in a mining sense, so alternatives have to be developed;
- Pre mining longwall block drainage is extensively used, with mixed results, mainly because of low in seam permeability, and the lack of natural permeability enhancement as part of the activity. Where the coals are water saturated, pumping leads to relaxation, which allows gas flow to follow the drainage through pumping. This two phase mechanism is not available in a dry coalfield. Artificial permeability enhancement, has to be engineered, such as hydrofracking;
- Drainage within the approaching face front abutment (60-80 m), is a technique under consideration, as the permeability’s of <0.01 increases to over 10 mD. The gas evacuation time is very limited because of the advancing face and the capacity control of the (vacuum) drainage system will need to be re-dimensioned;
• Post mining and goaf drainage is quite advanced in application. This has been described in some detail by Mukhamedzhanov et al. (2009). These techniques range from surface drainage boreholes, above seam drainage sewers and underground goaf drainage holes on vacuum and are well developed and applied where relevant. Selective directional drilling and eventual “reservoir” modelling will add to the available tools and capabilities in this sphere.

Typical degassing performance efficiencies are given in the following Table 2

<table>
<thead>
<tr>
<th>Degassing method</th>
<th>Typical capture performance%/</th>
<th>Efficiency%/</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pre drainage</td>
<td>3.75</td>
<td>12</td>
</tr>
<tr>
<td>Development face drainage</td>
<td>0.75</td>
<td>10</td>
</tr>
<tr>
<td>Pre mining block drainage</td>
<td>2.2</td>
<td>33</td>
</tr>
<tr>
<td>Goaf/LW block drainage</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Vertical wells from surface</td>
<td>35.4</td>
<td>38.2</td>
</tr>
<tr>
<td>Drilled goaf wells</td>
<td>9.5</td>
<td>9.7</td>
</tr>
<tr>
<td>Goaf drainage</td>
<td>14.0</td>
<td>14.4</td>
</tr>
<tr>
<td>Face ventilation</td>
<td>34.4</td>
<td>37.7</td>
</tr>
</tbody>
</table>

Due to gas management and operational safety concerns in the past, current production rates from “gas rich” longwalls have been restricted to 4000 t/day, which is well below the face equipment capability.

A limiting factor is also the face ventilation, where the air velocity along the face is restricted to <4 m/s. By applying Y ventilation, the face velocity may be reduced, by increasing the quantity of air over the tailgate. By reducing the seam gas content before longwall mining to a value of <4 m³/t, the face ventilation constraint and spontaneous combustion exposure become more manageable.

OUTBURST MANAGEMENT

Unlike Australia, no universally compatible threshold value (the “8/12” rule) is currently applied in Kazakhstan underground coal mining. Current practice is based on the Russian standards. Whilst these are robust in their nature, there is a case to be made to reconsider the existing “outburst probability” determination, and consider the redefinition of a scientifically sustainable Norm. The Norm must find its origin in the characteristics and behaviour of the seam conditions of the Karaganda coals. Typical development rates are 25-40 m/month, in the high gas, outburst prone seams such as the D6. In addition developments roadways are driven in stone, 10m below the future gate roads, to allow seam drainage to be carried out. In an economic future such development rates and stone drive efficiencies, need to be improved significantly to remain financially viable and allow for social and economic advancement of the operations.

GAS MANAGEMENT FOR EXTRACTION

Issues need to be addressed include the following:

Pre drainage from surface

During the early ‘60’s hydro and nitrogen fracking, from vertical boreholes, was tested in the K12 seam, with good results. Further tests proved successful for seams down to 500 m, with a reported gas reduction of 3.3-3.8 m³/t and when including the inseam (vacuum) extraction an overall reduction of 8-9 m³/t has been reported. In addition the outburst hazard was found to be below the critical limit.

Tests carried out at deeper horizons, 700-800 m, proved very disappointing. It was concluded that below 500-550 m, the deterioration of well productivity reduced significantly (e.g. 1.1 m³/min at 400 m, 0.5 m³/min at 550 m and <0.2 m³/min at 750 m depth) because of failure to keep the fractures open at depth.

In early ‘80’s an area of the D6 seam at Lenina mine was drilled and fracked from the surface, with water and other chemical active agents. By pre-draining over a period of ten years, it was possible to reduce the gas levels by 6-9 m³/t. An interesting observation was made at the time, in that the fine coal from
the shear zone, caused a fine coal plug to be formed, which effectively sealed the well, necessitating re-cleaning. Similar plug sealing has been experienced with wet underground drilling trials from the stone heading 10 m under the seam. It is postulated that clays within the seam react with the drilling fluid (water) and the super fine coal from the shear zone to form a plug. Using (vegetable) oil or gas/air as the fracking medium should overcome the swelling clay problem. Figure 1 shows a coal plug being cleaned out.

Figure 1 - Coal fines plug being cleared from hole, shown as a black jet flowing from the borehole as it is opened up after hydrofrac-pumping

Based on the positive results during the ‘80’s, pre-drainage from vertical boreholes, spaced at 250 m is standard practice for the D6 and other high gas seams. The gas from the seam is flared off on surface, where there is no natural gas flow; this is assisted by pumping out the frac-water with a donkey pump. Figure 2 shows a surface donkey pump installation with flare. Because there is no real water make, within the seam, there is no need to provide drainage infrastructure.

Figure 2 - Nodding donkey pump, with gas flare, with mine Kazakhstanskaya boundary shaft in background

More recently in seam gas make tests have been carried out at Kazakhstanskaya mine, also in the D6 seam, as the pre-drain area is being developed for mining. Of the wells shown, most were re-stimulated after three to four years of operation, to maximise the degassing. The effectiveness of the pre-drainage is shown in Table 3.
Table 3- Effectiveness of pre-drainage

<table>
<thead>
<tr>
<th>BH number</th>
<th>Drainage time /months</th>
<th>Gas extracted to date/Mm³</th>
<th>Seam gas content reduction/(m³/t)</th>
<th>Well characteristic</th>
</tr>
</thead>
<tbody>
<tr>
<td>23</td>
<td>126</td>
<td>0.93</td>
<td>4.02</td>
<td>Self flow</td>
</tr>
<tr>
<td>24</td>
<td>126</td>
<td>1.27</td>
<td>6.32</td>
<td>Self flow</td>
</tr>
<tr>
<td>25</td>
<td>120</td>
<td>1.09</td>
<td>5.44</td>
<td>Donkey pump</td>
</tr>
<tr>
<td>30</td>
<td>106</td>
<td>0.56</td>
<td>2.80</td>
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</tr>
<tr>
<td>31</td>
<td>103</td>
<td>0.60</td>
<td>5.46</td>
<td>Self flow</td>
</tr>
<tr>
<td>37</td>
<td>80</td>
<td>0.80</td>
<td>4.01</td>
<td>Donkey pump</td>
</tr>
</tbody>
</table>

The longwall blocks of such pre-drained areas are currently being developed. Because of the high outburst risk in development of the D₆ seam, the initial development takes place in stone, 10 m below the seam. During the development of these stone drives, the gas make of the overlying seam has been measured. The gas makes have been recorded in the zones as indicated in T, and in the section of the mine plan shown in Figure 3.

Table 4- Gas make data

<table>
<thead>
<tr>
<th>Zone</th>
<th>Average gas make (m³/min) for zone</th>
<th>Max gas make</th>
<th>Min gas make</th>
</tr>
</thead>
<tbody>
<tr>
<td>A</td>
<td>0.02</td>
<td>0.05</td>
<td>0</td>
</tr>
<tr>
<td>B</td>
<td>0.005</td>
<td>0.03</td>
<td>0</td>
</tr>
<tr>
<td>C</td>
<td>0.04</td>
<td>0.09</td>
<td>0.01</td>
</tr>
<tr>
<td>D</td>
<td>0.07</td>
<td>0.16</td>
<td>0.02</td>
</tr>
</tbody>
</table>

Figure 3 - Mine Kazakhstanskaya pre drainage holes (23, 24 and 37), and zones of measured UG gas production (A, B, C and D)

Pre drainage from underground

It is observed that for an initial gas content of say 20 m³/t, a reduction of up to 6 m³/t from the surface pre-drainage still results in a challenging underground gas environment. In-seam techniques and permeability enhancements are required to achieve desired operationally safe gas content levels.

Two aspects of pre-drainage from underground are important. The first relates to achieving safe background gas levels, to avoid outbursts during development. The second relates to avoiding being gassed out at the longwall face during production.
Drainage during development

The gas testing and pre-drainage within in-seam development headings is well regulated by the Kazakhstan Authorities. This essentially consists of drilling 17 m ahead of the face, in an overlap fashion. Outburst proneness detection is then undertaken and if it was considered safe, mining may proceed to within 5 m of the end of the holes. Unfortunately still too many near misses and outburst accidents occur. All outbursts appear to be associated with geological features.

Plans are in hand to trial underground remote sensing techniques to better pinpoint geological disturbances, abnormal gas accumulations or high diffusion rated zones. Trials with surface techniques have also been carried out.

In the most highly outburst prone areas, D6 in particular, the future roadway development locations are degassed by driving stone roadways 10 m under the seam, and drilling degassing holes, up into the seam as shown in Figure 4. Typically a fan of five holes is drilled into the seam from the roof of the roadway below at 4 m spacing, which are put on vacuum, up into the seam. Once a safe threshold (time or gas content related) has been achieved, within the seam, in-seam gate road development will take place.

![Figure 4 - 3D visualisation of in stone development (red) and degassing holes for in seam gate roads (brown)](image)

Drainage of longwall block

Drainage of the longwall block is accomplished by drilling a series of long parallel in-seam holes across the block. The spacing of these holes is about 4 m, but trials of 2 m spacing have been carried out. The existing drilling equipment is only capable of efficiently drilling 100-120 m auger air-flush holes within the seam. Hole wander has also been found to be a major constraint when advocating longer in-seam holes. The unsatisfactory trials of wet drilling have already been mentioned.

Little difference in drainage efficiency has been noted, when drilling parallel or at an angle towards/away from the face line. The beneficial effect of cleat direction does not appear to have been considered, or found to play a part.

Most benefit has been achieved by better sealing of the borehole collars and being able to regulate the vacuum attached to the hole. This has resulted in a doubling of the gas purity and subsequent greater effective extraction of methane gas as shown in Figure 5.

Trials with directional longhole drilling are planned and in-hole survey tool applications are foreseen. The availability of a simple IS survey tool for hole depths of up to 100 m has proven to be challenging.
Destressing

The most effective method of increasing the seam permeability is by undermining, (within 50-70 m) or to a lesser extent over mining, (within 30 m) of the target seam. When undermining, no drainage of the target seam is required, which is consistent with experience in Germany, Russia and elsewhere. Such an approach is common when mining the K_{12} seam, by taking the underlying K_{10} first. The added advantage in this instance is that the mining conditions for the K_{10} are also better, as the seam has lower gas and outburst characteristics.

The zone within the front abutment of the longwall is marked by having a high permeability. This potentially allows increased gas extraction from this zone by the in seam drainage holes. Careful vacuum management is required, as there will be the changes in the drainage dimensions. However there is a future opportunity for interactive purity and suction instrumentation and automation.

Permeability enhancement

Due to the absence of water within the seam, gas liberation is essentially a single phase process. This process is probably a balancing act between the seam pressure, the sorption pressure and the resultant effective pressure envelope with induced coal shrinkage.

Limited shrinkage tests have been carried out on the D_6 seam, and although found to be present, the results were inconclusive, possibly because of inadequate sample quality.

In practice, over time, may be years, sufficient gas liberation may come about. This clearly is not an option in a mining environment. Some method of managing the time-frame or artificially enhancing the permeability is sought. A mechanism of “kicking off” a self-sustaining gas release is required, such as by creating an above critical enhanced permeability volume, or stress condition, which induces permeability increase through shrinkage by way of releasing the initial gas volume and pressure. This needs to be the focus into the future.

CONCLUSIONS

Developments within the Karaganda coalfield have moved forward and there is a clear focus on the work that still needs to be done. It is hoped that this work will lead to mastering the safe mining of high gas and low permeability coal seams in an efficient and safe manner.
It is interesting to note that increasingly Australian mines are planning to work low permeability seams, or “difficult to drain” areas. There is a common desire to cracking the problems and achieving a safer working environment underground.

REFERENCES

Baimukhametov, S, 2006. Issues of Safe Coal Extraction from Coal Beds with High Gas Content, monograph, Karaganda, Kazakhstan.


DIRECTIONAL CONTROL IN LONGHOLE DRILLING

Frank Hungerford¹,², Ting Ren¹ and Naj Aziz¹

ABSTRACT: Directional drilling has provided the coal mining industry with a means to position in-seam boreholes to achieve goals such as gas drainage, exploration and inrush protection. The evolution of the technology has progressed to a point where most drilling operators use the standard down-hole drilling configuration which has proven effective in most applications. Survey techniques have been modified to reduce the distance between bit and survey point without consideration of the effects on survey accuracy, driller skills and data collection. Data from two long in-seam boreholes is analysed to show the response curves of the standard down-hole configuration. An indication of the effects of in-hole friction, of feed pressures and limitations put on the eventual borehole depths is shown.

INTRODUCTION

Directional drilling has given the mining industry the ability to place boreholes in designed locations to achieve specific goals such as gas drainage, exploration, barrier proving and water drainage. This has been possible through the use of Down Hole Motor (DHM) drilling which provides the ability to off-set the direction of drilling and surveying to accurately locate the borehole and orientate the DHM for steering. The off-set provided by the configuration of DHM bend and bit diameter has to be matched with the drilling environment to provide the ability to control the borehole trajectory and azimuth. Because of the variety of drilling environments likely to be experienced when drilling any long hole within a coal seam, this configuration is usually set to manage the most adverse environment.

Surveying and drilling practices have evolved to suit the requirements at each mine. The ability to drill long holes becomes an exercise in directional control with drilling practices to limit the in-hole friction which increases as a borehole increases in depth. Reviewing these practices and resultant frictional effects is intended to refine driller skills and practices to improve borehole drilling efficiency and depth capacity.

DIRECTIONAL CONTROL

In-seam drilling with DHMs has evolved into an industry standard over the past 25 years. Driving high pressure water through the DHM provides the rotation and torque while the off-set of the bend/bit Outer Diameter (OD) configuration provides the ability to steer a borehole.

The off-set of the bit at the front of a DHM is usually provided by a bend (or bent sub/housing) installed between the power section and the bearing pack. The off-set of the bend can be enhanced by the addition of an off-set (or kick) pad attached to the bend or the addition of a tungsten carbide wear resistant coating. In any case of adding to the thickness of the 73 mm OD DHM, the final thickness through that point should not exceed the Internal Diameter (ID) of HQ rods which may be required to over-core should the system become bogged. The use of an extension between the DHM and bit also provides additional off-set.

Directional control (in the first instance, vertical control) is only deemed to be provided if the DHM has the ability to climb when orientated through a range of upward facing positions. To do this, the off-set of the DHM must exceed the oversize that exists between the bit diameter and the OD of the DHM. To provide lateral control, the off-set must be of sufficient magnitude to provide a range of orientations either side of vertical that allows lateral deviations while still having the ability to climb.

The configuration, in the mid 1980’s, with the 74 mm Slimdril, 1-2 lobe DHM was to use a ¾ ° bend with the standard 89 mm PCD bit that was available (Hungerford, et al., 1988). Using the Surtron electronic survey instrument available at the time to survey at 6 m intervals with the surveying location positioned 6 m behind the bit, each survey point matched the location at which the orientation of the DHM (also

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referred to as Tool Face) was changed. The change between surveys represented the change (or response) in both the vertical and horizontal planes created by the previous DHM orientation. Over the course of drilling a long hole, combining these changes for the various orientations, response curves are established in both vertical and lateral planes. The first such response curves were compiled from the drilling of the first 1000 m in-seam borehole at Appin Colliery (Hungerford, et al., 1988) (Figures 1, 2). From this and subsequent drilling, a drillers guide was established as shown in Figure 3 and is included in the Appin Drilling Manual (BHP, 1996).

All the initial DHM drilling was done in slide motion. Attempts to use a combination of rotation with slide (Rotary/slide drilling commonly employed by surface drilling operations) led to the borehole dropping rapidly into the floor of the seam floor from which it could not be recovered in slide motion with the limited climbing ability available with the DHM configuration.

DEPTH CAPACITY

Using the early configuration with the 89 mm bit, the feed rate from the start of a borehole was steady out to the 60 m depth. From that point, it was noticed that the feed started to surge with the surge of water pressure to match that of the feed. The surging increased to the point that the spikes in the water pressure exceeded the “relief” pressure setting of the water pump relief valve and the DHM would stall. To prevent stalling, the feed rate was reduced to limit the surging and thus the magnitude of the water pressure spikes. As the borehole was extended in depth, the surging progressively increased, resulting in progressively slower feed rates being used to prevent stalling. Very slow feed rates were
being used to achieve the first 1000 m borehole and subsequent cross-measure/in-seam drilling in the Balgownie Seam only achieved 923 m and 870 m respectively (Hungerford, et al., 1988). In each case, surging feed and slow penetration rates caused the termination of drilling while the maximum available feed pressure had not been reached. This indicated that borehole depths were limited by the surging feed in the borehole. Drill rigs that operated under a constant feed rate using a hold-back valve on the return line did not suffer as badly, with surging, as systems with constant pressure feed and no restriction on the return line.

This surging characteristic contributed to the effect of the axial load required to overcome the frictional effects in the hole causing the rods to flex towards the outside of the curves in the borehole. As the axial load increases, the rods are pushed out against the outside of each curve until the friction is overcome. As friction is overcome to allow the rods to slide forward, they return to the centre position at the bottom of the borehole resulting in the bit surging forward at the face. This surge forward of the bit into the face rapidly increases the torque loading on the bit and DHM, resulting in spikes in the water pressure.

In an attempt to reduce in-hole friction, an increase in bit diameter to 95 mm was proposed. When the bit manufacturers were approached to produce a 95 mm PCD bit, they indicated that the standard diameter of the PCD bit used for open-hole drilling prior to HQ coring was 96.1 mm. This bit was sourced and combined with the newly available 73 mm OD, high-torque, low speed Accu-dril DHM (4-5 lobe). To increase the off-set of the DHM to provide vertical control, the magnitude of the bend was increased to 1.25°. This configuration provided a DHM offset of 17.5 mm which, when combined with the 11.5 mm oversize of the 96 mm bit, provided 6 mm lift to the underside of the bit when the bend of the DHM was in the 12 o’clock position. This configuration was first used on an exploration project at North Cliff Colliery with improved penetration rates, surging noticeably reduced and all boreholes reaching design depths out to 900 m (Walsh and hungerford, 1993). Two longholes (not restricted by structures) were then drilled to the north. In the first borehole of these boreholes, drilling achieved a depth of 1 250 m before closer attention to bends in the borehole led to a depth of 1 533 m being achieved with the second borehole. These depths were achieved with Eastman single-shot wire-line surveying at 18 m intervals with the drillers having to rely on the response curves modified from those of the previous configuration (Hungerford, 1995) to plan their DHM orientations between surveys.

Figure 4 - Vertical response guide - 1.25° bend, Accu-dril DHM

Figure 5 - Lateral response guide - 1.25° bend, Accu-dril DHM

STEERING LIMITATIONS

Access to electronic survey systems was limited to developmental, ineffective or unreliable systems.

The majority of surveying was with the Eastman, single-shot, wire-line survey tool which suffered increased delays with depth and reliability problems with the operation of the tools and wire-line. Because of these characteristics, survey intervals were increased to 18 m involving three orientation changes with the drillers having to design their drilling between the survey points. DHM responses were based on the original responses established with the Slimdril (¾° bend) and 89 mm bit.
In this environment, drillers of that era developed a high knowledge of DHM performance, orientation responses and rig performance to progressively plan and nurture drilling in a longhole rarely matched by the current drillers of using the latest survey technology. The development of the Directional Drill Monitor using Modular Electrical Connected Cable Assembly (DDM-MECCA) gave the industry a survey system that allowed reliable surveys at regular 6 m intervals with no time delays out to depths beyond 1500 m. The DHMs used at that time were the regular 73 mm Accu-drill DHM and the 2-3/8” Drilex DHM with a 3/4° bend and 10 mm kick pad. With the Drilex being a steel motor, surveying was located 3 m behind the DHM (6 m behind the bit) to maintain a non-magnetic survey environment. Drillers were able to develop an understanding of both vertical and lateral responses in a particular drilling environment and anticipate the survey result at the bit before planning the next 6 m of drilling.

**SURVEY POSITION**

With the eventual discontinued use of the steel Drilex DHM, all drilling reverted to using the non-magnetic Accudril DHM. With that, drilling departments were able to move the survey point up to the back of the DHM (3 m behind the bit) and still apparently be in a non-magnetic survey environment. The reasoning behind this move was supposedly to provide better directional control (being only 3 m behind a potential stuff-up) which did not require as high a level of driller skills so it would be easier to train new drillers to a level necessary to directional drill.

![Figure 6 - Survey positions with 6 m orientation changes](image)

Although drilling departments have managed operating with this system without adverse effects, several limitations have been identified:

- All survey calculations are based on a consistent curve between survey points (Figure 3). This occurs when a consistent orientation has been used between survey points as is the case with surveys 6 m behind the bit. With surveying 3 m behind the bit and using 6 m orientations intervals, the survey point is located at the orientation transition point. The change in survey results represents 3 m of the previous orientation and 3 m of the current orientation. The resultant survey calculations produce a nominal location of the borehole which relies on a random average of which the accuracy is not known. An interval of “flip-flop” drilling each side would be represented by relatively consistent surveys which don’t show what could be 2.5°-3.0° changes if surveyed at the transition points;

- In any drilling environment, response curves can’t be established to provide drillers and designers with knowledge to assist steering of the drilling. Limited knowledge can be gleaned from sections of holes where the same orientation has been used on consecutive survey intervals, as in drilling around an extended curve;

- No accurate assessment of the magnitude and location of significant bends in a hole can be made to assess the potential problems with any subsequent over-coring operations should the rods become bogged;

- In cases of unusual thrust pressures required during the drilling of a borehole, an assessment of the frictional effects due to the accumulated effects of bends in the hole can’t be determined;
The overall effect is to limit the skills developed by directional drillers.

This knowledge would assist in the assessment of the equipment, the drilling parameters and practices required to improve drilling performance and depths achieved. It would also assist in driller training.

The increased use of steel DHMs for both stone and coal drilling requires the survey point to be moved back 3 m behind the DHM (6 m behind the bit). Drillers with limited experience of directional drilling with this configuration (surveying 6 m behind the bit) would benefit from access to support information such as response curves.

Several cases exist of boreholes being found out of position. With calibration results being verified, the unusual survey results have put some doubt of the non-magnetic environment expected to exist directly behind an apparent non-magnetic DHM. Some trial calibration procedures involving the introduction of a DHM to the calibration environment has shown similar discrepancies. The inconsistent nature of these results would indicate that alternative practices would need to be put in place for accuracy sensitive projects.

**Borehole performance - Case 1**

Comprehensive data collection was undertaken during the drilling of a long in-seam exploration borehole in the Bulli Seam on the South Coast. As drilling advanced, the seam profile was progressively defined with roof intersections (Figure 7). The grade of the seam along the line of the borehole was relatively consistent out to 900 m where it flattened through an area of suspected faulting before rolling over to continue on a similar down grade.

The borehole was maintained along a Target Azimuth line with only two deviations more than 1 m from this line (Figure 8). The left/right and up/down deviations created by the changes in DHM orientation (Tool Face: expressed as clock face hours) every 6 m are not evident on these plots.

In the vertical plane, the change in borehole pitch over 6 m achieved by each Tool-face setting used has been plotted to create a Vertical Response Curve (Figure 9). The plot indicates a range of Tool Face settings between 10:10 and 3:30 which achieve a positive (climb) response with a maximum of 2.4°/6 m achieved with a Tool Face of 12:40. Negative (dropping) responses are achieved between 3:30 and 10:10 with a maximum drop of 3.0°/6 m at 7:00. Also evident is the variation in responses achieved by the various Tool Face settings. This is attributed to the non-homogenous coal strata environment being drilled and the various pitches relative to the bedding planes.

In the horizontal plane, a similar profile curve has been created as a vertical response curve (Figure 10). Positive lateral curve (right) was achieved between 12:50 and 7:00 with a maximum of 2.1°/6 m at 4:00. Negative curve (left) was achieved between 7:00 and 12:50 with a maximum curve left of 2.4°/6 m achieved at 10:10. The stronger curves to the left could be attributed to the effects of cleat. As expected, the Tool Face settings which achieved no lateral deviation (12:50 and 7:00) were those which achieved maximum climb and drop in the vertical plane.
The progress of the borehole was expected to be restricted by the increased surging feed, leading to progressively slower penetration rates. In this case, the penetration rate started at a comfortable 1.2 m/min and slowed progressively out to 600 m, from which a rate of 0.80 - 0.85 m/min was maintained (Figure 11). Although the depth of the hole was limited by the number of drill rods available on site, the drillers felt that the maximum feed capacity of the rig was being reached beyond 1 200 m.

The plot of feed pressure versus borehole depth (Figure 12) indicates that the increase in borehole friction is not linear. The plot shows that the final depth of this borehole would have been limited eventually to 1 300 m if drilling had continued until the max available pressure had been reached. There are three trends in the plot: 0 - 400 m, 400 - 1 150 m and 1 150+ m. The increasing trends beyond 400 m indicate that the increases in axial load in the rods must have an increasing effect on friction as borehole depth increases. The extent of the relationship between borehole depth, axial load, general curves and short 6 m orientation change curves is not known.

Borehole performance - Case 2

A second long exploration borehole was attempted in different conditions in a coal seam in the Hunter Valley using the same drill rig and equipment. This borehole was intended to cross the adjacent longwall block and run along the gate-road pillars with roof cores being taken at predetermined intervals (Figure 13). Normal exploration practice was employed of progressively defining the profile of seam (Figure 14). The Case 2 borehole starts from the south-east corner of Figure 15 and curves around towards the north for two legs to extend along a Target Azimuth of 348.0°. During drilling around the main lateral curve, over the interval 150-250 m, the borehole had a tangential intersection with a fault identified previously by an in-seam borehole. Drilling crossed the original exploration borehole in two places and was extended to 795 m, before water was lost into the original borehole preventing further drilling from that point. A second leg was designed to branch from 342 m, continue along the azimuth of the borehole at that point out to the left then curve around to continue along the target azimuth a further 25 m to the left (Figure 16).
At the point of losing water into the previous borehole, drilling had slowed with regular stalling of the DHM due to what the drillers thought was very hard drilling. The second leg was extended to 867 m with similar conditions being experienced beyond 831 m depth. The plot of Feed Pressure versus Borehole Depth (Figure 17) on the first leg showed a pattern similar to that of Borehole 816D40 but with a more rapid increase beyond 400 m. A further rate increase in Feed Pressure beyond 760 m, when extrapolated, indicated that the maximum feed pressure capacity of the rig would have limited the final depth to 820 m in this leg.

Conditions different from the 816D40 drilling included:

- an initial Entry Heading 54.4° offset from the Target Azimuth;
- a curve with an average curve rate of 1.0 ° / 6 m from the entry heading to target azimuth;
- the presence of small clay bands within the seam;
- a tangential intersection over 100 m between 150 m and 250 m with a fault zone;
- the presence of a second fault zone at 653 m which was crossed.

When comparing the plots of feed pressure versus borehole depth for both boreholes (Figure 18), the shape of the trends are similar but with the increased rate of change for the MG40-30-1 borehole beyond 400 m. The initial increase in the rate of change of each occurred from the feed pressure of 4.1 MPa while the final rapid increase in rate for both occurred from a feed pressure of 12 MPa. In either case, drilling out to depths of 700 m, did not present difficulties with feed.
The appearance of suspected hard strata and regular stalling of the DHM was similar to previous experiences with the smaller bit diameter. In both cases, the ability to drill further was restricted by both surging in the borehole due to friction and the maximum feed capacity of the rig being reached.

Figure 17 - Feed pressure (MG40-30-1)

Figure 18 - Feed pressure (816D40 & MG40-30-1)

FUTURE RESEARCH

The aim of future research on the subjects covered in this paper is introduce drilling and surveying practices in Vally Longwall International (VLI) to allow the collect more suitable data to establish “standard” response curves for each drilling environment and the variety of DHMs and configurations being used.

Emphasis will be towards data collection and analysis to determine:

- the effects of overall curves in boreholes and their rates of curve on in-hole friction effects and borehole capacity;
- the accumulated effects of the smaller bends in the hole created by orientation changes, the number of bends and where they are in the hole;
- the effects the initial Entry Heading off-set angle of the standpipe from the Target Azimuth;
- effect of maintaining design curve versus over-shooting or under-cutting the design lateral deviation;
- a comparison of feed-in pressure to pull-out pressure in each longhole.

With this data collection and processing, the VLI drillers will be exposed and trained in improved drilling practices and data collection and explained to them the results of the research. Training programs will be enhanced with this knowledge.

CONCLUSIONS

At this stage, drilling in one borehole may extend to a depth several hundred metres deeper than a previous borehole in the same environment without any apparent changes in drilling parameters. The aim is to develop an understanding on the aspects that affect in-hole friction to improve drilling efficiency and depth capacity.

More suitable data is required to establish the characteristics of each DHM configuration in the various environments experienced with in-seam drilling. Providing a DHM configuration specifically suited to each longhole project will limit in-hole curvature to enhance the borehole depths regularly achieved.

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PERMEABILITY TESTING OF COAL UNDER DIFFERENT TRIAXIAL CONDITIONS

Lei Zhang, Naj Aziz, Ting Ren, Jan Nemcik and Zhongwei Wang

ABSTRACT: Permeability refers to the ability of coal to transmit gas when a pressure or concentration gradient exists across it. The permeability of coal is dependent upon factors that include effective stress, gas pressure, water content, disturbance associated with drilling and matrix swelling/shrinkage due to adsorption/desorption. A programme of laboratory tests were conducted on coal samples from the Bulli seam for evaluating the permeability and drainability of coal. The study was conducted using two different types of permeability apparatus. The methods of permeability testing of coal under different triaxial conditions are discussed. Permeability testing of the Bulli seam coal sample with N₂ is described as an example in this study. Both the tests results and the values of calculated permeability were in agreement.

INTRODUCTION

Permeability is considered by many researchers to have a significant impact on a coal seam’s ability to produce gas (Jones, et al., 1982; Osisanya and Schaffitzel, 1996; Zutshi and Harpalani, 2005 and Lamarre, 2007). Permeability, which is closely related to the coal fabric (i.e. cleat spacing and aperture width), varies significantly as fluid pressure changes during coal seam gas production (Cui and Busten, 2006). Permeability has a strong effect on the gas production profile and gas well performance.

Permeability measurements results, tested in small coal samples in laboratory conditions, have been shown to be different from in situ measured values. Testing at Leichhardt Colliery, Gray (1982) found that, the measured core sample permeability was less than 5 mD, whereas the bulk permeability was found to be in the order of 200 mD, for drainage along the cleat. This clearly indicates that more research is needed to focus on the accuracy of different measuring methods and the relationship between the laboratory permeability results and in situ coal permeability result.

A number of different permeability testing apparatus have been reported. They are basically triaxial cells, which simulate the in situ conditions. Some apparatus consists of a conventional triaxial cell, modified to provide gas inlet and exist ports through the upper and lower platens, Harpalani and Schraufnagel (1990), while others are more elaborate in design, such as those reported by Lingard et al. (1984), Lama (1995), Gillies et al. (1995) and Nakahima et al. (1995). The mode of permeability testing, using these different apparatus however, can vary with respect to the way and role of the confinement pressure application.

Increased difficulty of seam gas drainage occurs in sections of some coal seams such as the Bulli seam, which is due mainly to the changes in the permeability of the coal. Such difficult to drain sections of the coal seam are generally associated with an increased percentage of the carbon dioxide. Often reliance is made on the determination of the sorption isotherms rather than assessing the permeability of the coal for effective management of the seam gas drainage. Reliance on sorption isotherms is understandable as it is much simpler method of estimating the gas content, and often decisions are made for gas drainage based on the gas content of coal. A recent study by Black (2012) examining factors contributing to effective drainage of gas from coal found significant lack of information or insufficient level of data on coal permeability in comparison to other parameters such as gas content estimation and proximate analysis values. Black’s study was based on studies of data collected from more than 10 mines in Australia. Difficulties associated with permeability determination in the laboratory or in the field experimentation, are mainly due to the fact both the laboratory and field tests raise concerns about the test method. The laboratory tests are generally carried out on competent core samples, not truly representative of the real in situ condition, while field tests, though yielding representative results, intrude on a mine’s operation and production.
In order to obtain representative permeability values with respect to effective gas drainage from the difficult to drain zone and permit a better understanding of the potential gas recovery through nitrogen injection and displacement process, a laboratory permeability testing programme was initiated by the gas research group of the University of Wollongong. The programme consisted of duplicate testing of coal using two different permeability testing apparatus. Both tests were carried out under triaxial test conditions. The first permeability testing method used is known as Multi Function Outburst Research Rig (MFORR) which was previously reported by Lama (1995), Aziz and Li (1999) and Farhang (2005). In this test, the sample was enclosed in a triaxial gas chamber. The coal sample was subjected directly to gas as the confining pressure. The pressured gas was made to filter through the coal sample while it is being loaded axially. A centrally drilled hole in the coal sample allowed the gas to flow out of the chamber in a controlled manner. The second permeability test apparatus used in this study, was a high pressure triaxial cell, initially built for determining the relative permeability of coal measure rocks under two-phase flow conditions (Indraratna and Haque, 1999; Jasinge, et al., 2011; Perera, et al., 2011). Both methods of testing and the results obtained are the subject of discussion in this paper.

COAL PERMEABILITY TEST WITH MULTI FUNCTION OUTBURST RESEARCH RIG (MFORR)

Apparatus

The Multi function Outburst Research Rig (MFORR) shown in Figure 1, is used to study the permeability of coal from parallel to stratification. MFORR comprises a number of components which can be utilised for permeability testing with the confining pressure being provided by the applied gas pressure which filters through the coal being tested. As a multifunction apparatus the MFORR has various components:

- The main apparatus support frame;
- A precision drill;
- A high pressure chamber which has a load cell for measuring the load applied to the samples of coal;
- A pressure transducer for measuring the pressure inside the chamber;
- Flow meters for measuring the gas flow rate;
- Two strain gauges for measuring the vertical and horizontal strains of the coal sample;
- A universal socket for loading a sample of coal vertically into the gas pressure chamber;
- A gas chromatograph (GC);
- A data acquisition system.

Figure 1 - Multi Function Outburst Research Rig (MFORR)
The gas pressure chamber containing the coal sample is a hollow rectangular prism of cast iron with removable front and back viewing plates. The dimensions of the box are 110 mm x 110 mm x 140 mm. The viewing windows are made of 20 mm thick glass in a cast iron frame. Housed in the chamber is a 1210-BF interfaced load cell with a capacity of 40 kN for monitoring the load applied.

Coal sample preparation

Prior to coring, the lump coal sample from the typical Bulli seam was cast in concrete to form a uniform block for easy coring. A set of standard core samples with a dimension of 54 mm in diameter and 50 mm in height were bored out of the core block. A 2 mm diameter hole was drilled in the middle of the cored coal sample to measure the permeability of this apparatus. Prior to testing, both ends of the prepared specimen were sealed with an adhesive 1 mm thick rubber layer to ensure effective gas flow along radius in the coal. Figure 2 shows the snapshot of the sample.

![Figure 2 - Coal samples for permeability test with MFORR](image)

Testing procedure

The procedure adopted for permeability test consisted of each sample being first mounted in the pressure chamber. The chamber was then sealed, the system then evacuated to remove air and subsequently repressurised to a predetermined level and maintained steady at that level. The \( \text{N}_2 \) gas was allowed to permeate the coal sample and flow out through the central hole which is shown in Figure 1b. The released gas from the coal flows through a measuring system, consisting of a vacuum pressure sensor and a line of gas flow meters of 0-2 L/min and 0-15 L/min measurement ranges respectively.

The test sequence was followed in steps of varying vertical stress of 1, 2, 3 and 4 MPa. For each selected vertical loading, confining gas pressures varying between 0.2 MPa to 3 MPa were applied. The load cell, flow meters, pressure transducer and strain gauges were connected to a PC through a data logger for data collection.

Testing results and analysis

The permeability of the sample was calculated using the following Darcy’s equation:

\[
K = \frac{\mu Q \ln \left( \frac{r_2}{r_1} \right)}{\pi L (P_1 - P_2)}
\]

Where \( K \) is the permeability of coal, \( \mu \) is viscosity of gas, \( Q \) is the flow rate of gas, \( L \) is the height of the sample, \( r_2 \) and \( r_1 \) are the external radius and internal radius of sample, \( P_1 \) and \( P_2 \) are absolute gas pressure inside and outside of chamber, respectively.
Figure 3 shows the permeability test result with MFORR with N₂ pressurisation. For each of the vertical stress, coal sample permeability decreases with increasing gas pressure and at higher gas pressure, coal permeability stays stable and changes very little, under different vertical stresses. Test results show that the permeability values stay below 2 mD when the applied confining gas pressures became greater than 0.5 MPa.

![Permeability Test Results with MFORR](image)

Figure 3 - Coal permeability test result with MFORR

Figure 4 shows coal strain behaviour in the MFORR permeability test. Test results show that the degree of the axial strain both axially and laterally are influenced by the level of pressures that sample being subjected under triaxial environment.

There is an increased compaction of the coal layers parallel to bedding with increased vertical stress due to applied axial loads perpendicular to layering. The degree of axial shrinkage has increased with increasing axial stress as demonstrated in Figure 4a. Also, the level of vertical or axial strain reduction has reduced with the increase in the applied lateral gas confining pressure. The level of lateral/horizontal strain was affected by the level of the applied axial load as well as the confining gas pressure, this time in reverse order. That is, at high vertical stress of 4 MPa, the confining lateral stress was the greatest, while the least applied axial stress contributed to increased maximum lateral strain. Also and irrespective of the level of the axial stress the horizontal stain levels tapered off gradually with gradual increase of the applied confining gas pressure as demonstrate in Figure 4b.

These results clearly demonstrate the coal sample underwent negative volumetric changes or shrinkage with increased confinement pressures axially and laterally, and that the degree of the volumetric changes will be dependent on the level of the applied axial and lateral pressures or stresses.

![Vertical Strain Results in Permeability Test with MFORR](image)

(a)

![Horizontal Strain Results in Permeability Test with MFORR](image)

(b)

Figure 4 - Coal strain behaviour in the permeability test with MFORR
TRIAxIAL PERMEABILITY STUDY WITh TRIAXIAL COMPRESSION APPARATUS

Apparatus

The setup of the triaxial compression apparatus is shown in Figure 5. This apparatus, which can be utilised in normal triaxial permeability test of coal comprises a number of components, including:

- A main apparatus loading system for holding and loading the pressure cell;
- High pressure cell for holding the coal sample in triaxial permeability test;
- A axial loading and measuring device;
- Oil pump for generating and maintaining the confining pressure applied to the coal sample;
- A pressure transducer for measuring the pressure inside the cell;
- A pressure transducer for measuring the pore pressure;
- Flow meters for measuring the gas flow rate;
- A data acquisition system.

![Triaxial compression apparatus](image)

**Figure 5 - Triaxial compression apparatus**

In this apparatus, the cell pressure is controlled manually by a hydraulic jack and a pressure transducer, which is mounted on the cell to ensure the required confining pressure. As the cell is made of high-yield steel it can withstand a maximum pressure of 150 MPa with a safety factor of two. The cell is capable of carrying out high confining pressure tests, making it suitable to simulate a high in situ stress environment in coal measure rocks. The axial load is applied by a servo-controlled compression test machine with the maximum force of 250 kN.

Coal sample preparation

The standard core samples with dimension of 54 mm in diameter and 100 mm in height were drilled from the same lump coal sample as the MFORR permeability test samples, which were also typical Bulli seam coal samples. Figure 6 shows the snapshot of the sample.

Testing procedure

The procedure for conducting each test consisted of the sample being correctly installed inside a membrane, the specimen was placed into the high pressure cell where a small axial load was applied firstly to keep it stable; then oil was pumped into the cell until the cell was filled with oil with both the axial load and confining pressure applied at predesigned values. Subsequently N₂ gas pressure was applied at a predetermined level and N₂ gas flowed through the coal sample from bottom to top which
was shown in Figure 5b. The released gas from the coal flowed through a monitoring system consisting of gas flow meters with 0-2 L/min and 0-15 L/min measurement ranges.

The test sequence was followed in steps, with different vertical stresses of 3, 4, 6 and 8 MPa respectively. The gas pressure was charged initially at 0.2 MPa then increased gradually to higher pressure in steps reaching a maximum of 3 MPa. The load cell, flow meters, pressure transducer were all connected to a PC through a data logger for data collection.

![Figure 6](image)

(a) (b)

Figure 6 - Coal samples for triaxial permeability test with Triaxial Compression Apparatus

Testing results and analysis

The permeability of the sample was calculated using the following Darcy’s equation:

\[
K = \frac{2\mu Q L}{A(P_1 - P_2)}
\]  

(2)

Where \(K\) is the permeability of coal, \(\mu\) is viscosity of gas, \(Q\) is the flow rate of gas, \(L\) is the length of the sample, \(A\) is the cross section of specimen, \(P_1\) and \(P_2\) are the inlet and outlet absolute gas pressure, respectively.

Figure 7 shows the triaxial permeability test results with \(\text{N}_2\) at different vertical stresses. Tests with a vertical stress of 3, 4, 6 and 8 MPa were examined. For each of the vertical stress, two horizontal stresses were examined, coal sample permeability decreased with the increasing gas pressure. At higher gas pressures, coal permeability stayed constant, a similar trend as with the permeability test with MFORR. At each vertical stress, coal permeability test decreases with the increasing horizontal stress.
Figure 7 - Coal triaxial permeability test with a certain vertical stress

Figure 8 shows the triaxial permeability test results at various horizontal stresses. Tests at horizontal stresses of 4 and 5 MPa are analysed in this study. At each of the horizontal stresses, coal sample permeability decreases with increasing vertical stress.

It can be observed from the tests that the permeability values are well below 2 mD under the triaxial test conditions.

Figure 8 - Coal triaxial permeability test with a certain horizontal stress

MFORR PERMEABILITY AND TRIAXIAL PERMEABILITY TEST RESULTS COMPARISON

Figure 9 shows a comparison of the permeability results between the MFORR and triaxial tests at suitable vertical stresses. Although the results show some significant difference in permeability values at lower confining gas pressure because of the relatively low confining pressure of MFORR test, the permeability converges to a steady level below 2 mD under high triaxial stress conditions portraying the near in situ conditions of the Bulli seam. There is no significant mathematical difference between the two different types of testing apparatus and calculation method.

Similar results are confirmed with the other studies, Hayes (1982) reported that the Bulli seam permeability is considerably less than 1 mD. Lingard et al. (1984) reported permeability of Australian coals from Appin, West Cliff and Leichhardt collieries that varied from less than 0.1 mD to 100 mD. Recently the Bulli seam permeability was measured using a combination of injection/falloff and step-rate testing methods (Jackson, 2004) and the results from 31 locations of the Bulli seam at West Cliff Colliery
(Fredericks, 2008; Black, 2012), the average in situ permeability is 2.2 mD, with the range extending from a low of 0.005 mD to a high of 5.8 mD.

![Figure 9 - MFORR permeability and triaxial permeability test results comparison](image)

**CONCLUSIONS**

Permeability testing with the MFORR can be used to study the relationship between axial stress, gas pressure and coal permeability. Tests show at each of the vertical stress, coal sample permeability decreases with increasing gas pressure and at higher gas pressure, coal permeability stays stable and its changes under different vertical stress become relatively smaller.

Strain gauge results from the MFORR test clearly demonstrate the coal sample underwent negative volumetric changes or shrinkage with increased confinement pressures axially and laterally. The degree of the volumetric changes is found to be dependent on the level of the applied axial and lateral pressures or stresses.

Permeability testing using the high pressure conventional Triaxial Compression Apparatus can be used to study the relationship between axial and confining stress, gas pressure and coal permeability under triaxial condition. Coal sample permeability decreased with the increasing gas pressure. At higher gas pressures, coal permeability stays constant, a similar trend as with the permeability test with MFORR.

In the permeability test with Triaxial Compression Apparatus, at each vertical stress, coal permeability test decreases with the increasing horizontal stress and at each of the horizontal stress, coal sample permeability decreases with the increasing vertical stress.

It is concluded that there is no significant mathematical difference between the two types of testing apparatus and calculation methods. Both of the permeability tests are comparable and tally's well with the Bulli coal seam tests result calculation from in situ condition. A permeability of <2 mD should be adopted under high triaxial stress conditions.

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APPLICATION OF A NUMERICAL MODEL FOR OUTBURST PREDICTION, CONTROL AND MANAGEMENT

Xavier S. K. Choi

ABSTRACT: The basic underlying mechanism for outburst initiation involves the expulsion of coal at a pressure gradient above a critical value which is directly related to the strength and porosity of the coal at the current state, and the composition (degree of gas saturation) of the pore fluid. Coal strength, porosity, stress, gas pressure and pressure gradient are important for outburst initiation. Permeability and rate of desorption can be important for outburst evolution by controlling the amount of gas that would become available to drive an outburst. The severity of an outburst depends on gas pressure, the hydrodynamic force, the strength and toughness of the coal, and the amount of free gas that becomes available during an outburst. For the same pressure gradient, the degree of violence is greater for weaker and more friable coal. Outburst propensity can be reduced by changing the method of mining, mine geometry, and the preventive and control measures adopted by the mines.

The relative importance of the various factors and parameters will depend on the conditions of individual mines. As the interaction among the various processes and factors leading to outburst can be very complex, it is necessary to treat the coal-rock-stress-structure-gas interaction as a system. For outburst prediction, one approach is to use a numerical model that can model the individual processes and their interactions. This paper lists some of conclusions that have been derived from the results of the laboratory experiments and the modelling studies conducted to date and describes how the model can be used to help a particular mine assess outburst proneness and the potential risks, and to identify the critical factors for the purpose of outburst control and management. Based on the assessed risk and the degree of uncertainty, one may choose complete prevention or suitable control and management measures, without undermining safety which is one of the most important considerations.

INTRODUCTION

An outburst is a mechanical process which involves the transport of coal, and possibly also some rocks from the adjacent strata, which have failed due to tectonic history or mining induced stress redistribution. The outburst coal is expelled by free gas which is under pressure and which can generate enough force to mobilise and transport the coal. The speed at which the coal is expelled depends on the size of the fragmented coal, the amount of potential energy in the gas, and the drag and pressure forces generated by the gas on the coal. Even though outbursts can be broadly defined as dynamic events involving the instantaneous expulsion of coal and gas in underground coal mines, each outburst may occur under different sets of conditions, with different manifestations.

A lot of research on outbursts has been conducted both in Australia and overseas over many decades. It has been suggested that the main factors for outburst initiation are stress, strength, gas pressure gradient and the amount of gas that is available to drive an outburst (Briggs, 1921; Ruff, 1930). The parameters which have been used for outburst prediction include strength, fracture toughness (or energy required to form new fracture surfaces), reservoir pressure, gas content, rate of gas desorption, porosity, and geological structures.

Various indices have been developed and used for outburst prediction by incorporating some of the factors and parameters mentioned above. However, as suggested by Lama (1995), all the methods based on some parameters or indices for outburst prediction “can be used for defining the proneness of a seam or a part of the seam prior to mining, but this is only a descriptive method and does not help in forecasting an outburst condition.” A specific set of parametric or indicial values may work well for a particular mine, but it may not work for a different mine because of operational issues or different in situ conditions. It is therefore not unusual that the adopted values are sometimes adjusted for different mines (Black, et al., 2009; Liu, et al., 2011).
Based on the work of Ripu Lama, the Outburst Mining Guideline: MDG 1004, prepared by the Outburst Guideline Committee of the Department of Mineral Resources of New South Wales in 1995, requires that for mines mining the Bulli seam, normal mining can only proceed if the gas in the barrier region around the mine opening has been drained to below the gas content Threshold Limit Values (THV’s). The THV’s depend on gas composition and THV’s of 6.4 m$^3$/t and 9.4 m$^3$/t for 100% CO$_2$ and CH$_4$ respectively were suggested by Lama (1995). The THV’s have later been revised slightly for some mines. The approach based on gas content thresholds has worked well, with a few exceptions, in preventing outbursts in Australia since its introduction. Pre-mining gas drainage is also a common practice in Poland, China and Russia for controlling outbursts (Lama and Saghafi, 2002). In China, gas pressure instead of gas content threshold value is used in some of the mines as one of the indices for outburst control. In one of the mines, tectonically undisturbed it was considered safe to mine if gas content less than 12 m$^3$/t (most of seam gas in the Chinese mines has a high methane composition). This coincides with Lama’s (1995) suggested THV for mining in a 100% CH$_4$ environment in the absence of structures. However, in some areas of the mine, based on the sorption properties of the coal, “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 a outburst threatened area and when gas content is higher than 20 m$^3$/t, the coal seam will be determined as having outburst potential” (Liu, et al., 2011). Based on a gas pressure threshold value of 0.74 MPa for tectonically disturbed coal, the corresponding gas content can be as high as 21.68 m$^3$/t because of the sorption properties of the particular coal (Liu, et al., 2011). It has been demonstrated in some Australian mines through grunching and remote mining that, in some areas, there can be no outburst when the gas content was as high as 14 m$^3$/t. It however raises the question, as suggested by Eade (2002), of what the inherent safety factor is for a given threshold value. Also, It has been suggested that CO$_2$ outbursts are more violent than CH$_4$ outbursts, but it should be noted that some of the largest outbursts in the world did occur in mines rich in CH$_4$ (Lama and Saghafi, 2002). As CO$_2$ is usually associated with structures in Australian mines, can the more violent nature of the CO$_2$ outbursts be partly explained by the characteristics of the structures that they are associated with besides the higher sorption capacity of coal for CO$_2$? Lama (1996) however did suggest that the threshold limit value can be increased to 10 m$^3$/t for 100% CO$_2$ in the absence of structures. In a mechanistic sense, it is the pressure and relative flow velocity of the free gas which contributes to outburst initiation and evolution. It is therefore important to understand how sorption capacity and rate of desorption affect the temporal evolution of gas pressure around the face in the seam. One may ask whether we should use reservoir pressure instead of gas content as the threshold for outburst management, taking into account the physical properties of the coal and geological structures, and their potential variability in the seam. This however suggests that, in the absence of structures, the gas content threshold value for CO$_2$ could be higher than CH$_4$ because of their adsorption properties (adsorption isotherms) even though the threshold values may need to be adjusted for the effects of higher sorption capacity and desorption rate of CO$_2$ compared to CH$_4$. There are some other questions that still need to be answered such as what would be suitable threshold values when in situ stress and reservoir pressure become higher, permeability may become lower, and CO$_2$ may exist in a supercritical state as mines get deeper.

**NUMERICAL OUTBURST MODEL**

Through a number of projects supported by ACARP and CSIRO (Wold and Choi, 1999; Choi and Wold 2003a; Wold, et al., 2006; and Choi and Wu, 2008), a numerical model for outburst initiation and evolution was developed by linking a geomechanical model (Choi, et al., 1991, 1992; Choi and Tan, 1998) with a coalbed methane reservoir simulator (Spencer, et al., 1987; Stevenson, et al., 1994; Stevenson, 1997). The model can be used to delineate the mechanisms, and to answer some of the questions mentioned. Details of the model and the modelling approaches, and examples of the model application can be found in Choi and Wold (2001a, 2001b, 2003b, 2004), Choi and Wu (2005), Wold and Choi (2001), and Wold et al. (2008). The numerical model can be useful where guidance from past experience may not be available. As there can always be some degree of uncertainty with respect to geology, and the variability of the coal and the adjacent rock strata, the main value of the model is its ability to answer some of the “what if” type questions.

**OUTBURST MECHANISMS**

**Laboratory model outburst tests**

In order to get a better understanding of outburst mechanisms, it may be useful to look at the results of some of the laboratory model outburst tests which were conducted during ACARP project C13012 (Choi
and Wu, 2008). The effects of coal strength, reservoir pressure, pressure gradient and gas composition on outbursts are demonstrated by the experimental results.

The model outburst tests were conducted using a cavity index cell as shown in Figures 1 and 2. The cell was initially developed for laboratory cavity completion experiments (Wold, et al., 1994; Paterson and Wold, 1995). The sample was placed into the cell by sliding it inside the rubber membrane. The back end of the sample rested against the end cap which had a port through which pore pressure could be measured. Steel rings with a central hole could be inserted between the sample and the end cap to ensure that the sample was thrust against the end cap.

During the model outburst tests, the gas pressure at the front was released by opening the air operated valve (see Figure 2), the pressure could be reduced to atmospheric pressure in the order of 200-300 milliseconds.

The cylindrical piston applied a compressive stress to the front end of the sample during the application of the pore pressure, and held the sample in place against the forces generated by the back pressure during the outburst experiments. The free surface area of the sample during the tests was that within the 30 mm inner diameter of the piston. The gas was discharged into an expansion chamber-muffler and the coal which was ejected during the outburst test was collected in a bag.

The results show that, if the gas pressure in the coal samples is higher than a certain value for a given uniaxial compressive strength, outburst will be induced with the formation of a cavity (see Figures 3a and 3b). The size of the cavity is larger at higher gas pressure and for weaker coal. Discing can occur at higher pressure (see Figure 4). However, for tests under the same gas pressure and for coal samples with similar strength, no apparent difference in the size of the cavity was observed.

![Figure 1 - Experimental set-up of model outburst tests](image1)

![Figure 2 - Closer view of test set-up](image2)
It was found that, from the work conducted in some of the earlier ACARP projects (Wold and Choi, 1999; Choi and Wold, 2003), for outburst risk analysis and for outburst control and management, it is important to be able to assess both the likelihood of an outburst event and the consequence in case such an event does happen. As risk is measured in terms of likelihood and consequence, the risk control measures can be dependent on the potential consequence.

A series of parametric studies was conducted using the “coupled” model to identify which are the key variables in outburst initiation, and which are the less important variables. These model results strongly support the importance of gas pressure and pressure gradient, coal strength and geological structures in determining threshold values for outburst risks. Some later work also suggested the importance of porosity and pore structure (including the geometry of the fracture network).

The influence of other variables such as the orientation of the principal components of the in situ stress, the effects of changes in stress on permeability, rate of mass transport between adsorbed gas and free gas, and heading advance rate were also studied. A certain degree of understanding of the significance of those variables on outburst initiation was obtained. However, in contrast to the general experience that areas of high CO₂ content are more hazardous with respect to outburst compared to areas with high CH₄ content, the model predicted, under the modelled conditions, a slight reduction in outburst initiation potential with an increase in the CO₂ proportion in the gas composition for the same initial reservoir and desorption pressures. The higher rate of desorption for CO₂ compared to CH₄ may however play a certain role during post-initiation outburst evolution. As CO₂ can cause a higher degree of coal matrix swelling/shrinkage compared to CH₄ when undergoing similar change in desorption pressure, strength reduction associated with CO₂ adsorption/desorption can be explained by the mechanical damage that is caused by the differential swelling/shrinkage as the strain distribution at different distances from the coal surface is not uniform. It should however be noted that no apparent
difference in the size of the outburst cavity was observed for the laboratory model outburst tests conducted on coal samples with similar strength under the same gas pressure.

Gas drainage to below the gas content threshold values would be much more difficult for CO\(_2\) than CH\(_4\) because of the much lower desorption pressure for CO\(_2\) corresponding to the threshold gas content value, this would imply a much higher degree of reservoir pressure drawdown (or drainage) is required for CO\(_2\). Application of suction would have obvious benefit for CO\(_2\) drainage. Borehole inclination for long drainage holes may also have an important impact on CO\(_2\) drainage because of the hydrostatic pressure from the water in the borehole.

An outburst occurs whenever the force provided by the gas at a given pressure gradient is enough to mobilise and transport the coal at the face. The required force is a function of the strength of the coal at its current state. Post-initiation evolution depends on additional factors such as fracture toughness (which is very low for sheared or mylonitised coal but can be quite high for some strong coal) and the source of free gas. Outburst occurs whenever the conditions are satisfied, including at shallow depths. Outburst management based on gas composition and gas content threshold values can be either under- or over-conservative even though outbursts that occur below the threshold gas content value are expected to be "mild".

Geological structures play a role in outburst through modifying the strength, permeability and/or porosity and pore structure of the coal, the amount of free gas, and/or pressure and pressure gradient.

Because of the difficulty of detecting some small local heterogeneity such as some pockets of very weak materials, some very small scale "outburst" is difficult to avoid. For some cases, body force due to gravity can contribute to an outburst.

Gas desorption rate may or may not play an important role depending on how it may contribute to the spatial and temporal variation in pressure distribution as mining progresses. For rate of desorption to have an important impact during an outburst the coal has to be in the form of very small particles. Mylonitic coal can be more outburst-prone simply because of its low strength and higher porosity than normal coal.

During an outburst, the first law of thermodynamics (or the law of conservation of energy) is obeyed. By identifying all the different forms of energy that are available in the system to drive an outburst and the energy that is required for the different processes during an outburst, it should be possible to make some initial assessment whether an outburst is likely to occur and the scale of a potential outburst.

Based on the underlying mechanisms, outburst prevention can be through reduction of pressure and pressure gradient (such as gas drainage) and/or minimisation of mechanical damage to the coal (through stress relief or strengthening of the coal), or by reducing the pressure gradient and hydrodynamic forces and energy (such as filling the pore space with a much less compressible fluid) that is available to dislodge and transport the coal.

By considering the outburst mechanisms and the first law of thermodynamics, and taking into consideration the effect of porosity on the pressure gradient and hydrodynamic forces, the current gas content threshold value can be too conservative for some low permeability but reasonably strong coal.

The numerical outburst model is able to predict how pressure, the relevant strength parameters and stress around the face evolve as mining progresses. However, one of the major challenges is the availability of field data, including the detection and characterisation of outburst prone structures.

It is possible that all outbursts are associated with some types of structures (including cleat fractures), whether they are pre-existing or mining induced unless the coal is inherently very weak for some reasons.

It may be more important to ensure that the pressure in the outburst prone structures has been reduced to below a certain critical level than trying to reduce the gas content of the seam.

For seams with very low permeability and porosity, reasonably strong coal, and if there is no problem with mine air ventilation and other gas issues, it may be even safer to mine without gas drainage (to keep the seam fully water saturated) under certain conditions.
It may be more important to use drainage holes to ensure that sufficient gas will be drained from outburst prone structures and to monitor reservoir pressure in addition to gas content. One major issue is the integrity of drainage holes as mylonite, sheared coal and coal associated with other outburst prone structures can be weak. In grounds with high *in situ* stress, borehole stability can be a problem. Borehole collapse may occur leading to blockage of drainage holes, which can lead to difficulty in draining the gas and allowing pressure to build up. Drainage may not occur where it is needed most.

**APPLICATION OF THE NUMERICAL OUTBURST MODEL FOR OUTBURST CONTROL AND MANAGEMENT**

The main advantage of the numerical outburst model is that outburst prediction can be made based on the conditions of the mine, and it can be used to predict how the various field variables such as pressure and stress, and coal properties may change as mining progresses, and the model can be updated if new information becomes available such as the detection of some previously unknown structures. Another major advantage is that sensitivity analyses can be conducted to predict different possible outcomes by taking into account the uncertainty in some of the field data (Wold, *et al.*, 2006; 2008). The model can also be used to predict what would be the likely mechanism for outburst occurrence with the given field data. Advancement in *in situ* measurement and ground characterisation ahead of mining and roadway development would certainly be useful in providing the required data, and in improving the accuracy of the model predictions.

**CONCLUSIONS**

The current use of gas content threshold values for outburst control and management has been very successful in preventing major outbursts from happening. It is apparent that the inherent safety factor for any adopted value can be different for different mines, and at the different stages of mining and roadway development. As it is largely an empirical approach, there are a number of questions that still need to be answered. The adoption of overly conservative gas content threshold values may cause some operational issues for some mines. Use of gas pressure instead of gas content threshold as one of the indices for outburst prediction is practised in some coal mines in China. The main difficulty in outburst prediction is that outburst is a phenomenon which involves the interaction of a number of factors and processes. Any analytical or numerical approach for outburst prediction needs to be able to account for the individual processes and their interaction, it is here where the numerical outburst model that has been developed to date would be useful. The model can be used to help a particular mine to identify the major mechanisms and critical factors for outburst control and management purpose. Based on the assessed risk and any major operational issues, one may choose complete prevention or suitable control and management measures may be chosen without undermining safety.

**ACKNOWLEDGEMENTS**

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**REFERENCES**


INVESTIGATING THE INFLUENCE OF REACTIVE PYRITE ON COAL SELF-HEATING

Basil Beamish¹,², Zhang Lin² and Rowan Beamish¹,²

ABSTRACT: The acceleration of coal self-heating has long been attributed to the presence of reactive pyrite. However, a definitive means of quantifying this effect has been lacking, particularly from the low ambient temperatures experienced at mine sites. A recently developed moist coal adiabatic oven test has been used to investigate the influence of reactive pyrite on self-heating of a high volatile bituminous coal containing sulphur concentrations from 0.62% to 17.95%. A relationship exists between the amount of pyritic sulphur in the coal and the time taken to reach thermal runaway. However, simply measuring the pyritic sulphur concentration of a coal is not sufficient to quantify the accelerated self-heating effect, as it is the form of the pyrite that determines the pyrite reactivity. These findings will be expanded on in the paper as they have a major significance for the risk assessment of coal self-heating.

INTRODUCTION

The presence of pyrite in coal has been recognised as a contributing factor to self-heating for many years, but there has been no standard test developed that quantifies the effect (Miron, et al., 1992). The fundamental reaction for pyrite self-heating is described by Weise, Powell and Fyfe (1987) as:

$$\text{FeS}_2 + 8\text{H}_2\text{O} + 7\text{O}_2 \rightarrow \text{FeSO}_4.7\text{H}_2\text{O} + \text{SO}_4^{2-} + 2\text{H}^+$$

This reaction is strongly exothermic and the resultant hydrated sulphate is known as melanterite, which has a well-defined crystal form. It can be seen that the pyrite oxidation reaction is dependent on the presence of both moisture and oxygen. This feature has practical implications for testing.

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 has routinely been used. This is the adiabatic oven R₇₀ self-heating rate test (Beamish, et al., 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 R₇₀ 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 R₇₀ values and coal characteristics. As such a reactivity rating scale has been developed for both New South Wales and Queensland conditions using this parameter.

Beamish and Beamish (2010) proposed a new moist coal adiabatic oven test that could 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 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. In particular, Beamish and Beamish (2011) showed that the new Moist Adiabatic Benchmark (MAB) test was able to quantify the effect of reactive pyrite on the self-heating of a Bowen Basin coal.

This paper presents the results of MAB testing on several samples of a Bowen Basin high volatile bituminous coal with sulphur contents ranging from 0.62% to 17.95% that show the effect of increasing pyrite content on coal self-heating rates.

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ADIABATIC OVEN TESTING

Coal samples

Details of the three samples used in this study are contained in Table 1. The two major benchmark coals are Kideco (Indonesia) and Spring Creek (New Zealand). The Bowen Basin high volatile bituminous coal is from an undeveloped coal deposit, which is not the same as that tested by Beamish and Beamish (2011). All samples were received as fresh cores 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. It should be noted that initially only R70 testing was conducted on the samples and the MAB testing was conducted some 18 months later. All samples were stored in a freezer between tests. The 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>R70 (°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>Benchmark coals</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</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>
</tr>
<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>
</tr>
<tr>
<td>Bowen Basin high volatile bituminous coal</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>BBHVB03</td>
<td>12.33</td>
<td>40.6</td>
<td>11658</td>
<td>hvCb</td>
<td>4.8</td>
<td>0.71</td>
<td>16.6</td>
<td>26.9</td>
</tr>
<tr>
<td>BBHVB06</td>
<td>7.20</td>
<td>47.4</td>
<td>12061</td>
<td>hvCb</td>
<td>7.9</td>
<td>4.93</td>
<td>12.7</td>
<td>27.3</td>
</tr>
<tr>
<td>BBHVB13</td>
<td>7.11</td>
<td>47.9</td>
<td>12882</td>
<td>hvCb</td>
<td>15.7</td>
<td>8.10</td>
<td>11.7</td>
<td>27.2</td>
</tr>
<tr>
<td>BBHVB01</td>
<td>5.52</td>
<td>41.5</td>
<td>11600</td>
<td>hvCb</td>
<td>29.3</td>
<td>19.53</td>
<td>10.3</td>
<td>27.5</td>
</tr>
</tbody>
</table>

Self-heating test procedure

The R70 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 R70 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 R70 method for MAB 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 DISCUSSION

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 the Bowen Basin samples have a very high to ultra high intrinsic spontaneous combustion reactivity rating that is associated with the change in mineral matter content of the coal as indicated by ash content. The highest ash content sample (BBHVB01) has the slowest self-heating rate and the lowest ash content sample (BBHVB03) has the fastest self-heating rate. This is consistent with the heat sink effect of increased mineral matter demonstrated by Beamish and Blazak (2005) and Beamish and Sainsbury (2008).
Figure 1 - Adiabatic self-heating curves for samples tested using the normal $R_{70}$ test procedure, showing intrinsic spontaneous combustion reactivity ratings based on Queensland conditions (H = High, VH = Very High, UH = Ultra High, EH = Extremely High)

Table 2 contains the breakdown of the forms of sulphur in each of the Bowen Basin coal samples. It is clear that the sample with the highest total and pyritic sulphur content (BBHVB01) has the lowest $R_{70}$ self-heating rate. However, this appears to be contradictory to the concept of increased self-heating rate due to the presence of pyrite in the coal. The reason for this is that the $R_{70}$ self-heating rate test is performed on a dry basis, after the moisture has been removed from the coal. Therefore, this test does not record any pyrite oxidation reaction as there is no moisture present, and hence the simple heat sink effect of additional mineral matter in the coal controls the self-heating rate under these conditions. It is for this reason that the $R_{70}$ self-heating rate test has not identified coals with possible pyrite self-heating acceleration in the past. Similarly, other spontaneous combustion testing methods where moisture has been completely or partially removed from the coal sample will produce the same erroneous result.

Table 2 - Forms of sulphur for Bowen Basin coal samples (% air-dried basis)

<table>
<thead>
<tr>
<th>Sample</th>
<th>Pyritic S</th>
<th>Sulphate S</th>
<th>Organic S</th>
<th>Total S</th>
</tr>
</thead>
<tbody>
<tr>
<td>BBHVB03</td>
<td>0.06</td>
<td>0.09</td>
<td>0.47</td>
<td>0.62</td>
</tr>
<tr>
<td>BBHVB06</td>
<td>1.70</td>
<td>0.13</td>
<td>2.55</td>
<td>4.38</td>
</tr>
<tr>
<td>BBHVB13</td>
<td>4.61</td>
<td>0.07</td>
<td>2.66</td>
<td>7.34</td>
</tr>
<tr>
<td>BBHVB01</td>
<td>10.43</td>
<td>1.02</td>
<td>6.50</td>
<td>17.95</td>
</tr>
</tbody>
</table>

Moist adiabatic benchmark comparison

Results of tests using the new moist adiabatic benchmark method are shown in Figure 2. These results clearly demonstrate the accelerating effect of pyrite oxidation on coal self-heating. As the pyrite content increases, the self-heating rate of the coal also increases. This is shown in Figure 3 where the time taken to reach thermal runaway (100°C) is non-linearly related to the pyrite content of the coal. In a practical sense, the time taken to reach thermal runaway for each sample is indicated in Table 3, based on the results for the two benchmark coals shown in Figure 2. Lower and upper limits are calculated as it is impossible to give a precise projection, due to the many competing variables that affect hot spot development in a bulk coal pile. However, to date these benchmark limits have proven extremely accurate and it is clear that mining and handling of this coal would require a rigid spontaneous combustion management plan that identified the extent of the hazard. Pyritic sulphur removal by washing would be required and the effectiveness of this strategy could be quantified by the MAB test.

The moist adiabatic benchmark test not only shows that the key chemical ingredients for the pyrite oxidation reaction to take place are moisture and oxygen, it also identifies whether the pyrite is present in a form that is readily oxidised. Tests on other coals with high pyritic sulphur contents (often readily identifiable by the presence of visible pyrite grains on cleat or as massive nodules) have shown no reaction using the MAB test. This is because the particle size of the pyrite is too coarse for the oxidation reaction to occur at a rate sufficient to cause accelerated coal self-heating. In the case of the Bowen Basin samples presented in this paper the pyrite occurs as fine layers that are parallel to banding (Figure 4). The thickness of the layers is of the order of microns and thus provides greater surface area access for the oxidation reaction to take place. These layers are presumably composed of submicron pyrite crystals.
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 - Relationship between the time taken to reach thermal runaway and pyrite sulphur content of coal.

Table 3 - Laboratory measured time to thermal runaway ($t_{TR}$) and calculated values for mining situations

<table>
<thead>
<tr>
<th>Sample</th>
<th>Laboratory $t_{TR}$ (h)</th>
<th>Minesite/Stockpile $t_{TR}$ (days)</th>
</tr>
</thead>
<tbody>
<tr>
<td>BBHVB03</td>
<td>31.9</td>
<td>16</td>
</tr>
<tr>
<td>BBHVB06</td>
<td>14.0</td>
<td>7</td>
</tr>
<tr>
<td>BBHVB13</td>
<td>9.1</td>
<td>5</td>
</tr>
<tr>
<td>BBHVB01</td>
<td>2.8</td>
<td>2</td>
</tr>
</tbody>
</table>

Figure 4 - Scanning electron microscope image of fine pyrite layering in a Bowen Basin high volatile bituminous coal.
The resultant products of the pyrite oxidation have been identified by X-ray diffraction and found to be composed of rozenite \((\text{FeSO}_4 \cdot 4\text{H}_2\text{O})\), melanterite \((\text{FeSO}_4 \cdot 7\text{H}_2\text{O})\) and roemerite \((\text{FeFe}_2(\text{SO}_4)_4 \cdot 14\text{H}_2\text{O})\). Crystals of these hydrated sulphates can be seen on the exposed coal surfaces in Figure 5. The formation of all these products is highly exothermic. From a practical viewpoint any coal seams that exhibit the growth of these hydrated iron sulphate crystals should be treated with extreme caution in terms of spontaneous combustion management planning. MAB testing of samples will readily quantify the spontaneous combustion propensity of such seams.

![Figure 5 - Bowen Basin high volatile bituminous coal sample showing crystal clusters of hydrated iron sulphates on exposed surfaces](image)

CONCLUSIONS

Adiabatic oven tests of samples from an undeveloped Bowen Basin coal deposit have shown the accelerating effect of pyrite oxidation on coal self-heating. This has only been possible using a new moist adiabatic benchmark test developed by Beamish and Beamish (2010). The older R70 self-heating rate test method is unable to quantify the effect as it is performed on a dry basis and one of the key chemical ingredients in the pyrite oxidation reaction is moisture. In fact, the R70 test produces a result that is completely erroneous due to the fact that it only measures the reactivity of the coal, which is strongly influenced by heat sink effects from the presence of increasing mineral matter. Hence, the R70 value of the coal decreases with increasing pyrite presence (increasing mineral matter).

The form of the pyrite present in the coal is also important in governing the rate of the pyrite reaction. Visible granular or massive nodular pyrite does not react fast enough to cause accelerated coal self-heating. This is readily identified by the new MAB test. The presence of finely disseminated pyrite that readily oxidises can be recognised in the field by the growth of hydrated iron sulphate crystals on exposed coal surfaces.

The MAB test used to quantify the pyrite oxidation effect can also be used to quantify the time taken to reach thermal runaway in mine site situations. A possible management strategy to combat the spontaneous combustion hazard of this coal would be to remove the pyrite by washing. The effectiveness of this strategy can be quantified using the MAB test.

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RADON MEASURING TO DETECT COAL SPONTANEOUS COMBUSTION FIRE SOURCE AT BULIANTA MINE, SHENDONG

Jianming Wu, Yuguo Wu, Junfeng Wang and Chunshan Zhou

ABSTRACT: The paper introduces the theory of using radon to detect the source of coal fires resulting from spontaneous combustion and its successful application at the No.12405 gob area in Bulianta Mine, Shendong. The practice shows that this method provides the scientific basis for coal seam spontaneous combustion control plan-making. It is a key technology and can be applicable for wide applications.

INTRODUCTION

Coal spontaneous combustion fires can be a kind of significant disaster for coalmines. The prediction of spontaneous combustion and fire source detection are the most critical techniques. The location where the spontaneous fire happens can be several hundred meters underground, this makes the fire source elusive and hard to be approached. Therefore, the technology of how to detect the exact fire source becomes more important for underground fire extinguishment. This is also a world-wide problem (Zhang, 2008; Wu, 2008). Taiyuan University of Technology (TUT) invented a system to detect the location of the spontaneous fire source by measuring radon. Using radon as a carrier, this system can not only detect the location and the range of the fire source, but also predict the trend of fire behavior and the changing status of the underground fire. The system has the advantages of easy operation, less cost, anti-jamming and fast reaction speed. Input of the detected data into the Radon Measuring and Fire Detecting Data Processing System (CDTH) can indicate the spontaneous combustion fire location, the fire area and the fire behavior. Compared with similar technology at home and abroad, the practice showed that it is more accurate (fire centre 90% precision, and 85% for the fire edge) and can detect down to 800 m in depth. The radon measuring method has been successfully used at coal mining areas in China and Australia and it is the only practical technology for underground coal spontaneous fire source detection (Wang, 2010; Jia, et al., 2002; Zhao and Wu, 2002).

THE BULIANTA MINE AND THE FIRE ZONE

Bulianta mine is the biggest underground mine in the world. Built and operated by Shendong Coal Group, it is located at Ordos Inner mongolia with 106.43 km² of mining area, 1 550 Mt of mineable reserve and 77 years of service life. The major coal seams are No.1-2, No.2-2 and No.3-1. The total production reached 25 Mt in 2010.

On 13th November 2011, at the transport gateroad of No.12 405 working face, CO was found over limit from the observation hole of the permanent anti-fire wall. The CO data went up to 900 ppm on 16th. The CO detection of the crosscut at No.12 406 working face reached 200 ppm and the data at the crack on the coal rib reached 2 000 ppm. The dangerous gases like ethane (C₂H₆), ethylene (C₂H₄) and ethyne (C₂H₂) were also detected. Preliminary analysis showed that spontaneous fire happened at the gob area near the main return of No.12 405 working face. It is very dangerous to the coalmine safety. The coal mine adopted all the fire combat measures they could such as surface grouting, nitrogen injection, thick mortar injection, pressure equalising for No.12 406 working face, ground sealing and ground borehole casting. But the fire could not been totally controlled from the evidence of the gas analysis. In order to find the exact location of the spontaneous fire source, TUT conducted a fire source detection from 17th to 23rd Nov, 2011. The total investigation area is about 120 000 m² and 609 surface detection points were used. Data collected from these points provided the scientific basis for the fire fighting work for the coalmine.
DETECTION METHODS

Detection theory

Radon is naturally existing within the coal rock and it is radioactive. Along with the rising temperature, the radon gas evolution will also increase. Based on this mechanism, the “Surface Radon Measuring Method” has been invented to detect the spontaneous fire source location. Detecting the radon variation from the surface, the fire source location, fire range and the fire behavior trend can be found through data processing. A research team from TUT used the large-scale experimental rig of coal spontaneous and radon measurement to study the variation behaviour of radon from coal sample, the radon gas evolution goes up when the temperature rises (Zhao and Wu, 2003; Xue, et al., 2008).

Figure 1 shows the structure of the experimental rig for coal spontaneous combustion and radon measurement. The device is located at the fire disaster laboratory of China-Australia Mine in TUT. It is used for radon research when coal spontaneous combustion happens.

![Figure 1 - The structure of the experiment](image1)

![Figure 2 - Rn concentration with coal temperature](image2)

There are three radioactive series widely existing in nature, they are the uranium series, thorium series and actinium series. They can exist in the rock, soil and coal as parent nuclide because of their longer radioactive half-life. Through a series of disintegration, uranium, thorium and actinium produce the radioactive nuclide radon and it can exist under normal pressure and temperature as aerosol. For its longest half-life period, the radon is taken as a carrier to research the earth dynamic phenomenon. The radon here mainly refers to the decay daughter of the uranium, Rn-222, and its own decay daughter.

Like the radioactive isotope radon, its decay daughter is a solid particle, so both radon gas and its decay daughter can reflect the shape and the variation of the parent nuclide. Radon daughter has a very strong adsorptive capability. It can stick at the surface of the container. Many options are available to collect radon and its daughter for measuring (Wu, et al., 2004; Xue, et al., 2010; Zhou, et al., 2003).

Detection method

There are many methods to measure radon. From timing measuring, there are differentiation and integration methods. According to the instrument being used, there are the α cup method, active carbon method, thermo-luminescence method and polonium 210 method. For this experiment, α cup with integration method was adopted (Xue, et al., 2003; Xue, et al., 2004).

Instrument and process

The high sensitivity, the CD-1α cup radon measuring instrument, was selected for the experiment. It is handy and easy to operate. It has an ion type detector and after the radon daughter is ionised, the Count Per Minute (CPM) will be displayed. Timing includes 1 min, 3 min, and 5 min, normally 3 min is chosen. The disadvantage of this device is that it is not shockproof. The matched α cup has about a 12x8 cm² detection area and it is made of highly adsorptive material for easier radon adsorption.
The detection cup was buried at the pre-selected spots with the spacing of 15 m×15 m, the spacing could be 20 m×20 m, 15 m×15 m or 10 m×10 m, depending on site conditions. The cup was put bottom up in the 40～50 cm pit with plastic cloth covering as show in Figure 4. Four hours later, the cup is taken out and immediately put into the instrument for measuring, set timing at 3 min and recorded the data. If the data showed anything unusual and the need for more measuring spots for proving, more could be added at anytime.

![Figure 3 - Field radon measuring instrument](image1)

![Figure 4 - Detection point and collection cup](image2)

Detection area arrangement

According to the field conditions, the Bulianta Mine fixed the boundary of the detection area, about 120 000 m², in which 609 detection points were arranged with the spacing of 15 ×15 m as shown in Figure 5.

![Figure 5 - The arrangement of spontaneous fire source detection at the gob area of N.12 405 working face at Bulianta Mine, Shendong](image3)

DETECTION RESULT AND ANALYSIS

The collected data from the field was then processed using the CDTH software package, Figures 6, 7, 8, and 9 respectively illustrate an actual data block diagram and its contour map and an unusual data block diagram and its contour map.

![Figure 6 - Actual data block diagram](image4)

![Figure 7 - Actual data contour map](image5)
From the above analysis, the plan view of the spontaneous combustion fire source location detection at the goaf area of N.12405 working face at Bulianta mine, Shendong was obtained and is shown in Figure 10.

**Figure 10 - Plan View of the spontaneous combustion fire source location detection at the goaf area of N.12 405 working face at Bulianta Mine, Shendong**

**Result analysis**

1. Two districts were found with unusual detected data; they were district A and district B with a total area of 2010 m\(^2\).
2. District A is a high temperature oxidising zone. It is located at the No.4 crosscut of the main return at the No.12 405 working face with 569 m\(^2\) area. District B is also a high temperature oxidizing zone. It is at the No.5 crosscut of the No.12 404 transport gateroad.
3. The extent, the location and the development trend of these two districts with unusual data were shown on the plan view of the spontaneous combustion fire source location detection at the goaf area of N.12 405 working face at Bulianta Mine, Shendong.

**APPLICATION IMPACT**

According to the detection, the boreholes were drilled at the corresponding position from the surface. The bottom temperature of the borehole was measured, it was 50~150°C. A gas sample from the borehole bottom was collected and subsequently analysed using a GC, which showed that the 10% of CO concentration along with the \( \text{C}_2\text{H}_6, \text{C}_2\text{H}_4 \) measured gas. It indicated that the location of the high temperature zone is corresponding to the reality of a fire.

After the fire source location was identified, the Bulianta Mine drilled boreholes to the high temperature zone and injected cement for extinguishment: First, a vertical borehole from the surface was drilled, then based on the unequal fire zone, boreholes were drilled evenly along the major axis and minor axis. Secondly, grout was injected to fire zone from boreholes. To avoid the fire zone spreading, drilling and grouting were also conducted along the edge of the fire zone. At the same time, the coal mine also
took measures from underground, like sealing, grouting and stopping leakage. Then samples were taken to analyse the gas taken from the fire zone (Figure 11), fifteen days later, it showed that the symbolising gas concentration, CO, C₂H₂, C₂H₄ and C₂H₆, were significantly lower. This indicated that the measure the mine taken had worked effectively.

According to the research result of the TUT team, the Bulianta Mine conducted drilling and grouting with clear goals, this not only improved the extinguishing effective, but also saved the material and manpower from blind drilling and grouting. The application of the radon measuring method was very successful with wonderful effect.

![Variation tendency chart of measured gas from the gateroad of the No.12403, Bulianta](image)

**Figure 11 - Variation tendency chart of measured gas from the gateroad of the No.12403, Bulianta**

**REFERENCES**


SAFE AUTOMATION: A PRACTICAL GUIDE TO UNDERSTAND AND MANAGE SENSOR RF EXPOSURE RISK

Andrew D. Strange and Jonathon C. Ralston

ABSTRACT: The increasing interest in remote and automation technology by the underground coal mining industry has resulted in the introduction of sensors and devices that emit electromagnetic radiation. The Australian Radiation Protection and Nuclear Safety Agency (ARPANSA) has defined clear maximum limits for exposure to electromagnetic radiation to ensure health and safety of people and the environment. However, it is often difficult to determine the actual radio frequency (RF) energy level a sensor radiates in a given operation context. Further, the ARPANSA regulation and associated Australian standards for electromagnetic radiation exposure can be complex to practically interpret. Unfortunately, this may leave the mining personnel in the position where they need to either solely rely on vendor device specifications or hope that the sensor is not presenting any radiation exposure risk. This paper provides an overview of the ARPANSA exposure regulations and sets out a practical approach to measure the radiation as per the Australian Standard. By focussing on an RF range from 3 kHz to 300 GHz, a number of commonly used active devices such as radars, wireless communication systems, and related RF imaging devices can be assessed. A simple method to allow basic in-house testing is described to allow end-users to make independent quantitative assessments of sensor radiation levels. The assessment method is demonstrated with a practical example using two ground penetrating radar systems as test cases.

INTRODUCTION

With the ongoing demand for automation and safety systems in underground coal mines, the rate of installation of devices and sensors that emit electromagnetic radiation will progressively increase. Sensors and devices in this domain include wireless communication (e.g. Wi-Fi), collision or proximity detection systems, radio frequency identification (RFID) and ground penetrating radar (GPR) systems. These types of devices function by transmitting electromagnetic energy from a transmitter to a receiver. The Australian Radiation Protection and Nuclear Safety Agency (ARPANSA) define the electromagnetic energy emissions limits for safe operation of these types of devices. ARPANSA has published restrictions with which these sensors and devices must comply (ARPANSA, 2002). One sensor that emits electromagnetic radiation and has the potential to enhance automation systems in the underground coal mining industry is GPR.

A GPR sensor, also called subsurface radar, is typically used to non-invasively determine information about the subsurface. The most common configuration of GPR involves an antenna module positioned in direct contact with the ground. When the antenna module is moved along a path, changes in the subsurface are shown on a display unit. During operation, the GPR system transmits extremely short pulses of electromagnetic energy into the ground. The electromagnetic energy is then reflected at interfaces beneath the ground and returns back towards the antenna module for processing and display. Previous research undertaken by CSIRO (Ralston and Hainsworth, 1999) and (Ralston, et al., 2001) has shown that GPR can be used to measure coal thickness in underground coal mining operations. Therefore, it is important to consider the safety of this technology and assess if the electromagnetic fields emitted by standard commercial GPR systems are within the limits defined by ARPANSA.

An overview of the ARPANSA regulations that apply to the typical GPR sensors is presented. A procedure to measure the electromagnetic radiation level to determine compliance is described. Experiments were conducted where the electromagnetic radiation levels of two commercial GPR systems were measured. These experiments and the results are presented.
ELECTROMAGNETIC FIELDS

Electromagnetic fields consist of two complementary fields - an electric field and a magnetic field. An electric field is created by the concentration of electric charge and a magnetic field is created by the motion of electric charge. Systems that utilise electromagnetic fields employ a transmitter antenna to radiate electromagnetic energy to a corresponding receiver antenna. For example, a broadcast TV system has a transmitter antenna which is usually located high on a mountain close to the TV studio. The transmitter antenna radiates the TV signals in the form of electromagnetic energy to the TV antennas in homes across the local region. The receiver antenna in this case is the TV antenna as it receives a portion of the electromagnetic energy transmitted by the transmitter antenna. In other cases such as airport radar systems, the transmitter and receiver antenna are the same physical infrastructure. For the radar case, the received signal is reflected back towards the antenna from a target such as an aeroplane.

Electromagnetic energy can be generated from a continuous or pulsed signal. A continuous signal is a signal that maintains full power. An example of a continuous signal is a TV broadcast signal which is continuously being transmitted. Conversely, a pulsed signal is one that consists of two distinct states that are repeated periodically. During an on state, a high level of power is being transmitted whereas during an off state, the transmitter power is typically very low. The rate at which the pulsed signal is repeated is called the pulse repetition frequency (PRF). Impulse GPR systems such as those evaluated as part of this investigation function as a pulse system.

OVERVIEW OF THE RADIATION PROTECTION STANDARD

The Radiation Protection Standard published by ARPANSA “sets limits for human exposure to radiofrequency (RF) fields in the frequency range 3 kHz to 300 GHz” (ARPANSA, 2002). Instructions and guidelines regarding the methods and instrumentation required to measure the levels of radiofrequency fields are outlined in Australian/New Zealand Standard AS/NZS 2772.2:2011 (Standards Australia Limited/Standards New Zealand, 2011). The guidelines described in these two documents may be used to determine if devices that emit electromagnetic radiation are safe during prolonged exposure.

The regulation published by ARPANSA provides mandatory limits of electromagnetic radiation exposure that contain basic restrictions which must not be exceeded to protect against adverse health effects. The physical parameters that must be measured to test compliance with the basic restrictions include current density, specific absorption rate, specific absorption and power flux density. These physical parameters, however, are often impractical to measure. Therefore, an alternative set of restrictions called reference levels are also provided in the regulations with parameters that are easier to measure. Essentially, the reference levels have been conservatively formulated such that compliance with the reference levels ensures compliance with the basic restrictions (ARPANSA, 2002).

Different exposure limits for two general groups are also included in the ARPANSA regulation. These two groups are the occupational group and the general public. The general public exposure group relates to the exposure of a member of the general public who would be unaware of their exposure to the electromagnetic field. The occupational group is defined as the exposure to a person being exposed to electromagnetic fields under controlled conditions as part of their work whilst on duty. The acceptable exposure limits for the occupational group are higher than for the general public. However, the occupational group limits are subject to certain risk management policies and are outlined in the standard (ARPANSA, 2002).

The restrictions provide two different limits for systems based on whether they radiate continuous or pulsed signals. These two limits are the time averaged and instantaneous limits. The restrictions imposed for the instantaneous fields are to prevent effects associated with high powered pulsed fields. The high powered short time pulsed fields are impractical to measure, therefore the time averaged approach addresses these impracticalities by imposing a six minute averaging time over which the radiated field must be measured. The device under test can be shown to satisfy compliance if, after the six minute time interval, the root-mean-square (RMS) signal does not exceed the time averaged restrictions.

The region surrounding a transmitting antenna can be separated into three zones based on the distance from the antenna: the reactive near-field; the radiating near-field; and the far-field. In the far-field, the
electric and magnetic fields are directly related and hence, compliance can be shown with a measurement of either the electric or magnetic fields being less than the reference levels. However, the relationship between the electric and magnetic fields is usually unknown in the reactive and radiating near-field zone. Therefore, both the electric and magnetic fields must be measured in this case to show compliance. Table 1 summarises the radiation zones and also indicates the distance from the radiating antenna for each zone. The value of $\lambda$ in Table 1 is the wavelength (in metres) of the electromagnetic field in free-space and is calculated as $\lambda = c/f$, where $c$ is the speed of light ($3 \times 10^8$ m/s) and $f$ is the frequency of the electromagnetic field (in Hertz). The value of $D$ in Table 1 is the largest dimension of the radiating antenna (in metres).

<table>
<thead>
<tr>
<th>Field Range Zone</th>
<th>Field to Measure</th>
<th>Min Distance</th>
<th>Max Distance</th>
</tr>
</thead>
<tbody>
<tr>
<td>Reactive Near-field</td>
<td>Electric and Magnetic</td>
<td>$0$</td>
<td>$\frac{\lambda}{2\pi}$</td>
</tr>
<tr>
<td>Radiating Near-field</td>
<td>Electric and Magnetic</td>
<td>$\frac{\lambda}{2\pi}$</td>
<td>Greater of $\frac{2D^2}{\lambda}$ or $\frac{\lambda}{2}$</td>
</tr>
<tr>
<td>Far-field</td>
<td>Electric or Magnetic</td>
<td>Greater of: $\frac{2D^2}{\lambda}$ or $\frac{\lambda}{2}$</td>
<td>No limit</td>
</tr>
</tbody>
</table>

The distance from the radiating antenna where the field strength should be measured to show compliance can be determined from how the device to be tested is used in typical operation. If the typical operation of a sensor or device is for a person to be located close to the transmitter antenna, then a near-field evaluation of the electric and magnetic field is warranted. However, if someone is typically located at a distance further than the near-field far-field crossover distance, then only the measurement of the far-field of the antenna is required. The field strengths are highest close to the transmitter antennas. Therefore, Table 1 is important because testing for compliance at a location that is very close to the transmitting antenna could be unnecessary if that is not the normal mode of operation (i.e. a person is not usually close to the transmitting antenna).

An extract of the reference levels for time averaged exposure to RMS electric and magnetic fields are shown in Table 2 (ARPANSA, 2002). A device can be shown to be compliant if the measured electric and magnetic fields are below these reference levels. The standard contains other frequency ranges not shown here as they are outside the range of the GPR sensors under test. Note that the frequency parameter in Table 2 is in the unit of MHz.

<table>
<thead>
<tr>
<th>Exposure Category</th>
<th>Frequency Range</th>
<th>Electric Field (V/m)</th>
<th>Magnetic Field (A/m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>General Public</td>
<td>10 MHz – 400 MHz</td>
<td>27.4</td>
<td>0.0729</td>
</tr>
<tr>
<td></td>
<td>400 MHz – 2 GHz</td>
<td>$1.37 \times f^{0.5}$</td>
<td>$0.00364 \times f^{0.5}$</td>
</tr>
<tr>
<td></td>
<td>2 GHz – 300 GHz</td>
<td>61.4</td>
<td>0.163</td>
</tr>
<tr>
<td>Occupational</td>
<td>10 MHz – 400 MHz</td>
<td>61.4</td>
<td>0.163</td>
</tr>
<tr>
<td></td>
<td>400 MHz – 2 GHz</td>
<td>$3.07 \times f^{0.5}$</td>
<td>$0.00814 \times f^{0.5}$</td>
</tr>
<tr>
<td></td>
<td>2 GHz – 300 GHz</td>
<td>137</td>
<td>0.364</td>
</tr>
</tbody>
</table>

**MEASUREMENT PROCEDURE**

There are several techniques that may be used to determine the level of electromagnetic field a device radiates. The two most common techniques are called the broadband method and the frequency selective method. To test compliance using the broadband method, a specialised unit called a broadband field meter is used to measure the electromagnetic field strength over a wide bandwidth and reports a value indicative of the maximum field strength. This method is suited for determining if the electromagnetic radiation observed at a single location exceeds ARPANSA limits. This technique does not, however, provide information about the field strength radiated by a specific device or sensor. An example of a broadband sensor is the NARDA NBM-550 (Narda Safety Test Solutions, 2011).

The frequency selective method provides the tester more flexibility over the broadband method. This technique can be used to measure the field strength over specific frequency ranges or to determine the...
field strengths of certain devices or sensors whose radiated field strength might be weaker than other fields radiated in different frequency bands. The instrumentation required for this method includes an antenna and a spectrum analyser. In the case of measuring the levels of low electromagnetic field strengths, a pre-amplifier in between the test antenna and spectrum analyser can also be used to boost the received signal. Typical antennas used for measuring the electric field strength include biconical or log periodic antennas as they have a wide operating frequency band with respect to other antenna types. The loop antenna (magnetic probe) is usually used to measure the magnetic field strength. Figure 1 shows a diagram representing the test configuration of a spectrum analyser and test receiver antenna and example transmitters of mobile phone and mobile phone tower.

![Diagram of test configuration to measure radiated field strength using the frequency selective method and examples of electromagnetic field transmitters](image)

In the configuration shown in Figure 1, the electromagnetic fields are radiated by transmitters such as mobile phones and mobile phone towers. Other examples include TV broadcast antennas, Wi-Fi enabled devices and radars. The radiated electromagnetic fields that intercept the test antenna are converted into an electrical voltage signal that is then displayed on a spectrum analyser as a frequency domain representation. The amplitude of the voltage signal displayed on the spectrum analyser is related to both the electromagnetic field strength intercepted by the test antenna and the characteristics of the test antenna.

Only a small portion of the electromagnetic field radiated by the transmitter antenna is captured by the receiver antenna. Therefore, to determine the actual field radiated by the transmitter, a correction must be applied to the signal level displayed on the spectrum analyser. This is in the form of an antenna correction factor called the antenna factor. Fundamentally, the antenna factor is the ratio of the incident electric or magnetic field strength to the output voltage of the antenna. The antenna factor is a function of frequency and antenna gain and is typically measured by the manufacturer as part of a calibration process. Therefore, the antenna factor data is required when calculating the actual radiated field strength and hence should be provided by the antenna manufacturer.

The equations to calculate the electric and magnetic field strengths are as follows:

\[
E = KV \\
H = KV
\]

where \( E \) is the electric field strength in Volts per metre (V/m), \( H \) is the magnetic field strength in Amperes per metre (A/m), \( K \) is the antenna calibration factor (1/m), and \( V \) is the voltage recorded by the receiver (spectrum analyser) in the units of Volts.

For the case when the receiver voltage and antenna factor data are measured using the decibel scale, the electric and magnetic field strengths are calculated as follows:

\[
E_d = K_d + V_d \\
H_d = K_d + V_d
\]

where the unit of \( E_d \) is dBμV/m, the unit of \( H_d \) is dBμA/m, the unit of \( K_d \) is dB/m, and the unit of \( V_d \) is dBμV. As the units of the electric and magnetic field strengths provided in the ARPANSA limits are V/m
and A/m respectively, the field strengths must be converted from dBµV/m and dBµA/m to V/m and A/m. This can be achieved using the following relationships:

\[
E = 10^{-6} \times 10^{(E_d/20)} \\
H = 10^{-6} \times 10^{(H_d/20)}
\]

(5) \hspace{1cm} (6)

It is important to ensure that the spectrum analyser is configured correctly to be certain the measurements are accurate. The spectrum analyser has four parameters that need to be considered when taking a measurement. These parameters are the resolution bandwidth (RBW), video bandwidth (VBW), detection mode and output units. The settings chosen for these parameters are dependent upon the source that generates the electromagnetic field.

The RBW is related to the ability of the spectrum analyser to resolve individual spectral components. If the RBW is set too small, multiple narrowband signals very close to each other in terms of frequency will not be resolved. If the RBW is set too large, the amplitude of the frequency component is artificially amplified due to more energy being sampled for each frequency component. For the case where the source of the electromagnetic field is a pulsed system such as the GPR systems (short time impulse transmitted at a constant repetition rate), the RBW must be set to a value that is less than 30 % of the PRF. If the RBW is set greater than this value, then the signal shown on the spectrum analyser is artificially greater which will result in a higher reading. When the RBW is set correctly, the data displayed on the spectrum analyser represents the time averaged frequency spectrum of the pulsed signal. Hence, the reference levels in Table 1 can be used to test compliance of pulsed systems with the ARPANSA regulations.

The VBW is related to how the data is displayed on the spectrum analyser. It is recommended that this value be greater than or equal to the RBW for pulsed signals. The detector mode in the spectrum analyser relates to how the voltage from the test antenna is sampled. Some examples include positive peak, negative peak and average. The spectrum analyser also has a trace mode function. This is used to determine when the individual values of the trace on the display are updated. Some examples include maximum hold, minimum hold, and average. When using the maximum hold mode, the maximum value observed at a given frequency by the spectrum analyser will be held in memory until a higher value is recorded. To ensure that the maximum observed field strengths are measured, the positive peak detector mode and maximum hold trace mode may be used. Finally, the output units of the spectrum analyser can be in the form of RMS power (dBm or Watts), RMS voltage (dBµV or Volts) or RMS current (dBmA or Amps).

It is the intention of this work to provide a general method to determine if a device that emits electromagnetic radiation is compliant with the ARPANSA regulation. The Australian Standard for measuring these devices (Standards Australia Limited/Standards New Zealand, 2011) states that the user should have the correct experience and technical skills to understand the area of electromagnetic emissions. Therefore, it is recommended that the reader familiarises themselves with the ARPANSA regulation and Australian Standard prior to conducting evaluations. If any doubt with regards to the measurement process arises or if preliminary testing indicates that a device may exceed the limits imposed by ARPANSA, it is recommended that formal testing and certification be sought from a registered testing facility.

EXPERIMENTS

The electric and magnetic field strengths radiated by some GPR systems were measured. The test equipment for the electric field measurements consisted of an Aaronia HyperLOG 4060 antenna and Rohde & Schwarz ZVL6 Vector Network Analyser (VNA) with the spectrum analyser option. For these experiments, the VNA was configured to spectrum analysis mode, hence is herein referred to as the spectrum analyser. The antenna type is log periodic and has a frequency range of 400 MHz to 6 GHz. The antenna factor data for the antenna was provided by the manufacturer. For the magnetic field measurements, an Aaronia PBS-H2 magnetic field probe was used as the test antenna. The log periodic antenna and magnetic field probe were mounted on plastic tripods. The test equipment is shown in Figure 2.
Figure 2 - Test equipment used to measure the electric and magnetic fields radiated from the GPR systems include the spectrum analyser (left), 400 MHz to 6 GHz log periodic antenna (top right), and 12 mm loop antenna (bottom right).

The GPR systems tested were the Geophysical Survey Systems Incorporated (GSSI) 900 MHz and 1500 MHz antenna modules powered by the SIR-3000 control unit. These are broadband systems and often have a bandwidth that approaches the centre frequency of the antennas. The broadband nature of GPR systems means that the spectrum of the radiated electromagnetic fields is broad rather than narrowband spikes. The antenna modules are shielded bi-static antennas, therefore, each antenna module contains a separate transmitter and receiver antenna enclosed within the antenna module. These antennas also have internal shields which focus the electromagnetic energy into the ground rather than behind the antenna. These systems can penetrate through coal up to approximate ranges of 1 m (900 MHz) and 0.5 m (1500 MHz). These systems are shown in Figure 3.

Figure 3 - Commercial GPR equipment under test. The equipment includes the (a) GSSI 900 MHz antenna (orange unit) and (b) GSSI 1500 MHz antenna (red unit). The display unit on the left side of each image is the GSSI SIR-3000 control unit.

The standard method of GPR operation involves the GPR antenna on the ground with the energy radiating directly into the ground. However, it is possible that an operator could lift the antenna whilst the system is still in operation. Therefore, three configurations were tested where the test antenna was within 50 mm of the GPR antenna. Based on Table 1, the minimum far-field distances for these GPR antennas are 160 mm for the 900 MHz unit and 100 mm for the 1500 MHz unit. In this case, the test antenna is within the near-field therefore both the electric and magnetic fields must be measured.

The first scenario was to test the standard operating practice with the operator to the side or behind the GPR antenna whilst it is in direct contact with the ground. The second scenario was to determine the field strengths for the case when the antenna was pointing directly at the operator. Therefore, the electric and magnetic fields were measured directly in front of the antenna, which is the point of the strongest field as it is the main beam. The third scenario was with the GPR antenna pointing away from the test receiver antenna to determine the effectiveness of the shielding within the GPR antennas. A background test was also conducted to determine the level of ambient electromagnetic noise with the GPR systems inactive. Note that the antennas have a specific polarisation, therefore, the test antenna was oriented to ensure that the maximum field strengths were observed during these experiments.
The test scenarios for both the log periodic antenna and magnetic field probe are shown in Figure 4 and Figure 5 for the 900 MHz GPR antenna.

![Image](image1.jpg)

(a)

![Image](image2.jpg)

(b)

![Image](image3.jpg)

(c)

Figure 4 - Configurations for the electric field measurements include test antenna (a) to the side of the GPR antenna; (b) in front of main beam; and (c) behind main beam. Note that the main beam of the GPR antenna is outwards through the white plastic base in (b).

![Image](image4.jpg)

(a)

![Image](image5.jpg)

(b)

![Image](image6.jpg)

(c)

Figure 5 - Configurations for the magnetic field measurements include magnetic probe (a) to the side of the GPR antenna; (b) in front of main beam; and (c) behind main beam. Note that the main beam of the GPR antenna is outwards through the white plastic base in (b).

For these experiments, the spectrum analyser was configured such that the trace mode was set to maximum hold and the detector mode was set to positive peak. These parameters were chosen to ensure that the maximum possible level of signal strength radiated from the GPR was measured to test the worst case scenario and hence confirm compliance with the ARPANSA limits. The test antennas
and probes were also moved around the antenna to ensure that the configuration shown in Figure 4 (b) and Figure 5 (b) indicate the maximum field strength values. When the maximum field strength position was found, the test antenna and probe were left in position for six minutes as required for the time averaged test. The RBW setting on the spectrum analyser was set to 30 kHz. The PRF of the GPR systems was 100 kHz so the RBW of 30 kHz was at the upper limit for satisfactory measurement.

The raw signal strength data measured by the spectrum analyser was the RMS voltage from the test antenna and was in the units of dBμV. The antenna factor data for the HyperLOG4060 antenna and PBS-H2 probe were added to the raw spectrum analyser data as per equations (3) and (4). These measurements were then converted into the units of V/m and A/m for direct comparison with the ARPANSA regulations. The final field strengths measured for the 900 MHz and 1500 MHz GSSI antennas are shown in Figure 6 (electric field) and Figure 7 (magnetic field) respectively.

![Figure 3 - Measured electric field strengths for the (a) 900 MHz and (b) 1500 MHz GPR antennas. The maximum electric field strengths from these GPR systems are well below the ARPANSA limits of 47 260 x 10^{-3} V/m (900 MHz antenna) and 56 490 x 10^{-3} V/m (1500 MHz antenna).](image)

![Figure 4 - Measured magnetic field strengths for the (a) 900 MHz and (b) 1500 MHz GPR antennas. The maximum magnetic field strengths measured from these GPR systems are well below the ARPANSA limit of 729 x 10^{-4} A/m.](image)

**DISCUSSION**

The frequency range for the electric field strength measurements of the 900 MHz and 1500 MHz antennas were 400 MHz to 2200 MHz and 400 MHz to 4000 MHz respectively as shown in Figure 6.
The measurements indicate that the electric field strength measured at the side and rear of the GPR antennas is very similar to the background ambient noise. This means that in normal operation with the GPR antenna in direct contact with the ground, the electric field radiated towards an operator by the GPR systems is negligible. However, the maximum electric field measured very close to and in front of the main beam of the GPR antenna is approximately 0.0013 V/m at 1190 MHz for the 900 MHz system and 0.0011 V/m at 1700 MHz for the 1500 MHz system. From Table 2, the maximum time averaged reference level electric field strength for the general public at these frequencies is calculated to be 47.26 V/m (1190 MHz) and 56.49 V/m (1700 MHz). This shows that the maximum electric field strengths measured in the main beam of these antennas is approximately 36 000 times lower than the reference level limit for the 900 MHz system and 43 000 for the 1500 MHz system.

The results in Figure 6 also show spikes in the electric field at certain frequencies. These spikes are due to far-field transmissions by external systems observed during the experiments. These specific frequency bands are as follows: 527 MHz is the analogue broadcast of the SBS television station (SBS, 2011); 825 MHz to 960 MHz is mobile phone communications systems (ACMA, 2011a); 1880 MHz to 1900 MHz is for cordless telecommunications devices (ACMA, 2011b); 1950 MHz is used by 3G mobile devices (ACMA, 2006); 2400 MHz is used by Wi-Fi enabled devices. A special note must be made regarding the large spike in Figure 6 (b) at 2400 MHz. This spike was due to a Wi-Fi access point located within several metres of the experiment. The peak electric field measurement for this frequency is not shown as the figure is zoomed in to focus on the broadband GPR signal. However, the peak electric field value measured was 0.03 V/m, which is 2000 times lower than the ARPANSA limit of 61.4 V/m at 2400 MHz.

The frequency range for the magnetic field strength measurements of both GPR systems was 20 MHz to 2200 MHz. These plots indicate that the frequency of the maximum magnetic field strength radiated by these GPR antennas is outside the bandwidth of the electric field. The general trend is the magnetic field decreases as the frequency increases. Note that these maximum magnetic field values were only observed when the magnetic probe was in direct contact with the base of antenna unit as shown in Figure 5 (b). The maximum measured magnetic field strength was 0.0004 A/m for the 900 MHz GPR antenna and 0.0002 A/m for the 1500 MHz GPR antenna, both at 20 MHz. The maximum magnetic field strength reference level at this frequency is 0.0729 A/m. Therefore, the measured magnetic field strengths are approximately 180 and 360 times lower than the reference levels.

**SUMMARY**

This paper provided an overview of the ARPANSA regulations for the exposure to electromagnetic radiation. A practical approach to measure electromagnetic radiation was presented based on the Australian Standard. The method was demonstrated with the measurement of two commercial GPR systems to determine if they are compliant with the ARPANSA regulations for safe prolonged exposure. In summary, the electric and magnetic fields measured from the GPR systems are well below the general public reference limits, which are the most conservative limits published by ARPANSA. Therefore, it is concluded that these devices are safe to use in terms of electromagnetic radiation.

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RISKGATE: PROMOTING AND REDEFINING BEST PRACTICE FOR RISK MANAGEMENT IN THE AUSTRALIAN COAL INDUSTRY

Philipp Kirsch1, Sarah Goater1, Jill Harris1, Darren Sprott2 and Jim Joy1

ABSTRACT: RISKGATE, an ACARP funded initiative, is an interactive online risk management system designed to assist in the analysis of priority unwanted events unique to the Australian coal mining industry. The system is innovative in that it is built upon a foundation of expert knowledge gathered through action research workshops, further supported by a substantive and diverse array of industry, academic and technological resources. In operation, RISKGATE offers an innovative tool to assist industry partners and regulators alike in the design, management and reporting of organisational and regulatory compliance requirements. In practice, RISKGATE provides a continuum for knowledge transfer and redefining best practice in risk identification, assessment and management in the coal industry. The RISKGATE prototype in its early phase of testing for priority of unwanted events of most relevance to coal mining is presented.

INTRODUCTION

RISKGATE is a web-based tool providing clear, up-to-date and practical checklists for controlling risks across 15 specific high priority unwanted events in Australian coal mining. Based on interactive ‘Bow-Tie’ methodology to assist in the implementation of safer operations, each RISKGATE topic and each bow-tie is centred on a specific unwanted or initiating event. The funneling of causal factors and consequences through this initiating event keeps the information concise, intuitive and targeted. Users can generate checklists that will deliver on-site managers and engineers quick and relevant access to best-practice controls for consideration within their own procedures and practices. These checklists are designed to assist with risk assessment, auditing, accident investigation, and training.

RISKGATE is funded by the Australian Coal Association Research Program (ACARP); managed and implemented by the University of Queensland; and each of the thousands of specific controls loaded into the RISKGATE system have been instigated and assessed by industry experts from Australia’s leading mining companies.

CONTEXT

The Australian coal mining industry is concentrated in Queensland and New South Wales. In Queensland, the industry is regulated by the Queensland Mines Inspectorate, a division of the Department of Employment, Economic Development and Innovation (DEEDI). The NSW Department of Primary Industries (NSWDPI) Mine Safety program provides the framework and direction to the mining industry in that state.

Queensland statistics on injuries and high potential incidents (Table 1) have been extracted from the latest data (DEEDI, 2011). It was not possible to pool Queensland and New South Wales data due to differences in the methods these agencies use to collect and report mining statistics. The number of recorded events and potential incidents (Table 1) demonstrates the timeliness and importance of developing the RISKGATE system. As Open Cut mining accounts for about 65% of Australian coal production (Scott, et al., 2010), this may provide a partial explanation for the different number of incidents between Open Cut and Underground operations in Table 1.

Risk assessment is a widely used process in the mining industry that involves the identification, evaluation, and estimation of the levels of risk involved in a given situation, their comparison against

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benchmarks or standards, and a determination of an acceptable level of risk (e.g. Iannacchione, et al., 2007; Joy, 2001, 2006; NSW DPI, 1997; Allanson, 2002; Ross, 2011).

**Table 1 - High potential incidents in the Queensland coal industry (2006-2011) organised by RISKGATE topic categories**

<table>
<thead>
<tr>
<th>Coal Mining Environment</th>
<th>RISKGATE Topics</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Strata</td>
</tr>
<tr>
<td>Open Cut</td>
<td>88</td>
</tr>
<tr>
<td>Underground</td>
<td>31</td>
</tr>
</tbody>
</table>

* *While HPI Data for Vehicle and Mobile Plant are pooled, some mobile plant HPIs may not be collisions; ** Electrical and Hydraulics/Compressed Air HPIs are pooled; Source: DEEDI 2011 “High Potential Incidents - Hazards Identified 1/7/2006 to 30/6/2011”.

The RISKGATE mission is to research, design, develop and operate an on-line resource that will focus the Australian coal industry on prioritising CONTROL MANAGEMENT as a way to achieve acceptable risk. As designed, RISKGATE does not specifically assess risk, but instead provides a decision support tool, resources and outputs, such as tailored checklists, that can assist users in their site-specific risk assessment tasks. RISKGATE will assist the coal mining industry in improving minimum standards for safety performance, efficiency, operating practice and training.

**RISKGATE conception and work plan 2011-2014**

In 2006, ACARP funded the development by the Minerals Industry Safety and Health Centre (MISHC) of two topic-specific risk management portals based on Incident Cause Analysis Method (ICAM) methodology: TYREgate (MISHC, 2011a; Kizil and Rasche, 2009) and ISOLgate (MISHC, 2011b; Kizil and Rasche, 2011). In late 2009, ACARP identified the need for a broader risk management system that could carry the coal mining industry to the next level in system-wide risk management and reduction of incidents (Kizil and Rasche, 2011).

In response, The University of Queensland’s Minerals Industry Safety and Health Centre (MISHC) developed a scope for such a system, RISKGATE, in consultation with select coal industry representatives (via workshops and individual interviews involving >25 industry leaders) (Joy, 2011). These industry participants initially identified 12 unwanted priority events for phased development to form the foundations of the current RISKGATE programme. This set of events (now called Topics) included fires, tyres, collisions, isolation, strata, hazardous substances, explosions, trips/slips/falls, manual handling, interface and displays, inrush and workplace hazards, such as dust, noise, and vibration.

After the first year of research and workshops, 2011, the RISKGATE programme is now targeting completion of 15 topic areas identified to be of highest importance to the Australia coal industry. Each topic is an unwanted event, not a hazard. The 2011 program tackled an ambitious target of five topic areas, comprising fire, strata-UG/ground-OC (now split into two), collisions, tyres and isolation. In 2012, the research and development program is forecast to expand to address an additional four new topic areas, followed in 2013-2014 by the remaining five topics (Table 2).

**RISKGATE 2011: Topic definitions**

Every RISKGATE topic is focused on coal industry activities (mining, processing, transport and storage) in both open cut and underground mine environments. The scope includes mine sites, lease areas, and mine infrastructure (e.g. mobile, fixed plant, field equipment, buildings, transport; including road and rail); and all aspects of the mine life cycle from design through to decommissioning. The topics uniformly recognise that ‘loss of control’ can result in personnel injury and/or fatality, equipment damage, production loss, reputation loss and environmental damage. However, the priority focus throughout RISKGATE is personnel safety.
FIRES relates to the unwanted or unexpected combination of a fuel source and an ignition source resulting in fire. This topic identifies ignition and fuel sources that potentially interact within different mine site locations (e.g. mobile plant, infrastructure, stockpiles). As a result, controls have been developed that take into account specific fuel sources, ignition sources and locations. Mitigating controls provide comprehensive information regarding the development of an emergency response plan for use within the open cut and underground environments.

TYRES provides information about the prevention of incidents and accidents associated with the use of tyres/rim assemblies throughout their lifecycle in the mining environment - including transport and storage, fitting (installation and removal), and operation. Experts from mining organisations, tyre Original Equipment Manufacturers (OEMs) and ancillary service providers have concentrated on developing a comprehensive and specific set of controls highlighting best practice in tyre management systems.

ISOLATION delivers information about the loss of control of specific relevant energies (including electrical, hydraulic, and pneumatic) due to failure to lock out and make safe (i.e., isolate) equipment throughout the period of its use (equipment’s lifecycle). Design processes cover the full life cycle management of the asset (design and procurement, identification of energy sources, assessment of risk and implementation of methodology to manage the risk). Operational processes include identification of an energy source, isolation, confirmation, dissipation and lock out of that source. Maintenance processes include inspection, testing, verification, shutdown and start-up.

STRATA-UG provides information about management and prevention of incidents and accidents due to loss of strata control in the underground mine environment. In this topic, loss of strata control refers to the longwall face, outbye roadways, roadway face, pillar system, caving, goaf edge in pillar extraction, shafts, and coal burst/bumps domains. Causes and controls are separated into design and operational elements.

GROUND-OC provides information about management and prevention of incidents and accidents due to ground instability that is outside expectation of the ground/strata control management system. Within the open cut mine environment, ground instability is considered in the highwall, endwall, low wall, truck dump, tailings dams and embankments, stockpiles, truck and shovel benches and box cuts domains. Causes and controls are separated into design and operational elements.
COLLISIONS relates to the unwanted or unexpected interaction between people, mobile and field equipment, or fixed plant that results in collision or rollover. The topic has a particular focus on behavioural based causes of these unwanted events and innovative design solutions that mitigate these events.

The MISHC-based RISKGATE project team has assembled individual topic panels (teams of industry experts) that meet separately to discuss and develop the expert content (industry knowledge) for each of these specific topics. These panels develop the optimal system content and wording through discussion and debate within the group. As a result, each of the thousands of specific causes, controls and consequences within RISKGATE have been identified, created, assessed and confirmed by industry experts from Australia’s leading mining companies prior to upload into the system.

2011 RISKGATE RESEARCH PROGRAMME AND PARTICIPANTS

RISKGATE’s research programme consists of workshops (focus group format) supported by analysis of resources (industry/regulatory guidelines, protocols and accident/incident reports). Workshops started in April 2011, to collect causal, consequence and control information for five of the original 12 priority topic areas: Strata-UG/Ground-OC, Tyres, Collisions, Isolation and Fires. As the programme developed, the original forecast of two to three workshops per topic (12-18 total) more than doubled to a total of 37 separate workshops (one to two days/workshop) for a total of 61 workshop days in 2011.

To date, industry experts (and the coal mining companies) have contributed a collective equivalent of 296 individual personnel days to the RISKGATE programme. These experts bring broad ranging experience and training in underground, open cut, coal and hard rock environments (Table 3). Additional workshops will be necessary in the first quarter of 2012 to complete the development of these initial five (now six) topics.

Table 3 - Workshop Programme*: Number of workshops per topic, attendance in personnel days, and mean workshop participant experience (and range)

<table>
<thead>
<tr>
<th>Topic</th>
<th>No of workshops</th>
<th>Workshop attendance (days)</th>
<th>Mean workshop participant experience (yrs)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Strata-UG</td>
<td>6.5**</td>
<td>41.5</td>
<td>18.18 (5-37)</td>
</tr>
<tr>
<td>Ground-OC</td>
<td>6.5**</td>
<td>41.5</td>
<td>21.86 (5-43)</td>
</tr>
<tr>
<td>Tyres</td>
<td>6</td>
<td>34</td>
<td>19.95 (3-45)</td>
</tr>
<tr>
<td>Collisions</td>
<td>5</td>
<td>22</td>
<td>20.50 (1-39)</td>
</tr>
<tr>
<td>Isolation</td>
<td>5</td>
<td>29</td>
<td>23.64 (14-36)</td>
</tr>
<tr>
<td>Fires</td>
<td>8</td>
<td>47</td>
<td>26.50 (9-48)</td>
</tr>
</tbody>
</table>

* Data are totals up to December 21, 2011. The workshop programme is ongoing with further activities planned for Q1 2012 to complete the initial set of six topics;
** At the first strata workshop it was decided to create two discrete topic areas (strata-UG; ground-OC) to recognise different terminology and geotechnical practices between the open cut and underground mining domains.

A separate panel of experts is assembled for each single topic area, constituting a wide range of risk management proficiencies with participants carefully selected or recommended to maximise that panel's knowledge base. Workshop participants represent a broad array of industry knowledge and professional expertise acquired across a spectrum of seven mining companies, five OEMs, three ancillary industry organisations, two universities, and one regulatory agency (Table 4).

In total, representatives from mining enterprises contributed the largest number of workshop days to the development of RISKGATE 2011 (212 industry days, 72%), followed by Tyre OEMs and other tyre service groups (21%) and DEEDI (7%). As a percentage of the total number of days (212), mining company participation was spread as follows: Adani (1%), Anglo American (23%), BMA/BHP (17%), Centennial (10%), Peabody (19%), Rio Tinto (11%) and Xstrata (19%). Joy (2011) provides mid-year company participation rates by individual topic panel.
Collaborative industry efforts provide the foundation upon which RISKGATE is built, with outcomes a reflection of how leading practitioners share and negotiate best practice. These outcomes are further strengthened by the academic rigour and integrity of delivery within a Group of Eight (Go8) university vehicle. This approach makes RISKGATE both innovative and novel when compared to previous research programs relating to risk management. Integration of cross-sectorial industry knowledge, further supported by a substantive and diverse array of industry, academic and technological resources, means RISKGATE can offer a continuum for knowledge transfer and redefining best practice in risk identification, assessment and management in the coal industry.

Table 4 - Workshop participants

<table>
<thead>
<tr>
<th>Mining enterprises</th>
<th>OEMs</th>
<th>Ancillary industry organisations</th>
<th>Research institutes</th>
<th>Regulatory bodies</th>
</tr>
</thead>
<tbody>
<tr>
<td>Anglo American BMA/BHP</td>
<td>Bridgestone</td>
<td>Otraco</td>
<td>The University of Queensland</td>
<td>Old Department of Employment, Economic Development and Innovation (DEEDI)</td>
</tr>
<tr>
<td>Centennial Coal</td>
<td>Good Year</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Peabody Energy</td>
<td>Marathon</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Rio Tinto</td>
<td>Michelin</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Xstrata</td>
<td>Titan</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Adani</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Action-research workshops

The RISKGATE workshop series signifies an action-research-like program (Dick 2002, 2010) in that three core phases underpin the facilitated agenda, while program delivery remains flexible to encourage maximal participant input to formulate the final product (Figure 1).

Figure 1 - Sequential RISKGATE workshops, knowledge acquisition process, core workshop objectives, and embedded RISKGATE team communication feedback loops

Workshop facilitators provide guidance by explaining key concepts (e.g. bow-tie analysis technique), maintaining a consistent approach to collecting and collating information, communicating project outcomes with and between topic area experts, and understanding and explaining the evolving software’s capabilities and restrictions. Topic leaders were contracted from various university centres, professional organisations and or regulatory bodies to attend workshops and guide the development of information in adherence with the bow-tie method and the parameters in which the topic was defined by each panel of experts.
Development of each topic progresses through several phases, starting with a topic-specific industry workshop to define that topic’s database terms and structure, and identify initiating events. Subsequent workshops undertake detailed bow-tie analysis for each of the identified initiating events, and conclude with analysis and feedback for quality control of the RISKGATE system to ensure that technical accuracy is upheld and industry user requirements are fulfilled. This process is refined after each individual workshop based on participant feedback, and will be further refined for the new topics planned in 2012.

All participants are continually encouraged to evaluate the strengths and weaknesses of the RISKGATE system design and content, and tailor outputs that will ‘add value’ to current industry practice. Thus the RISKGATE program and final product is constantly growing and evolving through workshop participant testing and product evaluation. This process improves RISKGATE’s validity and reliability.

**Data acquisition and Bow-Tie development**

RISKGATE’s knowledge content (data) is acquired from each workshop using the bow-tie approach (Joy, 2006). The Bow-Tie Analysis (BTA) is a constructive risk management tool that illustrates the relationship between causes or hazards, the initiating event, how this event could lead to negative consequences, and how controls could be used to prevent this occurring. Chevreau et al. (2006) document how the use of bow-ties contributes to organisational learning for safety.

Importantly, BTA allows for systematic examination of unwanted events, graphically represents the interaction between causal, consequence and control information, is a tool familiar to the target users and can be readily learned by a broader industry audience. This method is being researched theoretically and adopted broadly across industrial sectors (e.g. de Dianous and Fievez, 2006; Duijim, 2009; Ferdous, et al., 2011; Ferdous, et al., 2012), and in specific areas such as chemical engineering (Cockshott, 2005), sea ports and offshore terminals (Mokhatri, et al., 2011), ship building (Jacinto and Silva, 2010) and pharmaceuticals (Chevreau, et al., 2006).

**Bow-Tie elements**

There are typically between four and eight bow-ties within each RISKGATE topic with each bow-tie centred on a specific **initiating event** (Figure 2) The initiating event, or ‘knot’, of the bow-tie represents the point at which energy control is lost; with the primary causes and the unwanted consequences of the initiating event tabulated on either side of the knot. When graphically presented the tool conceptually represents a bow-tie; hence the name.

![Figure 2 - Bow-tie conceptual diagram](image)

A **cause** is any occurrence or reason that could lead to an event via the release of the hazard(s). Correspondingly, a **consequence** is any negative outcome that arises from an initiating event. This may commonly include damage to people, equipment and/or the environment, though there may be other important negative consequences. RISKGATE targets consequences that impact upon people,
either by direct or indirect means. Note, there can be any number of causes or consequences to any given initiating event.

Controls include any process, policy, device, practice, or other action that is intended to reduce the likelihood of an event cause or to ameliorate (reduce the magnitude) of an event or consequence. Causal factors are prevented from triggering the event through specific preventative controls. Should these preventive controls fail, the occurrence or severity of consequences can be further minimised through mitigating controls designed and implemented before the event occurs.

The RISKGATE BTA tool has sufficient flexibility to accommodate a growing depth of knowledge beyond forecast user requirements, as has been experienced during this project. This expanding range of knowledge can be illustrated in comparing early vs current RISKGATE software flow diagrams and the growing number of controls now input into the system.

In 2011, the RISKGATE project has expanded from five to six topics, the result of splitting Strata into Strata-UG (underground) and Ground-OC (open cut) domains. The ACTUAL number of data elements collected to date for one initiating event per topic is summarised in Table 5. Note, these estimates are conservative as many of the controls are further broken into multiple options or ‘sub-control’ data points.

### Table 5 - Summary of RISKGATE bow-tie element data (one initiating event per topic)

<table>
<thead>
<tr>
<th>Topic</th>
<th>No. Initiating Events</th>
<th>Causes</th>
<th>Preventative Controls</th>
<th>Consequences</th>
<th>Mitigating Controls</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tyres</td>
<td>4</td>
<td>40</td>
<td>118</td>
<td>2</td>
<td>11</td>
</tr>
<tr>
<td>Collisions</td>
<td>4</td>
<td>33</td>
<td>88</td>
<td>18</td>
<td>6</td>
</tr>
<tr>
<td>Strata</td>
<td>7</td>
<td>30</td>
<td>112</td>
<td>9</td>
<td>3</td>
</tr>
<tr>
<td>Ground</td>
<td>8</td>
<td>11</td>
<td>22</td>
<td>6</td>
<td>4</td>
</tr>
<tr>
<td>Fires</td>
<td>6</td>
<td>26</td>
<td>93</td>
<td>2</td>
<td>14</td>
</tr>
<tr>
<td>Isolation</td>
<td>9</td>
<td>67</td>
<td>168</td>
<td>20</td>
<td>4</td>
</tr>
</tbody>
</table>

Commensurate to the increase in project bow-tie elements and related data requirements, the complexity of the software and web page development has also increased considerably.

### Web-based prototype development and outputs

The RISKGATE system is being built to requirements identified through research, individual surveys and a series of coal industry workshops (RISKGATE Phase I) held in 2010 (Joy, 2011). This work identified the following set of requirements:

- Information must be accessible, timely, up-to-date, practical, useful and useable.
- Control information must include classification by Levels of Practice (expected through to cutting edge), Hierarchy of Control, and background information and resources that support the controls;
- User outputs need to be designed for application in risk assessment, auditing/monitoring, incident/accident analysis or investigation, and for training and education;
- RISKGATE inputs (system data) would be derived from incident reports, expert analysis (workshops), risk assessments, company protocols and technical literature;
- A feedback mechanism would be created to keep the system current and ready to incorporate new information from safety alerts, user contributions, legislative changes and ongoing workshops.

Proposed structural requirements and navigation pathways through the RISKGATE system altered dramatically during Phase I of the workshop series in response to mining company expert feedback that clarified the practical requirements from this system within the mining workplace.
This feedback required a shift in the knowledge capture process and software processing requirements from a relatively simple linear information delivery process to a more complex, robust and user-intuitive system. Additional search functions and feedback loops were incorporated with bow-ties and checklists capable of being tailored or personalised to user requirements (Figure 3).

The Bow-Tie Analysis method that was used to assimilate information specific to each Initiating Event is graphically represented as a series of panels to maximise screen real-estate (Figure 4). The proposed objective of the bow-tie selector screen was to identify control information to be included in a user-defined checklist tool that can be saved, edited and/or printed for evaluation relative to individual mine site conditions.

![Figure 3 - The evolution of the RISKGATE software requirements. (a) Early software depictions; (b) Current evolved software depictions](image)

![Figure 4 - RISKGATE bow-tie selector and checklist web page](image)

The RISKGATE bow-tie selection and checklist development process has now been further refined to reduce the number of steps to move through the web tool - by combining the education, training and checklist development capacity in one screen. It is possible to navigate through the bow-tie structure, hover over elements to seek summary bow-tie information, or select an information icon to obtain more details about individual bow-tie elements. By doing this, the bow-tie interface has become interactive and allows the user to view and explore content at their discretion and tailor the checklist output. When the user is ready, they can proceed to create a customised checklist output from this same screen.
Design considerations and website development

RISKGATE has been designed to appear as simple and uncluttered as possible to help users navigate what will be a complex model. Styling and fonts are sans-serif with clean lines to appear both friendly and scientific. The font Gotham was chosen for its humanistic and clean styling. On the site Verdana was chosen for the body font for the same reasons.

Careful consideration was given to colour choices within the site. A dark navy blue was chosen as the main corporate colour with a light blue chosen as the secondary corporate colour. The only other colours introduced into the site were amber to represent causes; and an orange-red to represent consequences. Initiating events were given the dark corporate blue and controls were given the light blue. This simple colour scheme helps to define the main concepts in use within the bow-tie and RISKGATE. Green was purposefully rejected as a colour for controls because it implies ‘go’ while controls themselves are about ‘stopping’ or ‘blocking’.

RISKGATE uses language that is easily understood, credible and appropriate to the industry, yet broad enough to accommodate a range of industry expertise. All of the data (initiating events, causes, controls, consequences) have been directly developed by industry experts on the topic panels. The visual flow of information is logical with descriptive guides imbedded within each page to assist user self-navigation, and clearly signpost where the user is relative to the four core stages of the RISKGATE checklist development process. Finally, RISKGATE promotes and enables self-directed user enquiry by offering multiple access points to support information.

Access to RISKGATE information and the process of enquiry supports the user to define a goal before beginning the search, and provides a clear process to derive the required output. Movement through the web-based tool is intuitive comprising a logical set of structured steps (e.g. the operator can insert keyword search terms, or select from pre-populated information that develops a deductive set of steps toward the user goals). The system caters for a moderate level computer-skilled operator to navigate, comprehend and use.

The modular structural development of the RISKGATE site allows for easier future growth of the system and for powerful implementation of new web technologies. Areas within RISKGATE are password protected to only allow access to select users. Content can be structured on pages to present different results based on the level of access a user has. For example, ‘admin’ users have extra powers within the navigation of the site than ‘authorised’ users.

Quality assurance and quality checks - review process (web and data)

A rigorous three-tiered approach to RISKGATE data entry and product delivery has been devised - comprising an Alpha and Beta testing program to assess user requirements for web portal functionality against project objectives (Alpha) and useability testing of web portal functionality and content delivery (Beta).

Alpha testing shall occur with a selected small group of stakeholders from ACARP companies. Beta testing will involve workshop participants (part a), and a select group of potential system users that are not knowledgeable in the background or design of RISKGATE but rather representative of typical target users (part b). After the preliminary Alpha and Beta testing is completed, RISKGATE will be made available on-line to the ACARP workshop experts for a period of 12 months to seek further feedback.

Challenges and proposed solutions

The pilot year of RISKGATE programme research and system development has identified a series of challenges:

- **Workshop attendance** is critical for generating an accurate and ‘complete’ set of data within each topic area, especially in the initial phase when the topic parameters are being defined. Continuity between workshops is also important so that the majority of time can be dedicated to project needs and project briefings are minimised;

- Like many new research projects, the programme scope was expanded/redefined through interaction with the industry experts. As a result, expanded project deliverables were expected within the same end of calendar year deadlines, leading to a considerable increase in the
number of workshops. Workshop participation did decline over time: potentially reflecting workshop fatigue and the approach of the summer holiday season;

- Commensurate with the reduced workshop sizes, many new participants have joined the program as existing workshop attendees spread the word and encouraged peer collaboration, and company management providing strong encouragement and support for attendance. This has meant the diversity of input into RISKGATE has increased over time, providing a continual cycle of workshop output critique through ‘fresh pairs of eyes’;

- Continuous updating will be critical to maintain the useability of the RISKGATE system. At this initial stage in the project, new information from industry guidelines, regulations, safety bulletins, is incorporated into the system on an ongoing basis. However, when the project team completes any one topic area, it will be critical to put processes in place to maintain currency of the RISKGATE content;

- At this stage, the RISKGATE team is proposing an annual conference where all topic panels would reconvene for parallel sessions (one to two days) to review and update the online information. In a pharmaceutical plant, the bow-tie method is used to facilitate and update organisation-wide learning for safety through adding specific learnings from local-level incident and accident analysis (Chevreau, et al., 2006). On a national scale, the same approach might be used to maintain currency of the RISKGATE system for the Australian coal industry;

- An internal RISKGATE programme review of the 2011-2012 workshop series (initial set of topics) will be undertaken after completion of the Alpha and Beta testing period. This review will identify unanticipated challenges, formulate strategic changes, and assess future milestones to complete the RISKGATE work program for the remaining topics. In 2012, the intensity of workshop programs is expected to be significantly reduced, with the workshop format and methods of delivery now established. Workshops will also be located in areas outside of Brisbane if strategically appropriate - for example, the OUTBURST topic panels may be convened in the Southern NSW coal region;

- Diversity of language must be managed in developing a system for implementation throughout the Australian coal industry. Terminology can be specific to New South Wales, or Queensland, and these differences need to be captured. Further, the coal industry employs people from a wide range of backgrounds (coal/hardrock/heavy industry, technical disciplines, training/education, country of origin, culture, first language) and these differences can inadvertently create miscommunication or misunderstanding about specific terms or phrases. The RISKGATE workshop facilitators have taken a holistic approach in attempting to capture as many synonyms as possible; and every topic panel is actively encouraged to recruit additional members from diverse backgrounds.

CONCLUSIONS

The redefined scope and direction of RISKGATE generated through action-research workshops, exceeds the original expectations proposed in the RISKGATE Phase II 2010 grant application. This has resulted in delivery of a more complex, robust, industry-driven control management system soon to be available to ACARP and its members. By making the system available on-line, RISKGATE provides a continuum for knowledge transfer within and between mining sectors and provides a single-point risk management tool that promotes integration of available control information across a broad range of mining environments and worker expertise. Consequently, RISKGATE offers innovative technology to help redefine best practice in risk identification, assessment and management in the coal industry.

The RISKGATE team are now seeking industry experts for the 2012 program in the topic areas of explosions (methane gas), explosives, trips/slips/falls, and manual handling. Workshop participation offers benefits to individuals and the companies they represent including opportunities for networking, reflection and sharing of lessons learned, keeping abreast of current and emerging control technologies, instigating a shift in existing safety culture, and elevating the accepted levels of minimum best-practice.

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REFERENCES


A QUANTITATIVE APPROACH TO ENGINEERING FIRE LIFE SAFETY IN MODERN UNDERGROUND COAL MINES

Frank Mendham¹, David Cliff² and Tim Horberry³

ABSTRACT: Emerging from a history of ‘blanket approach’ prescriptive fire protection design, underground coal mining is rapidly embracing fire life safety analysis techniques that have been successfully used in performance-based fire engineering in the built environment.

In Australia, the leading practice fire engineering approach is to apply the International Fire Engineering Guidelines (ABCB 2005) and its methods as a design assessment framework. This approach has recently been used to quantify the performance of mine fire detection and therefore control of fire spread, paving the way for improvements in mine fire intervention and mine worker escape.

This paper presents a method of early fire detection using closed circuit television cameras and video analysis software associated with fixed plant fires leading to increased available safe evacuation time compared with contemporary point type fire detectors and gas monitoring sensors.

Successful pilot tests of the fire detection technology have been carried out in simulated mine conditions. A quantified and scientifically informed risk-based approach, offering improvements in mine fire rescue intervention and evacuation methodologies was achieved.

INTRODUCTION

The problem of mine fires is not a new problem for mine safety practitioners, with most research to date addressing the natural phenomenon of spontaneous combustion of coal. Similarly, a significant amount of research has previously been carried out involving potentially flammable gases released during mining operations. The subject research examines the lesser-understood problem of fires associated with fixed plant.

The current approach to fire detection in underground coal mines involves a reliance on gas analysis systems, rather than fire detection systems. The gas analysis systems are used to detect the concentration of carbon monoxide in air, which is not considered in the subject research to be appropriate for the detection of smoke in relation to fixed plant fires.

Carbon monoxide sensors, especially those distributed at various points within the mine workings, have a tendency to drift and become inaccurate over time. The more traditional tube bundle gas analysis approach, involving the remote collection of samples through tubes, inherently involves significant transport delays between the time of collecting the sample air and the time required to analyse the sample.

A recent comparative study by the National Institute for Occupational Health and Safety (NIOSH) (Edwards, 2002) was carried out in the NIOSH test mine. The tests evaluated the effectiveness of fire detectors and gas analysis sensors and showed that CO sensors were not as effective as smoke detectors for early fire detection of smouldering belt particulates.

Experimental results showed the clear advantage of smoke sensors over CO sensors for early mine fire detection. It was shown that smoke sensors could detect smouldering belt combustion smoke particulates long before alarm levels of 5 ppm CO were detected (Edwards, 2002).

Early fire detection is considered a critical aspect of mine fire life safety and asset loss control due to the potentially onerous evacuation and intervention considerations associated with the operational mining

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environment. The subject research assesses an emerging technology of early fire detection referred to as Video Based Fire Detection (VBFD). This technology is used to detect the early onset of fixed plant fires and is considered to be an efficient solution for early fire detection in underground coalmines.

Fixed plant equipment, where VBFD might be used in mining, includes: conveyor systems; refuelling stations; and electrical installations, however it is not limited to these examples. This improved fire detection technology applies Closed Circuit Television (CCTV) and video analysis algorithms to the video images to automatically recognise early fire development and subsequently initiate fire intervention.

For the purpose of quantifying the effectiveness of this technology, the International Fire Engineering Guidelines IFEG (ABCB 2005) is considered a suitably robust assessment framework.

VOLUME DETECTION - A NEW APPROACH FOR DETECTING MINE FIRES

Point type fire detection

Based on recent observations in underground coal mines and discussions with miners, it is reported that coal miners rely on point type gas analysis systems, such as remotely located carbon monoxide sensors, as their primary means of fixed plant fire detection.

Gas sensors are intended for the measurement of diffused carbon monoxide. The source of the carbon monoxide is typically that produced from the spontaneous combustion of coal or belt material. These types of gas analysis systems are not designed, intended, suitable or certified for the detection of combustion products associated with fixed plant fires.

The Australian Standards approved point type smoke detector (Figure 1) is certified under a system that is similar to many international approval systems. It is designed, tested and approved for the purpose of early fire detection, but intended for use in commercial facilities and not mines. It is the only current but impractical alternative to point type gas analysis systems as used in underground coalmines.

Figure 1 - Typical point type smoke detector

The point type fire detection system is configured using various types of certified fire protection point detectors, such as smoke, heat or gas detectors, or a combination of these. Point type fire detection may also describe Multi-point Aspirated Smoke Detection (MASD), where gas or smoke is remotely drawn through a single point or multiple entry points in a network of pipes, then centrally sampled. This approach is similar to tube bundle gas analysis systems used in the majority of Australian coalmines, however even with multiple detection sampling points in each sampling pipe, it is still considered a point type detection system.

The very design nature of point type detection incorporates its inherent limitation, which results from gaps in detection capability between detection points. Realistically, a detection point may not be located in the vicinity of a fire, so a delay in detection must occur.

Smoke, heat or gas is required to migrate from the fire source to the nearest point type fire detector before a fire alarm can be generated. A transport delay of smoke, gas or heat to detectors reduces the Available Safe Evacuation Time (ASET) before the onset of potentially untenable conditions. Under growing fire conditions, increasing levels of fire suppression capacity is subsequently required to control the advancing fire spread.
Inefficient fire detection not only reduces ASET for mine worker escape, but mine fire rescue and intervention capacity is reduced due to increasing fire spread.

A further major roadblock to early fire detection in mining is the physical contamination of point type detection devices from mining pollutants, whether it is individual detectors or aspirated systems with remote sensing heads. For example, point type smoke detectors using either the ionisation detection technique or the photoelectric detection technique, are readily contaminated by coal dust, which leads to false alarms and subsequent alarm complacency by occupants and mine controllers.

**Volume type fire detection**

Volume type fire detection involves the use of a form of detection, where the total space, or at least a substantial part of it, is monitored for early fire growth.

The most important aspect of this approach is that it excludes the need for smoke, heat or gas to physically come in contact with the sensor elements, unlike typical point detectors. Volume type detection is considered to be a potential solution to the problem of gas sensor drift and fire detection delays.

**VBFD - AN EMERGING VOLUME TYPE FIRE DETECTION TECHNOLOGY**

VBFD is an emerging volume type smoke and flame detection technology that utilizes CCTV cameras to capture real time video data for analysis. This detection technology offers considerably faster response times than other forms of fire detection currently used in underground mines.

VBFD cameras are enclosed within explosion protected (EX Rated) housings with no direct contact between the camera lenses and the mine environment. This makes VBFD more resistant to contamination from mine pollutants than point type smoke detectors. VBFD is seen as a solution to the problem of air pollutant contamination and subsequent unreliable early fire detection in underground mining and similar environments. Figure 2 shows a VBFD image taken in an experimental mine environment. Note the outline of the detected smoke in the image, as generated by the analytics of the VBFD system.

![Figure 2 - Video detection of coal fire - experimental pilot tests, Wollongong 2010](image)

Therefore, the proof of concept trial undertaken in Wollongong has shown ‘volume type’ detection has been achieved using VBFD.

CCTV that is configured as VBFD is the proposed means of achieving volume type fire detection in mining (Refer to Figure 3).
On-site observations indicate that mines currently having major fixed plant items monitored by CCTV systems, such as conveyor belt transfer points, would readily and cost effectively be able to incorporate volume type fire detection into their mine fire protection systems.

The existing cameras could be utilised in illuminated areas at a relatively low cost to operate as both VBFD and for the original intent, being surveillance and monitoring of belt blockages.

**IFEG - A FRAMEWORK FOR ANALYSING MINE FIRE SAFETY**

The visual early warning sign of fire, in relation to conveyor systems in coal mines, is the production of smoke as a result of overheating of conveyor components, such as pulleys and bearings. This event is generally followed by increased production of carbon monoxide as a result of the oxidation of coal in the vicinity of the overheated parts, and subsequently the burning of the fixed plant components themselves.

In addition to belt systems in underground mines, other potential fixed plant fire risks involve refuelling bays and equipment where electrical arcing and overheating may occur. Early detection of combustion products is a significant factor therefore in minimising loss through early fire intervention by achieving early fire suppression and evacuation where applicable.

The current internationally accepted framework for fire engineering, which provides a guideline to the assessment approach but does not specifically stipulate the methodology, is the International Fire Engineering Guidelines (ABCB 2005). This document is referred to within the fire engineering profession as the ‘IFEG’.

The New South Wales Guideline for the prevention, early detection and suppression of fires in coal mines MDG1032 (NSW-DP1 2010) is a useful document to be used in conjunction with the IFEG. MDG1032 provides context to the assessment, because it suggests mine specific guidance for the establishment of a risk-based approach to early detection and suppression of fires in coalmines.

It is understood that the IFEG methods were not initially intended for mining applications and that the analysis of mine fire safety has not previously been carried out using the IFEG format. The unique and challenging opportunity was to apply the established systematic fire risk analysis approach of the IFEG in the context of underground mining.

The scope of the IFEG framework is as follows: “These Guidelines have been developed for use in the fire engineering design and approval of buildings. However, the concepts and principles may also be of assistance in the fire engineering design and approval of other structures such as ships and tunnels, which comprise of enclosed spaces.” (ABCB 2005).

**The IFEG framework**

The IFEG framework is divided into ‘Sub Systems’ within the fire engineering process. A pictorial description of the IFEG Sub Systems (ABCB 2005) is provided in Figure 4.
Figure 4 - IFEG fire engineering assessment framework (ABCB 2005)

Related mine fire engineering analysis

Further motivation for applying the IFEG in a coal mining context was derived from previous studies of mine evacuation under fire emergency conditions (Gao, et al., 2008).

Scoping studies have also been achieved into the application of performance-based fire engineering design in relation to spontaneous combustion fires (Deng, et al., 2008). These previous works are examples of a growing interest in the application of a defined fire-engineering framework to mining.

This area of interest has been assessed previously by Gillies and Wu (2004), but this work did not apply the systematic IFEG approach. This work was part of an Australian Coal Association Research Program (ACARP) research grant (Refer Figure 5).

Figure 5 - Mine fire smoke movement (Gillies, 2004)

Gillies and Wu’s work involved the computer analysis of a range of mine fires to understand factors, such as rescue responses, preplanning of escape scenarios, and general interaction with mines rescue organisations. The research utilised the Polish fire simulation software ‘Ventgraph’, in relation to smoke movement under ventilation conditions.

APPLICATION OF THE IFEG

IFEG sub-system A - Fire initiation and development and control

The development of a design fire is a critical first step in analysing the fire development within the mine. A design fire describes graphically, the characterisation of a fire in terms of its growth and its decay, shown as Heat Release Rate (HRR) in Kilowatts over time.
Analysis of the design fire and knowledge of its characteristics is required for all of the Sub-System stages of the IFEG assessment that follow Sub-system A, as described in Figure 4.

The purpose of very early fire detection is to detect the fire in its earliest growth stages, which may or may not exhibit a flaming fire. This is typically the case in fixed plant fires involving overheated bearing houses, hot idler pulleys and similar components.

Figure 6 describes a design fire typical of smouldering fuel that never reaches a flaming stage. The heat release rate reduces over time as the fuel converts to char or the applied heat supporting the low level combustion is removed. This design fire curve represents the experimental heating of powdered coal in the pilot tests in this research. The pilot test was intended to simulate smouldering coal on the surface of an overheated bearing housing.

Compare this design fire curve with Figure 7, which shows a design fire that increases rapidly in heat release rate following flaming combustion, flashes over and then decays. This type of fire is of great interest to early fire detection, as it typically produces very little carbon monoxide and therefore may not be detected by carbon monoxide gas sensors.

Pilot tests carried out during 2010, as shown in Figure 2, simulated a portion of an overheating bearing housing surface covered in 15 mm of coal dust.

For each of the multiple tests carried out, the heated coal dust generated visible smoke within four minutes when heated to approximately 400 °C.

A low energy design fire, having smouldering characteristics for a considerable time period, resulted in each pilot test and graphically represented, would be typical of that shown in Figure 6.
**IFEG sub-system B - Smoke development spread and control**

An iterative process that quantifies the energy of the developing design fire for each increment of its growth is applied beyond early smouldering. This analysis is carried out through stages of growth to eventual flashover and subsequent smoke spread outside the enclosure of origin. Stages of fire growth beyond smouldering were not addressed in this study, as the focus of the study involves the analysis of early fire detection.

The management of smoke in a mine is assisted using Computational Fluid Dynamics (CFD) to estimate smoke spread scenarios under operation or non-operation of smoke barriers and whether controlled leakage through barriers is manageable. CFD is particularly useful for analysing smoke development and spread under various ventilation scenarios and has been applied in this research.

**IFEG sub-system C - Fire spread and impact and control**

Quantitative analysis methods, including Event Tree Analysis (ETA) validated using Monte Carlo probability simulation techniques, are applied to estimate and represent the probabilistic outcomes relating to fire spread. A range of data from previous fire science research and mine fire statistics, characterises the methods applied.

Determining the fire load density of the combustibles within the location in which the fixed plant fire has developed and spread is critical for estimating the likely severity of the fire, therefore a location specific knowledge of the environment is critical.

Understanding the impact of fire on structural elements at the affected location within the mine is a significant factor in managing mine roof collapse and therefore maintaining emergency egress. The onset of untenable conditions, in terms of heat, smoke, toxic vapor and potential explosion is equally a key factor for achieving mine worker escape or access to refuge.

**IFEG sub-system D - Fire detection, warning and suppression**

IFEG Sub-system D deals with fire detection and suppression. Whilst fire suppression analysis is not dealt with in this research, the IFEG provides a robust basis for the analysis of water-based fire suppression systems.

Interestingly, the IFEG does not offer a specific methodology for the volume type fire detection approach outlined earlier in this paper, primarily due to the recent evolution of VBFD and the lack of available test protocols.

The subject research addresses the application of VBFD in mining by comparing it with existing fire detection methods, as well as with currently applied gas analysis techniques. The measures of success have been based on:

- Does video based fire detection VBFD provide reliable early detection in the mining environment? (Addressed under sub-section D of the IFEG)
- Is it possible to significantly increase ASET using VBFD? (Addressed under sub-section D of the IFEG)
- Is it possible to use VBFD to assist in Mines Rescue intervention? (Addressed under sub-section E of the IFEG)?

Testing the reliability of VBFD to operate in the mining environment is ongoing. Results from pilot tests of VBFD under simulated mine conditions have achieved a repeatable response to detect low levels of smoke from incipient burning of coal at an early stage. Reliable early detection using VBFD appears to be present under simulated conditions.

The benefit of VBFD over current forms of fire detection is that it uses a volume detection approach, which is considered more effective than a point detection approach used by smoke detectors and similar fire detectors. Additionally, VBFD sensitivity is not subject to drift or chemical cross interference, as are current gas analysis sensing techniques.
“At a battery-charging station, several hundred ppm of H\textsubscript{2} produced a false indicated CO alarm value because of chemical cross interference in the CO sensor’s chemical cell. Without the smoke sensor signal, the battery-charging activity would be considered a fire.” (Edwards, 2002)

**IFEG sub-system E - Occupant evacuation and control**

Occupant evacuation analysis estimates the time taken for the events that make up the mine evacuation. It does this to determine the time required for mine occupants to reach a place of safety, whether it is a mine refuge or a location outside the mine. This is referred to as Required Safe Evacuation Time (RSET).

Improved early fire detection capability assists in increasing the available safe evacuation time ASET with the aim of ensuring ASET > RSET.

Methods referenced by the IFEG analyse cue periods, response periods, delay periods and movement periods.

Phases of evacuation include the detection phase, pre-movement phase, movement phase and evacuation phase, which combined contribute to RSET.

**IFEG sub-system F - Fire services (mines rescue) intervention**

IFEG Sub-system F provides a framework for the analysis of mines rescue and fire brigade intervention activities.

It addresses the arrival/establishment of mines rescue on site, investigation, set-up, search and rescue, fire attack, fire control and fire extinguishment.

An important aspect of the use of VBFD in mining is that it provides almost immediate fire detection. More importantly for mines rescue activities, it also provides current status of the occupants within the mine, or at least the most recently recorded status, perhaps prior to a significant event, which may cause the loss of cameras.

**CONCLUSIONS**

The significance of this research is apparent from on-site interviews at mines and actual mine observations carried out. Underground coalmines, in relation to the sites visited during the study, do not have adequate fixed plant fire detection and therefore a significant risk to mine occupants and development assets is assumed. The opportunity to improve fire life safety and asset loss control is considered significant using a performance-based solution. The performance-based fire engineering approach is not a ‘blanket style’ approach, it is specific and targets achieving a quantifiable level of fire risk management.

The most significant foreseen benefits of this research is that it equally and substantially builds upon the body of knowledge of both fire engineering and mining engineering. It does this by using the IFEG as a tool for managing and guiding mining fire risk analysis and the analysis of trial solutions, such as VBFD.

In more detail, four tentative conclusions are drawn:

- In relation to fire engineering, the research addresses one of the most onerous fire detection situations likely to be encountered. Based on the proof of concept trials undertaken to date, VBFD may be a solution for effective fire detection in underground mines.

- In relation to mining engineering, the IFEG tool specifically addresses mine occupant fire life safety and asset loss control in relation to plant and equipment fires - an area scarcely addressed in previous research. VBFD offers the ability to improve the ASET within a mine under fire conditions, so offers the ability for mines rescue to take earlier action and more informed action during intervention procedures.

- Neither fire engineering research applied to mining, or mining engineering research addressing early fire detection of fixed plant mining systems, has been previously robustly researched. This approach to fire engineering in mines is new, as is the application of VBFD.
• Further research is required involving IFEG assessment of fire scenarios in operating mines. Mines that currently have CCTV cameras fitted should consider carrying out relatively low cost, but high safety gain modifications by enhancing their surveillance and monitoring systems to detect fires.

In terms of future research in the area this ongoing study therefore focuses on a fire detection solution with significantly greater performance under aggressive environmental conditions, which are typically encountered in an underground coalmine.

The next stage of this research is to carry out a range of experiments at SIMTARS in Queensland under simulated mine conditions. Australia does not currently have a suitable test mine location to allow VBFD to be tested under more realistic fire conditions.

These experiments will simulate mine environment conditions including air pollutants, varying light levels and varying air velocity.

Comparative assessment against current gas analysis methods will be undertaken to quantify the performance of VBFD against gas analysis.

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OPPORTUNITY FOR RE-ENTRY INTO A COAL MINE IMMEDIATELY FOLLOWING AN EXPLOSION

Darren Brady¹ and David Cliff²

ABSTRACT: Following an explosion in a coal mine commentators within and outside the underground coal mining industry often make comment that the best time to enter the mine is immediately following the initial explosion. These comments often lead to the questioning of the action or lack of; if mines rescue services do not re-enter the mine.

The concept that there is a “window of opportunity” to enter a mine immediately following an explosion is based largely on the assumption that the fuel (mainly methane seam gas) required for an explosion has been consumed by the initial explosion and will take time to build up again to an explosive level.

This paper examines the merit of deploying personnel into an underground coal mine following an explosion.

INTRODUCTION

When persons remain unaccounted for underground following an explosion in a coal mine, there is a sense of urgency for rescuers to be deployed. Sending rescue teams underground must always take the safety of those deploying underground into consideration. McAteer, et al. (2011) made the following comment “In the United States alone, the history of mining and rescue efforts is filled with examples of rescuers being overcome by toxic gases or killed as a result of a second or third explosion”

A key consideration when sending in rescue teams is whether there is likely to be another explosion. For an explosion to occur an explosive gas mixture must come into contact with an ignition source. Prior to sophisticated mine gas monitoring systems being available to establish the status of the underground atmosphere, decisions were often based on the assumption that all of the methane was consumed in the initial explosion. If this was the case, a flammable gas mix would not be present, and a “window of opportunity” could exist until methane levels built up again, during which time mines rescue teams could enter the mine without risk of a second explosion. This assumption that no fuel existed justified deployment without the need to collect and evaluate data on the status of the underground environment. It was just assumed that this “window” existed.

The National Institute for Occupational Safety and Health’s (NIOSH) Pittsburgh Research Laboratory undertook a project whereby they interviewed recognised experts in mine emergency response. Vaught et al. (2004) quoted one of the interviewees’ on the perceived window of opportunity. “We Bureau people [operated] on the theory that a majority of the methane in the place was burned out by the explosion, and I think that’s another thing Scotia taught us—that doesn’t necessarily happen. There’s too many variables to even consider that I guess, but I think that was an assumption made by an awful lot of people. That depending upon the weight of liberation of that mine, you have a considerable amount of time to do some things before you had an explosive mixture reoccur. Well, that’s not true.”

The Australian coal mining industry has adopted a risk management approach to ensure that the risk that those in the industry are exposed to, is at an acceptable level. Mines rescue activities are not exempt from risk management processes in the conduct of their activities. Prior to deploying personnel underground following an explosion Australian mines rescue agencies will sample and evaluate the underground atmosphere and make decisions on re-entry based on the likelihood of secondary explosions. It is essential for the safety of the rescue teams that a conservative approach is taken towards the evaluation of secondary explosions. Evaluation of data may indicate that the risk to mines rescue teams if deployed underground would be unacceptable and as a result teams not be deployed. This understandably can be a source of frustration to all involved and where there is a time period

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between explosions, often leads to public questioning of the decision not to deploy rescue teams when a “window of opportunity” determined in hindsight existed.

The determination of whether a “window of opportunity” exists for re-entry into the mine can only be done with the data available to predict what could possibly occur.

**REQUIREMENTS FOR AN EXPLOSION**

For an explosion to occur a flammable gas mix must come into contact with an ignition source. Following an explosion there is every chance that an ignition source is present, either the ignition source of the initial explosion or a fire started by the explosion. A large volume of methane is not required for an explosion as it can simply be the catalyst for a coal dust explosion as is thought to be the case at Upper Big Branch Mine, 2010 (McAteer, et al., 2011).

As already mentioned it is often assumed that an explosion will burn all of the available methane making entry to the mine safe immediately after an explosion. However this is not always the case and consideration must be given to the time it will take for rescue teams to enter the mine, carry out their tasks and exit the mine and the possibility of an explosion during this whole period.

**Fuel sources**

It is possible flammable mixes of methane can build up very quickly after an explosion. If the build up is in an area (not necessarily the same area as the initial explosion) where an ignition source exists a subsequent explosion can occur:

- The fuel for the initial explosion may have come from a non homogeneous mix of methane that contained fuel rich concentrations of methane that due to the “disturbance” of the explosion results in formation of an explosive atmosphere;
- The initial explosion damages seals segregating fuel rich atmospheres (high methane) that are released into the workings forming explosive gas mixes;
- Ventilation control devices such as stoppings and overcasts are destroyed or compromised by the initial explosion resulting in no ventilation in some areas and build up of methane;
- Methane drainage lines could be damaged allowing the flow of methane into workings;
- If the initial explosion was fuel rich then percentage levels of carbon monoxide and hydrogen may be generated. Both of these gases are flammable and under these circumstances can be generated in quantities great enough that a flammable gas mixture can exist even when methane is less than 5%;
- Combination of any or all of the above.

**Example 1**

If an initial explosion damages ventilation control devices compromising the ventilation to a 4 m x 5 m roadway 100 m long and damages a gas drainage pipe from which 60 l/s of methane flows, enough methane is generated for the whole roadway to be explosive in 29 minutes. Obviously with the methane coming from a point source a smaller explosive gas mix could be formed much faster and some means of mixing would be required for the whole roadway to be explosive.

**Example 2**

If rib emissions from a 5 m x 4 m roadway 2 km long total 125 l/s of methane then in 280 min enough methane is generated to make the whole roadway explosive if there is no ventilation.

**Example 3**

In a ventilation flow of 60 m$^3$/s the required flow rate of methane to generate an explosive gas mix is 3.2 m$^3$/s.
Volume of methane

The volume of methane estimated to have been involved in the 1979 Appin Colliery explosion that resulted in the deaths of 14 workers was much less than 400 m$^3$ and probably less than 150 m$^3$ based upon the maximum volume of the mine atmosphere that could have exploded.

The volume of methane involved in the initial 1994 Moura Number 2 explosion was estimated to be less than 70 m$^3$, based upon the effects on the surviving miners (Stephan, et al., 1994).

An explosion in 1995 at the Endeavour Colliery in New South Wales was determined to be as a result of approximately 6 m$^3$ of methane being mixed with air to about 6-7% composition. This would indicate that the total volume of the flammable atmosphere that was ignited was only about 100 m$^3$ (Anderson, et al., 1997).

The amount of methane estimated as being involved in the initial explosion in 2000 at the Willow Creek Mine in Utah, United States of America was reported to be as little as 1.5m$^3$ (McKinney, et al., 2001).

In 2001 the initial explosion at the Jim Walter No. 5 Mine was determined to involve in the order of 3 m$^3$ of methane (Mckinney, et al., 2002).

These examples show that the amount of methane required for an explosion with devastating consequences can be minimal and doesn’t necessarily take long to accumulate.

REQUIREMENTS FOR RE-ENTRY

McAteer, et al., (2011) identified the need for a scientific, numbers-based approach to mines rescue deployment stating in their report on the Upper Big Branch Explosion for the Governor “Life and death decisions - whether to send rescuers in or pull them back - are questioned, discussed and second-guessed, allowing the emotion of the moment to infringe upon the detached discipline and scientific approach that forms the basis of mine rescue. At its core, mine rescue is best served when decisions are based “on the numbers," the raw data as to the toxicity of the atmosphere and the potential for secondary explosions or fires. The emotion generated by media reports should not ever be a factor in those decisions. The mining community needs to address the rescue and recovery system in light of the new challenges presented by technology and the now ever-present media.”

It is imperative therefore that if rescue operations are not going to expose rescue personnel to an unacceptable level of risk that the relevant information must be available or gathered and used in risk management processes for rescue operations. This means that prior to deploying mines rescue teams underground it is essential to ensure that an explosive gas mix cannot come into contact with an ignition source, including residual fires.

If a flammable gas mixture exists it must be positively established that an ignition source does not exist. Evaluation of the underground environment must be mine wide with consideration of interfaces between different sampling locations. Although accurate measurements of the flammable gases can be made, it must always be remembered that the samples being analysed at any location are just from that point and the concentration of gases around that sample location may not be the same. Put simply the sample point may not be representative of the whole area. A conservative approach is required in interpretation of the status underground and the requirement for a conservative approach is increased when insufficient sampling locations are available. It is more important to know at an early point in time when a location is trending towards an explosive gas concentration rather than when it has been reached.

EXAMPLES OF MULTIPLE EXPLOSIONS AT MINES

Raspadskaya mine

In May 2010 rescue teams were killed in a secondary explosion during underground rescue operations following an initial explosion three and a half hours earlier.
Jim Walter Resources No. 5 mine

In September 2001 two explosions occurred at the Jim Walter No. 5 Mine only 55 minutes apart resulting in the deaths of thirteen miners. At least twelve of the miners were killed in the second explosion. The amount of methane involved in the first explosion was determined to be in the order of 100 ft$^3$. The second explosion also involved dust and a different ignition source.

Willow Creek

In 2000 there were four explosions at the Willow Creek Mine in Utah, United States of America. The timing between the first and second was seven minutes and then only one minute between the second and third with a fourth explosion twenty one minutes later. In total there were four explosions in twenty nine minutes. The amount of methane involved in the initial explosion was reported to be as little as 1.458 m$^3$ (50 ft$^3$). The initial explosion compromised ventilation assisting in the accumulation of methane and also started a fire (subsequent ignition source). The second explosion most likely created turbulence that caused high concentrations of methane in the goaf to mix with air.

Consol No.9 mine

In 1968 in Farmington, West Virginia the Consol No. 9 Mine suffered nine explosions over nine days with four in the first day alone. The second explosion was just two and a half hours after the first, with another two less than ten hours later. A day passed without further explosions but then two days after the initial explosion there were two within two hours. There was another nearly twelve hours later but then five days till the next, with the last explosion nearly one day after the previous.

A limited review on readily available information on incidents at mines involving more than one explosion is summarised in Table 1. There are multiple examples of second and third explosions within the first three hours, the time often referred to as being the “window of opportunity” for rescue operations. These examples show that there is no “window” during which it is safe to enter a mine after an explosion without risk of a secondary explosions and dispels the myth that all of the methane is used up in the initial explosion.

Table 1 - Examples of times between explosions

<table>
<thead>
<tr>
<th>Period Between Explosions</th>
<th>Mine</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 - 5 mins</td>
<td>Castle Gate No. 2 Mine (usmra)</td>
</tr>
<tr>
<td></td>
<td>Willow Creek (McKinney, et al., 2001)</td>
</tr>
<tr>
<td>5 -10 mins</td>
<td>Willow Creek (McKinney, et al., 2001)</td>
</tr>
<tr>
<td>10 - 20 mins</td>
<td>Castle Gate No. 2 Mine (usmra)</td>
</tr>
<tr>
<td></td>
<td>Robena No. 3 (usmra)</td>
</tr>
<tr>
<td></td>
<td>Eccles Nos 5 and 6</td>
</tr>
<tr>
<td>20 - 30 mins</td>
<td>Willow Creek (McKinney, et al., 2001)</td>
</tr>
<tr>
<td>30 - 60 mins</td>
<td>Jim Walter Resources No. 5 Mine (McKinney, et al., 2002)</td>
</tr>
<tr>
<td>1 - 3 hrs</td>
<td>Consol No.9 Mine (usmra)</td>
</tr>
<tr>
<td></td>
<td>Consol No.9 Mine (usmra)</td>
</tr>
<tr>
<td></td>
<td>Sayreton No. 2 Mine (usmra)</td>
</tr>
<tr>
<td>3 - 6 hrs</td>
<td>Bilsthorpe Colliery (usmra)</td>
</tr>
<tr>
<td></td>
<td>Raspadskaya Mine (Marquardt 2010)</td>
</tr>
<tr>
<td></td>
<td>Consol No.9 Mine (usmra)</td>
</tr>
<tr>
<td>6 - 12 hrs</td>
<td>Pond Creek No. 1 Mine(usmra)</td>
</tr>
<tr>
<td></td>
<td>Consol No.9 Mine (usmra)</td>
</tr>
<tr>
<td>12 - 24 hrs</td>
<td>Consol No.9 Mine(usmra)</td>
</tr>
<tr>
<td></td>
<td>Consol No.9 Mine (usmra)</td>
</tr>
<tr>
<td>1 - 2 days</td>
<td>Consol No.9 Mine (usmra)</td>
</tr>
<tr>
<td>2 - 3 days</td>
<td>Scotia Mine (usmra)</td>
</tr>
<tr>
<td></td>
<td>Moura No. 2#</td>
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<tr>
<td></td>
<td>Pike River Mine#</td>
</tr>
<tr>
<td></td>
<td>Pike River Mine#</td>
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<tr>
<td>3+ days</td>
<td>Consol No.9 Mine (usmra)</td>
</tr>
<tr>
<td></td>
<td>Pike River Mine#</td>
</tr>
</tbody>
</table>

# - Author’s own notes
CONCLUSIONS

- It is possible for flammable mixtures to build up very quickly after an explosion;
- Relatively small volumes of methane in explosive mixtures can have significant consequences if ignited;
- There are no fixed rules or guidelines for timing between explosions that can be followed after an initial explosion that allows a guaranteed non-explosion period and every case needs to be assessed;
- Deployment of mines rescue personnel needs to be risk managed;
- Decisions on whether to send rescue teams underground or to withdraw them once deployed need to be done so based on a scientific approach using data on the explosibility of the underground workings.

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CHALLENGES OF GAS MONITORING AND INTERPRETATION IN UNDERGROUND COAL MINES FOLLOWING AN EMERGENCY

Peter Mason

ABSTRACT: While modern coal mines are commonly equipped with a range of gas monitoring resources during normal operations, they are often lacking in their contingency planning when a mine loses communication with these systems following a major incident. Recent occurrences in underground coal mines in both Australia and overseas has highlighted the challenges in relation to effective atmospheric analysis and interpretation. This paper looks at the issues that face those involved, when not only the communication with existing gas monitoring systems has been affected, but the systems themselves have been destroyed.

INTRODUCTION

In discussing some of the challenges facing coal mines in relation to their gas monitoring and interpretation of data following an emergency, this paper will focus on those events that necessitate the evacuation of a mine as a result of heatings, fires and explosions. These are the occurrences that present the biggest challenges to ongoing gas monitoring at an affected mine.

While many modern coal mines would argue that they have effective atmospheric gas monitoring systems installed in their mines, few have adequate contingencies in place for the ongoing monitoring should a catastrophic event occur underground and these systems are destroyed. The lack of contingencies can severely impact on the effective management of an event as it is essential that representative and reliable data is available so that informed decisions can be made.

Speed of response is of the essence in these situations, particularly where workers may be trapped underground. The decisions in relation to evaluating the safety of the underground atmosphere are paramount if consideration is to be given to sending rescuers into a mine to assist in their escape.

GAS MONITORING

Typical gas monitoring systems

A range of different types of gas monitoring systems are used at underground coal mines. Aside from regulatory requirements, the selection, use, application and scope of these systems often depends on the individual mine and with regard to factors such as the seam gas content and propensity for oxidation of the coal seam. Typically one or more of the following are used:

- Telemetry fixed sensor system (sensors are located in situ underground – real time data);
- Tube bundle system (tubes are used to transport the underground atmosphere from selected locations to surface analysers – the lag time for sample retrieval is dependent on the length of tubes);
- Portable gas detection devices;
- Gas chromatographic system.

Telemetry fixed sensor systems are the most common type of gas monitoring system used by underground coal mines for atmospheric monitoring, providing real time data. Both telemetry and tube bundle systems are generally employed to monitor the same gases, namely oxygen, methane, carbon monoxide and carbon dioxide. With minimal restrictions in terms of the intrinsic safety, certifications, approvals, etc. for the analysers used for tube bundle systems (as they are generally located at the mine surface), a broader selection of analysers can be used for these systems.

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Portable gas detection devices are primarily used by mining personnel for routine inspections and personal monitoring while underground. A variety of multi sensor models are available for use by coal mines, but commonly measure the same gases as those monitored by the fixed and tube bundle systems.

Gas chromatographs have been used as an analytical tool for the analysis of underground coal mine atmospheres for over 30 years. However, analysis times for the earlier model chromatographic systems were generally too slow for the high volume sampling and analyses required during mine emergencies, lacked the desired sensitivity for critical gases and as a consequence were not widely considered as ideal for an on site gas monitoring system. The introduction of ultra fast micro gas chromatographs into the market has resulted in a wider acceptance and use of gas chromatographic systems at mine sites and they are now commonly used at mines in a number of mining regions. The ability of these systems to determine key gases such as hydrogen, carbon monoxide, ethylene, acetylene and ethane at parts per million levels is a significant advantage over other techniques.

System limitations post an event

While one or more of the above systems may provide adequate atmospheric gas monitoring during normal operations, the design and installation of telemetry and tube bundle type systems commonly do not take into account the issues that arise following a catastrophic event that results in loss of power underground and sealing of a mine. In these situations the likely damage to part or all of the underground gas monitoring components will render them ineffective following such an event. These systems are not generally designed to withstand the impact of significant heat, and in the case of the sensors that are utilised for telemetry fixed systems, their design ensures susceptibility to damage.

Even if underground power is able to be maintained and the integrity of the monitoring system has not been significantly compromised, other factors may also impact on the ability to manage an event using existing underground fixed sensor systems. The types of sensors used for these systems have a limited range of detection, can be cross sensitive to some products of combustion and may provide erroneous data regardless of the fact that they remain functioning.

While tube bundle analysers located on the surface would remain operational, the associated underground sample tubes, filters and other fittings are likely to have sustained damage. Depending on the severity of the event, it may take some time to determine which of the sample lines may have escaped damage and which are able to be used for sampling.

Although more common for the underground gas monitoring components to be damaged and rendered inoperable following a heating, fire or explosion, there have also been instances where surface equipment has also been damaged. Figure 1 shows photographs of damage to tube bundle system sample pumps following the surface ignition of methane that was being sampled from an underground sealed area and also the pumps as they appeared prior to the damage.

Figure 1 - An example of damage to a tube bundle pump system located on the surface

Gas monitoring post an event

Following a major event that has resulted in the sealing of a mine and loss of power, the mine will be restricted to using surface analysers for its ongoing gas monitoring requirements. These may include a
gas chromatograph if available, and tube bundle system analysers for mines who have an existing system installed.

A typical tube bundle system would normally be configured at a mine site to monitor specific gases such as oxygen, methane, carbon dioxide and carbon monoxide. However, they are not able to accurately determine key gases such as hydrogen, ethylene, acetylene and other gaseous products of combustion. The determination of these gases is essential for effective decision making in relation to the management of an event. The most effective method for these determinations is by gas chromatography.

Access to a gas chromatographic system is essential for mines with coal seams that have a high propensity for oxidation. Modern ultra fast micro gas chromatographs are able to provide accurate atmospheric analysis for the common coal mine gases (eg. oxygen, nitrogen, methane, ethane and carbon dioxide) and also for essential products of combustion (eg. hydrogen, carbon monoxide, ethylene and acetylene). A complete analysis for all components can be achieved in two to three minutes. These analysers have been successfully installed and used at coal mines for many years. While a mine may not utilise this type of system on site during normal operations, it should have a process in place for access to the technology from an external provider if required. Figure 2 shows a typical ultra fast gas chromatograph installed and operated by a mine at their site.

![Figure 2 - A typical ultra fast gas chromatograph installation](image)

Coordination of post event monitoring resources is also an important consideration. There may be external providers assisting a mine in delivering analytical support services and therefore the roles and responsibilities for all those involved needs to be clearly defined. Commonly in the early stages of a major occurrence, there is a period where those responsible are still establishing what resources remain available for use and what additional resources are required for the ongoing incident management. Time is a key factor in this process. Early decision making can significantly impact on the final outcome of the situation, and prevent a further escalation.

**Sampling considerations post an event**

The importance of a well coordinated gas sampling program cannot be understated. The interpretation resulting from the analysis of the samples collected will be a critical element in the decision making process that will follow. Sampling locations should be valid and sufficient in number to be representative of the area required to be monitored.

While the ideal situation would be to have fire and explosion proof gas sampling lines installed in a mine during the development process, it is not common for these contingencies to be put in place. Generally existing tube bundle sample lines (for mines that have these systems installed), or surface boreholes are the only available options for remote sampling of the underground atmosphere. As indicated previously, if a heating, fire or explosion occurs underground, it is often difficult to determine the extent of damage to the tube bundle sample lines and therefore the integrity of the data in relation to those sample locations. Existing borehole sampling may then be the only option.

Remote sites and adverse terrains pose their own problems for sampling. In these situations not only can the logistics for the collection and transportation of samples raise difficulties, but the establishment of new sample locations can also pose problems. If all underground monitoring components have been
destroyed, then a site will have to rely on the existing surface boreholes in order to obtain gas samples. Depending on the mine and the location of these boreholes, they may not be adequate in providing a representative overview of what is occurring underground. The drilling of additional boreholes to provide the sampling coverage required may also be difficult in these situations and take some time for completion.

Ideally all sample boreholes should be cased, capped and sampled when breathing out. Down borehole sample tubes should be used to ensure that a representative sample is collected from the specific location required. Samplers should be trained and ensure that gas lines and sample containers are sufficiently purged prior to taking gas samples. All relevant details in relation to the samples should be recorded, e.g. sample location, type of sample, date, time, whether the sample point is breathing in/out and any other relevant information. Recommended standards or guidelines in relation to gas sampling should be adhered to (Queensland Mines and Energy, 2009). Sample collection must be consistent to ensure meaningful comparison of the analytical data collected.

If surface boreholes are used for sample collection, and if distances between the borehole locations are such that it is possible to run some or all sample lines back to a central analyser system, then continuous sampling and data collection from all these sites is possible. Suitable gas sample pumping systems would be required to achieve this outcome. Apart from the ability to monitor and collect gas data continuously and provide as close as possible to real time trending, it significantly reduces the requirement to manually collect and transport samples.

Consideration must also be given to the safety of the gas samplers. Both terrain and sample locations need to be considered. If potentially flammable atmospheres are being sampled, then the risk of ignition at both the surface sample point and underground atmosphere must be considered. Sufficient care must be taken to ensure samples are collected in a safe manner. If samples are being collected from boreholes, this may require the sample tubes to extend a safe distance from the actual borehole.

**INTERPRETATION OF THE DATA**

Essential to the effective interpretation of the analytical data is the integrity of the information provided to those interpreting the results. While absolute accuracy is desired, it is not always possible to achieve in the initial period following a major occurrence. While modern analysers such as ultra fast micro gas chromatographs and other quality analytical systems are able to provide consistent and reliable analytical data in laboratory environments, commonly these devices are used in temporary make shift situations at mine sites during the early stages following an emergency, where the operating environment is not always ideal. Attention to routine maintenance and regular calibration of the on site systems is therefore essential in these circumstances to ensure the provision of consistent and reliable data.

When interpreting the data in relation to an event such as a heating, fire or explosion, explosibility of the atmospheres being sampled must be calculated, trended and graphically displayed. Regardless of what method is used for this purpose, expertise is required in reviewing the data to ensure other parameters that may affect the explosibility of the atmosphere being monitored are taken into consideration. These include not only the levels of flammable gases present, but also the impact of the ingress of air and known or suspected sources of ignition.

Although reviewing the raw analytical data is essential, trending of it in conjunction with relevant ratios and indices provides a better indication of what is happening in the underground environment. While there are a variety of ratios and indices that have been developed to assist in the interpretation of gas data produced by heatings, fires and explosions, relatively few have stood the test of time and used on a consistent basis. These have been well documented (MDG 1006 Technical Reference, 2011; David, et al., 1996; David, et al., 1998; Mine Rescue Board NSW, 1998), with the following to have shown to be of consistent value over a long period:

- Graham’s ratio;
- CO/CO₂ ratio;
- CO make;
- Jones-Trickett’s ratio;
• Young’s ratio;
• H₂/CO ratio;
• Air free calculation.

As the ratios used for heatings and fires are a generally a measure of the conversion efficiency of oxygen to the products of combustion, it is not necessary to calculate and trend significant numbers of these deficiency ratios as the data will simply show the same trends.

Ratios such as Graham’s, CO/CO₂, Jones-Trickett’s, H₂/CO are useful tools for interpretation as an increase in these ratios normally indicates an increase in temperature. However, if a mine is sealed, it is likely that the site will deploy some form of inertisation to manage the situation. If this occurs then care must be taken when reviewing the gas data as some of the common ratios used for interpretation and prediction will be compromised by the addition of the inerting gases into the underground environment.

This situation raises additional complications when more than one type of inerting system is deployed. Inertisation systems such as membrane nitrogen generators (commonly called “Floxyl” systems), vaporising liquid nitrogen systems and pressure swing absorption systems (PSA) all essentially produce high purity nitrogen as the inerting product. Other systems such as the Gorniczy Agregat Gasniczy (GAG) jet engine and the Tomlinson boiler also produce a relatively high purity nitrogen product, but also produce up to 15% carbon dioxide. While the analytical data will show the presence of these inerting systems, and therefore their effectiveness, their use then negates the application of ratios that contain these gases in the calculations.

Although commonly used, care should be exercised when using air free calculation of the raw analytical data for interpretation purposes. Whilst successfully used for many years to compensate for the ingress and expulsion of air from behind seals and boreholes that may mask increases/ decreases for other gases being monitored, it can provide erroneous estimates of residual gases in atmospheres close to fresh air. While a useful tool, it should be used by experienced personnel.

During a major occurrence there can be significant external pressure on those reviewing and interpreting the data to provide comment and advice in a timely manner. This may come from sources such as mine owners, governments, media, mine workers, families and others. A simple approach is often the most effective in the decision making process, with those placed this position of responsibility ensuring that the task at hand is not influenced by these external pressures. Clear and concise comment in relation to the findings is essential.

CONCLUSIONS

While some aspects of the challenges of gas monitoring and interpretation in underground coal mines following an emergency have been discussed in this paper, the topic is significant and only a brief overview has been provided. It would be an ideal situation if increased diligence in safe mining practices resulted in such papers being viewed as pure discussion. Unfortunately this is not likely to be the case in the near future, and the need for increased expertise in this field is an essential component of underground coal mining.

REFERENCES

CHALLENGES ENCOUNTERED BY NEW ZEALAND MINES RESCUE AT THE PIKE RIVER MINE DISASTER

Trevor Watts

ABSTRACT: The loss of 29 lives in the Pike River Mine Disaster of 19th November 2010 will be forever remembered as one of the darkest days in the history of coalmining in New Zealand. The effects of this tragic event have also been felt by the mining industry in Australia. As an industry we are constantly aware of terms such as “Emergency Preparedness” and “Emergency Response Management Plans” and in fact, numerous seminars and forums are facilitated to study these topics in detail. This begs the question, "how well is your organisation really prepared if it was faced with a major disaster such as that which occurred at Pike River"? The incident management team, mine manager, mines rescue and other emergency organisations responding to the Pike River mine explosion faced significant challenges on planning a re-entry into the mine by rescue teams.

INTRODUCTION

Coal mining disasters can occur from a variety of circumstances such as explosions, inundation of water or fires to name a few. Every incident will provide different and at times unique challenges to people charged with control and management of the situation. Incident management teams rely on accurate and reliable information available to form the basis of robust decision making processes.

An emergency event can rapidly escalate depending on the dynamic nature of the situation and can also deteriorate quickly on the back of poor decision making. The gathering, analysing and interpretation of information form a critical function of incident management. Obtaining information during an emergency can be particularly challenging as often a significant amount of diverse information is required quickly.

Incident controllers are required to make numerous critical decisions and in some instances lives are dependent on people in incident management roles making decisions based on accurate, reliable and timely information. One of the most critical decisions that an incident management team and incident controller will be required to make in response to a dynamic incident such as an explosion, is the decision to deploy rescue teams into a mine.

This paper will provide an overview of the challenges that the incident controller, incident management team and mines rescue officer in charge were faced with in the hours following the explosion at the Pike River Mine. The known and unknown information that was available following the explosion will be described along with the complexities associated with gathering information.

The Royal Commission of Inquiry on the Pike River Coal Mine Tragedy was still sitting at the time of writing of this paper. Every endeavour has been made to ensure that this process is not jeopardised by the inclusion of subject matter that is subjective or still before the Commission. This paper is not intended to cut across the vital Royal Commission in any way, but rather to provide summary information in the context of the Coal 2012 Conference.

OVERVIEW OF THE PIKE RIVER MINE

The Pike River Mine is located 50 km north of Greymouth on the West Coast of the South Island of New Zealand. Construction of the main access tunnel commenced in 2006 and eventually intersected the coal seam at approximately 2.3 km. The main access tunnel is inclined at an average gradient of 1 in 10 with the tunnel dimensions approximately 6.5 m wide by 5.5 m high. The coal seam is between 3 and 12 m thick and the methane seam gas was measured from one sample as high as 10 m³/t.

The design of the mine with the incline tunnel intersecting the coal seam at the lowest point meant the seam gas make was at its highest in the early development stages of the mine. A gas drainage system
was used at the mine and a number of inseam boreholes were linked to a gas drainage line. Once the main tunnel had accessed the coal seam, a 110 m return ventilation shaft was constructed using a raise bore technique. The bottom 40 m of the shaft collapsed before strata stabilisation work was completed.

Following the shaft collapse a 2.5 x 2.5 m vertical raise shaft was driven from within the mine which then intersected the supported section of the main ventilation shaft. The vertical section of the shaft was 55 m high in which a ladder was installed. The 2.4 km access tunnel was the only practicable means of egress.

The mine was still in the early stages of development with only one small extraction panel completed. Development headings were driven using two continuous miners and one roadheader. Shot firing was required in areas of stone drive development and a hydraulic monitor was the method of mining coal in the extraction panel. Coal was cut using high pressure water and the water sluiced the coal to the pump bay located in the pit bottom area.

At the time of the disaster, Pike River Coal Ltd employed approximately 160 permanent staff and also had up to 60 contractors engaged in a variety of infrastructure work.

OVERVIEW OF THE NEW ZEALAND MINES RESCUE SERVICE

The New Zealand Mines Rescue Service was established in 1930 and employs six full time staff, the general manager, three training officers and administration officer based at the Rapahoe Rescue Station on the West Coast of the South Island and a station manager based at the Huntly Rescue Station in the North Island.

At the time of the Pike River Mine Disaster, the New Zealand mines rescue service brigade strength consisted of 24 underground rescue personnel in the North Island and 33 underground personnel on the West Coast of the South Island. Additionally the West Coast has 30 surface rescue personnel.

THE EVENTS OF 19TH NOVEMBER 2010

At 3:44 pm the control room officer was talking to a miner underground when communications were suddenly lost. At the same time the computer monitoring system started alarming and indicated that power had been lost underground. The control room officer attempted to contact personnel throughout the mine but was unable to raise anybody. He contacted the mine manager who was in a meeting with other senior mine officials and informed him of the situation.

The mine manager discussed the loss of power with the engineering manager who subsequently arranged for an electrician to go underground to investigate possible faults at the main electrical bay. The loss of power was not an unusual occurrence at the mine due to the fact that the mine was at the end of a main supply and had been damaged on previous occasions as the line was close to trees for 7 km.

The electrician entered the mine in a drift runner and after driving 1 500 m he came across a loader and a man lying on the ground. At this point he suddenly found breathing difficult and his vehicle was losing power. He returned to the surface and contacted the control room to report his observations and stated that he believed that there had been an explosion in the mine. This was the first confirmation that an explosion was believed to have occurred in the mine and the mine management initiated the mine emergency response procedures.

Phone call from survivor

At approximately 4:35 pm; 50 min after the explosion; a phone call was received from the pump bay area which is located 1 900 m into the mine. A miner who had survived the explosion spoke to the mine manager and stated that he could hardly breathe and the mine was full of smoke. The manager told him to stay low and start making his way down to the mine portal. He managed to make his way down the main tunnel until he came across the other survivor who was in a semi-conscious state. The miner then continued out of the mine dragging his colleague with him and both men exited the mine at 5:25 pm; almost 90 min after the explosion. The men were treated by Paramedics and immediately taken to hospital. No information could be obtained from the men prior to leaving the mine due to their poor physical condition.
Mines rescue response

At 4:30 pm the Rapahoe Mines Rescue Station received a call from the Pike River mine control room and the caller stated that they believed there had been an explosion underground. The staff at the rescue station initiated the emergency response procedures and began the task of contacting brigade personnel. Three teams were assembled at the station within 30 min and Drager BG4 breathing apparatus and other essential rescue equipment was readied for deployment.

The first two teams were transported to the mine site by the local rescue helicopter and an additional two teams travelled to the mine by road. The rescue helicopter had the first rescue team and officer in charge on site at 6:00 pm. On the approach to the landing site the mines rescue officer in charge instructed the pilot to fly past the return ventilation shaft.

The sight of the damaged évasée was confirmation that a significant explosion had occurred. In addition to the damaged évasée and infrastructure at the top of the return shaft, soot was observed in the trees across the valley directly in line with the outlet of the évasée and smoke was seen to be drifting from the return shaft.

When mines rescue arrived at the mine site the mine manager stated that no-one, including rescue teams; was to enter the mine due to a lack of information on the underground environment. By 6:20 pm four fully equipped mines rescue teams were on site and were completing preparation of their equipment and planning strategies for a rescue operation.

SUBSEQUENT EXPLOSIONS

The second explosion occurred at 2:37 pm on 24th November. The second explosion came as a devastating blow to the families of the twenty nine men as it was conveyed to them that it was clear from video footage that this explosion was not survivable. Many families had clung onto the hope that their loved ones had survived the first explosion and were still alive in the mine. The news of the second explosion crushed those hopes. It was officially announced that the operations had moved from one of rescue to that of recovery.

A third explosion occurred on the afternoon of 26th November. This explosion was not as violent as the previous explosions but reinforced the urgency that was required to seal the mine.

On the afternoon of 28th November the fourth extremely violent explosion occurred. The force of this explosion blew the seven tonne évasée off the top of the ventilation shaft, landing some distance away. Thick black smoke immediately began rising out of the ventilation shaft which clearly indicated that a large fire was now burning in the mine.

A few hours later flames emerged out of the shaft and these were initially up to 50 m high. The fire raged for another eleven days before the flames were finally extinguished with the use of the Queensland Mines Rescue Gorniczy Agregat Gorniczy (GAG) jet engine unit. Following the fourth explosion a major fall occurred in the main access tunnel as the natural ventilation flow into the mine ceased.

CHALLENGES OF OBTAINING CRITICAL INFORMATION

Information required for the deployment of mines rescue teams

In the early stages of this incident it soon became apparent that there was a lack of critical information available to the incident management team. The mine manager had given a clear directive that mines rescue teams were not to enter the mine due to a lack of information on the atmospheric conditions underground. This position was maintained by the NZ Police. The author fully supported this decision.

In the hours and days following the first explosion, the NZ mines rescue focus was on gathering critical information on the underground atmospheric conditions. Until such time that accurate and reliable intelligence could be analysed and interpreted, the level of risk associated with the deployment of mines rescue teams could not be adequately determined or fully understood.
The decision to deploy or not deploy rescue teams into the mine could only be based on the information available at the time. Decisions were based on “known” and “unknown” information, underpinned by robust risk management principles.

**Known information**

The known information available to the incident management team and the mines rescue officer in charge in the hours following the Pike River mine explosion was extremely limited but of significant importance to the decision making processes associated with determining the level of risk associated with the deployment of mines rescue teams. Known information is described as follows:

- The mine manager and the mines rescue officer in charge had flown over the return shaft and viewed the damaged évasee and adjoining infrastructure. This observation along with soot in trees on the side of the valley opposite the évasee was confirmation that a significant explosion had occurred.

- Smoke was seen to be drifting out of the return shaft. This was a critical piece of information as it had to be accurately determined if the smoke was a product of the explosion or combustion that still remained in the mine.

- The mine had a high gas make. Information provided stated that the gas make from the upper workings of the mine was approximately 200 L/s. Only six weeks prior to the disaster the main fan on the surface malfunctioned and the entire upper mine workings “gassed out” in a few hours. NZ mines rescue was called to assist with this event so was well aware of the mine gas make.

- The mine gas drainage line had been fractured somewhere underground. Information provided indicated that this may be delivering up to 800 L/s into the mine in addition to the normal mine gas make. Approximately 1000 L/s of methane was filling the upper workings of the mine. Total void volume of this area was approximately 50 000 m$^3$.

- The main ventilation fan was located underground and power was off. It was considered highly probable that the ventilation control devices in the mine would be destroyed, particularly the double doors in the first crosscut between the intake and return shaft.

- Natural ventilation was entering the mine due to the inclined main tunnel and elevation differential between the intake portal and return shaft. Sufficient oxygen would be present to support further explosions (the quantity of air was measured at between 12 - 15 m$^3$/s).

- The gas monitoring system had been lost as the mine only had a real time monitoring system installed in the mine. There was absolutely no intelligence on the underground atmospheric conditions.

To summarise the bullet points above, this was the information known to the mine manager and mines rescue officer in charge in the first few hours of the incident. A significant explosion had occurred in the mine, there was a high make of methane, smoke was coming out of the return and there was natural ventilation into the mine. Of the three main factors required for another explosion, two were present and the third (an ignition source) was possible and had to be suspected.

Additionally the following information was also known to the incident management team:

- Video footage of the windblast exiting the mine portal had been viewed. When the size of the mine is taken into consideration, the significance of the video footage is brought into perspective. This was a very small mine and a 52 sec windblast was recorded exiting the mine portal. Excluding the void volume of the main access tunnel, the total volume of all mine workings was approximately 80 000 m$^3$;

- Two men had survived and walked out of the mine, 1 h and 45 min after the explosion;

- Possibly 34 men unaccounted for underground. The correct number of men missing was not confirmed by Pike River Mine until 10 h after the explosion;

- No communications had been received from any other locations in the mine. Additionally mines rescue personnel had lowered a radio down the slimline shaft into the area known as the “fresh air base” and remained at this location for many hours;
There was only one practicable means of egress from the mine via the main, 2.4 km main access tunnel. The second means of egress was via the return shaft which would not have been possible following the explosion. The main drift was also the only access for rescue teams to re-enter the mine but this route meant they would have also been in direct line of any secondary explosion. The walking time to the top end of the main tunnel under normal circumstances was approximately 45 min;

The walking time for rescue teams would have been considerably longer given the equipment the rescue teams were required to carry. If rescue teams were required to be withdrawn due to changes in the underground environment this could not have been achieved within a period of time acceptable to mines rescue. A loader operated by one of the survivors was blocking the main tunnel at 1 500 m which precluded the use of vehicles beyond this point;

The workforce at the mine was trained to self-escape and all underground personnel were equipped with 30 min self-contained self-rescuer units;

There was a cache of approximately 100 self-contained self-rescuers stored in the Fresh Air Base (F.A.B) at the slimline shaft;

There were no boreholes into the upper workings of the mine to obtain gas samples;

The real-time gas monitoring system had been lost. There was no road access to the return shaft; helicopters were required to obtain gas samples for Gas Chromatograph analysis at the mines rescue station. Only three samples were obtained in the first few hours as poor weather conditions prevented helicopters from flying until the following morning;

The first gas samples analysed were highly diluted due to the difficulties in obtaining samples and the natural ventilation short circuiting through the first crosscut. These two factors meant that the gas samples obtained were not truly representative of the underground environment. Table 1 contains the details of the gas chromatograph analysis.

<table>
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<th>#</th>
<th>Location/Time</th>
<th>CO%</th>
<th>H2%</th>
<th>O2%</th>
<th>N2%</th>
<th>CH4%</th>
<th>CO2%</th>
<th>C2H6%</th>
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<td>0.0507</td>
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<td>77.29</td>
<td>0.2940</td>
<td>0.1020</td>
<td>0.0030</td>
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<td>0.0154</td>
<td>20.49</td>
<td>77.49</td>
<td>0.1280</td>
<td>0.0770</td>
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<tr>
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<td>20.44</td>
<td>77.53</td>
<td>0.2530</td>
<td>0.1090</td>
<td>0.0032</td>
</tr>
</tbody>
</table>

Unknown information

- What was the composition of the atmosphere underground? This was perhaps the most significant issue facing the incident management team. Without accurate and reliable intelligence on the underground environmental conditions, it was not possible to adequately determine if there was an acceptable level of risk to deploy rescue teams into the mine.
- No intelligence from within the mine coupled with the fact that smoke was observed at the return shaft meant an ignition source had to be suspected as it could not be eliminated.
- What was the ignition source and did it or a secondary ignition source still exist underground? It was absolutely critical to ascertain if an ignition was present as it was highly suspected that a large volume of methane would be present in the mine.
- Where in the mine did the explosion occur?
- Was the smoke drifting from the return shaft the product of Afterdamp or combustion?
- What was the location of all the men working in the mine?

Obtaining critical information

Obtaining gas samples, temperature and pressure readings from within the mine were identified as a top priority. In the days following the explosion samples were obtained from the ventilation shaft hourly and analysed by parallel gas chromatograph analysis at the mines rescue station and mine.
However, obtaining gas samples proved to be a considerable challenge due to a number of factors. As previously described, the only access to the return shaft was by helicopter and to travel by foot, was a six hour round trip on foot in steep, mountainous terrain. Highly diluted samples from the return shaft meant there was no confidence in the data. Frustratingly there were no other sampling points available from the upper mine workings.

Early interpretation of gas data from the return shaft by Mr Robin Hughes, a highly respected and experienced New Zealand Ventilation Officer, Mine Manager and ex Chief Inspector of Mines was that there was a strong likelihood that a fire was burning in the mine. He stated that based on the data available he suspected this could be a clean burning methane fire.

In the early stages of the incident it was identified that additional boreholes were required. Suitable locations for boreholes were ascertained and an expert drilling team was quickly assembled. A suitable pad was constructed at the drill site with helicopters used for transporting all equipment and personnel. Crews worked around the clock to drill the 165 metres into the mine. Extremely difficult conditions were encountered, slowing the rate of drilling due to the very hard rock encountered. Breakthrough was achieved early on the morning of the 24th November.

Gas samples were taken following the completion of the borehole and expert interpretation of the gas analysis was provided by Professor David Cliff from the University of Queensland. Professor Cliff stated that it was his opinion that a methane fire was burning in the mine. A short time later this interpretation was confirmed when a second, violent explosion occurred in the mine.

COMPLEXITIES ASSOCIATED WITH THE DISASTER

Along with the many challenges that continually tested the incident management team, this was a complex incident on many different levels.

From a human factor perspective, it is difficult to fully articulate the effect that an incident of this scale and magnitude had on all those involved in the rescue efforts. The West Coast of New Zealand is a small tight knit community and all the men who lost their lives in this tragedy were well known to many of the people involved in the rescue efforts.

Highly trained mines rescue personnel were desperate to enter the mine. Eventually this desperation turned to despair and immense frustration as time went on as no rescue operation could be launched. However, the professionalism of the rescue personnel came to the fore. The rescue teams remained in a high state of operational readiness and continually worked on developing strategies that would be implemented if the incident management team could determine that there was an acceptable level of risk to allow entry into the mine.

The raw emotion that was displayed when mines rescue teams and all those involved in the incident were told of the second explosion is difficult to express. This is the moment that even the most remote hopes of a miracle were lost and the terrible realisation that 29 family members, friends and colleagues are dead.

The mountainous, inhospitable terrain tested the toughest of men and if it had not been for the skills of the highly experienced helicopter pilots, the work required to obtain gas samples would not have been possible. The West Coast is renowned for its high rainfall. Fortunately weather conditions only prevented helicopters from flying on a few occasions in the first days of the incident. At times rain can set in for many days or weeks on end which would have grounded helicopters.

SUMMARY

This paper has focused on the challenges associated with gathering information in the early stages of the Pike River mine disaster and what impact that the known and unknown information had on the rescue efforts.

This disaster has been and continues to be a complex and challenging event that has been continuous for just over a year at the time of writing. To fully encapsulate all of the complexities and challenges associated with this tragedy in one paper would be virtually impossible.
PIKE RIVER MINE RE-ENTRY AND EMERGENCY MINE RE-ENTRY GUIDELINES APPLICATION AND LEARNINGS

Geoffrey Nugent, Darren Brady, David Cliff and Seamus Devlin

ABSTRACT: Prior to the Pike River Mine Disaster the Queensland Mines Rescue Service and The NSW Mines Rescue Service undertook a project to develop a guideline and a practical prototype software tool to demonstrate how decision makers could be better assisted during a mine emergency which required re-entry to the mine by competent mines rescue trained personnel. The research and development of the prototype software tool (funded through ACARP grant C19010) coincided with the unfortunate events at Pike River Mine on and after the 19th November 2010.

This paper will discuss the relationship between this project's outcomes and the re-entry strategy and operation at the Pike River Mine, along with learning's the researchers gained from this operation undertaken by the New Zealand Mines Rescue Service and Pike River Coal Limited (in receivership).

INTRODUCTION

The aim of ACARP project C19010 Emergency response - mine entry Data Management (Nugent, et al., 2011) has been to provide industry with a practical and cost effective example of how software based information management systems could be developed and utilised to assist Incident Management Teams (IMT) with effective information management and critical decision making during an emergency.

This research project was successful in developing a functioning proof of concept software tool titled "Mine Re-entry Assessment System" (MRAS), which supported and reflected the paper based guideline for Emergency Mine Entry or Re-entry (EMER), developed by Queensland Mines Rescue Service (QMRS) and New South Wales Mines Rescue Services (NSWMRS). The proof of concept software tool has been developed in Microsoft Access which is inexpensive and commonly available software to most computer users. The need for such a tool and process has been identified and recommended in disaster investigations, emergency exercise reports and research reports alike, highlighted by the following examples:

- From Moura No.2: Inquiry Task Group four (Mines Rescue Strategy Development) 1994; Knowledge of conditions in a mine following an incident is essential in planning any rescue effort. Information systems must be provided to support implementation of the most appropriate rescue measures;
- From Moura No.2: Inquiry Task Group four Recommendation (17) 1994; Industry should develop an effective computer-based emergency decision support system for incident management and training;
- From finding No.8: Upper Big Branch, Independent Report to the Governor 2011; The Upper Big Branch disaster raised concerns about how decision-making was conducted in the command centre and the manner in which mine rescue teams were deployed underground. Standard protocols were not followed, effective records were not kept and rescuers' lives were placed in jeopardy.

PROTOTYPE TOOL DEVELOPMENT AND FUNCTIONAL SPECIFICATION

Approximately 70% of required relevant information to make an informed risk based decision during an emergency could be known before an incident occurs. Therefore the identification of, along with the maintenance and accessibility to this relevant routine information can significantly enhance an incident management team's ability to reach critical decision points in a timely manner (Nugent, et al., 2011).

Queensland Mines Rescue Service and NSW Mines Rescue Service have been working together with other parties on a three phase project to develop new guidelines for emergency mine entry and re-entry.

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using risk management logic based on a formal documented risk assessment (risk assessment for emergency mine re-entry, 2009) as well as a “tool” for information management. Use of such a tool to minimise the time to make a decision, may in many circumstances eliminate the issue of not acting because of what is not known.

To have the critical information required to make informed decisions, the controls from the mine re-entry risk assessment were analysed to identify who was responsible for the collection or interpretation of the information (i.e. someone from the mine site, mines rescue or an external provider). The ability to have and maintain this information prior to a response was also determined as was whether the information could be generated automatically or had to be collected manually. A significant volume of information may be required so the importance of the information required was also ranked to help those involved to set priorities for information collection.

A task group then performed a gap analysis on what information was identified as required and what information was available. A cross section of mining operations in Queensland and New South Wales were selected and a total of eight mines visited by the task group to establish what information was and wasn’t available. The site visits identified there were some very good systems for making some of the information available but also identified some common trends in relation to deficient emergency response information management.

It was found that although a lot of information was captured or available it could not be provided within an acceptable period of time. Critical information or knowledge was sometimes held by only one or two people and thus dependent on their availability. This availability is made worse by the dependence on persons in key roles (particularly technical, such as ventilation officers), who are responsible for many tasks in the response with little or no backup.

It was also identified that not always do the people responsible for monitoring understand what it is they are monitoring. There were also areas where information required was not available. This was particularly the case for information required for validation of operational systems.

Information relating to the actual event unfolding is often missed due to inefficient and inconsistent debriefing of key witnesses. Communication officers or control room operators through whom most communications from underground go are often so inundated with communications that key information is either missed or not identified as important and not passed on to decision makers.

There were many similarities between the findings of the gap analysis and reported findings from Queensland’s level one emergency exercises for both good and deficient practices.

The software system eventually developed was never intended to make decisions for those tasked with making decisions during an incident, instead it is a decision support system, providing the information required or identification of absence of information allowing the decision makers to make informed decision. The functional specification incorporated in the checklist is developed as part of the guidelines phase of the project as well as store information from mine site sources into a central repository. This repository would allow mine site personnel to make an objective assessment of the conditions for re-entry, based upon the quantity and quality of the information available.

The intent of the software development was not to create or develop a new standalone software package unique to an emergency mine re-entry information management and decision making tool. The researchers assessed and evaluated the capability and capacity of commonly available software tools for their suitability and level of functionality against the objectives of the project.

The three main software tools considered for trial were Microsoft Excel, web based and Microsoft Access. The researchers engaged three separate technical experts for each software tool to assist with the evaluation of software suitability along with design and creation of the functional specifications required of the software. This evaluation led to the further development of the Microsoft Access prototype primarily due to its flexibility and reporting functionality (Brady, et al., 2012). The functional specification requirements were determined to include:

- Identify and access or collate relevant information already existing within the mine’s safety and health management system;
- Identify and access the critical information relevant to the specific incident, to avoid unnecessary information overload;
- Provide reports that quickly summarise the information status relevant to identified hazards to assist the decision makers determine what information is known or unknown;
- Assist decision makers to prioritise required and outstanding information to facilitate efficient resource management;
- Maintain an up to date log of an incidents status;
- Force decision makers to formally acknowledge the adequacy of information;
- Prompt a formal process for the assessment and acknowledgement of explosibility risk;
- Provide a formal approval process for entry or re-entry into a mine during or after an incident.

It must be recognised that this, or any tool, developed to support and facilitate the process of information management and decision making during an emergency at a mine can only be effective when:

- As a minimum its content and structure reflects the current requirements set out in the risk based guideline the tool has been established from;
- The users of the tool have a good understanding of its functionality and purpose along with a high understanding of risk management processes and practices.

Pike River

During this project a significant mine disaster occurred in New Zealand claiming the lives of 29 men. The Pike River Mine disaster which occurred on the 19th November 2010 came only two days after the QMRS and NSWMRS Emergency Mine Entry or Re-entry (EMER) Guideline was presented to industry at the QMRS Inertisation Seminar in Mackay, Queensland. At this stage, although the EMER Guideline was developed in full in a paper based format, the proof of concept software tool development was only in its infancy.

One of the key roles carried out by the researchers at the Pike River Mine rescue and recovery operation was advice and assistance in the strategy for and development of an effective mine re-entry management plan. The assistance provided to Pike River mine management and NZMRS by the researchers included:

- Risk Assessment facilitation and participation;
- Gas Analysis and interpretation;
- Hazard management plan and procedural development and review;
- Incident Management Team Participation.

NZMRS who were responsible for the establishment of the Pike River Mine Re-entry Hazard Management Plan (HMP), after consultation with the researchers elected to base the logic and process for the development of the re-entry plan on this project’s research outcomes.

Whilst the Pike River mine Re-entry HMP was being developed by the NZMRS in consultation with this project’s researchers, the development of the MS Access MRAS software tool was also being further progressed with both completed at the end of May 2011.

Due to the Pike River Mine Re-entry HMP being closely aligned with the strategies and processes outlined in the EMER Guideline, the risk logic and process applied during the re-entry was also compatible with the newly developed MRAS software. Because of this, the Mine Re-entry Control Team (MRCT) in control of the re-entry operation agreed to utilise the MRAS software to assist the MRCT with information management and decision making processes.
MRAS practical application at Pike River re-entry

The primary objective of the re-entry operation was to install a fit for purpose seal within 300 m of the main portal to provide a more effective seal preventing oxygen ingress and to augment the natural stabilisation of the underground mine atmosphere. Additionally, the successful installation would allow Pike River Coal Limited (in Receivership) employees to reclaim the temporary seal at the portal and install fit for purpose double vehicle doors for potential future access.

Regardless of the distance mines rescue teams are required to travel into a mine after a significant incident (e.g. 300 m or 3 000 m) the precautions taken and the diligence demonstrated must be no less comprehensive in either circumstance as the consequences of potential hazards, if realised, would be no less severe.

Based on this logic the NZMRS initiated and facilitated a comprehensive re-entry risk assessment for the re-entry of the Pike River Mine. This risk assessment underpinned the establishment of the NZMRS Pike River Mine re-entry HMP which also incorporated the following hazard management plans, procedures and Trigger Action Response Plans (TARPs):

- Pike River Mine Atmospheric Monitoring HMP;
- Pike River Mine Atmospheric Monitoring TARP for Re-entry;
- Pike River Mine Atmospheric Monitoring Check Sheet;
- NZMRS Re-entry Explosibility TARP and Check List;
- NZMRS Mine authority to Enter;
- Strata Management Plan and TARPs;
- Pike River Mine Water Management Plan;
- Pike River Mine Seal design and installation SOP;
- NZMRS seal installation and operational Risk Assessment and Guidelines.

The overriding structure outlined in the NZMRS Re-entry HMP was to ensure that:

1. A MRCT was responsible for managing all activities required to control the Pike River Mine re-entry from an operational perspective. It would manage and co-ordinate the interface between the MRCT functional groups (Operations, Planning and Logistics) and with stakeholders outside of the mine re-entry management structure;
2. A defined process and logic was established to ensure the gathering and assessing of relevant information and decision making was systematic, unambiguous and well informed.

The management of the re-entry operation was conducted via the established Queensland Mines Rescue Service Mine Emergency Management System (MEMS), Figure 1. The established information management and decision making process was founded on process outlined in the QMRS and NSWMRS Emergency Mine Re-entry Guideline and is outlined in Figure 2.

Prior to the initial decision by the MRCT to re-enter Pike River mine the Pike River Information Management check sheets (listed below) were completed and all information collated deemed adequate by the MRCT for the assessment of known information against all applicable control procedures and TARPs:

- Mine atmospheric monitoring check sheet;
- Mine ventilation management check sheet;
- Explosibility check sheet;
- Mine strata management check sheet;
- Mines rescue operational deployment check sheet;
- Mine re-entry critical support and resources check sheet.
The objective at this stage was to evaluate and determine a number of issues:

- Where there was outstanding information, what was its significance for making an informed decision on the risk to rescuers entering or remaining in the mine;
- Did the outstanding information impact on an authorised person’s ability to evaluate and conform to the requirements of the mine re-entry control procedures.

Figure 1 - Mine re-entry control team structure

Figure 2 - Information management and decision making process
This process was conducted and maintained via a paper based gap analysis process (consistent with the EMER guideline) developed by the NZMRS and reviewed as a minimum on a daily basis or where change occurred.

Once it was determined by the MRCT that all required information was adequate to make an informed decision the data and information would be applied to the established control measures for the MRCT to decide whether the risk to rescuers re-entering the mine was within acceptable limits.

The key functional elements of the MRAS software utilised by the MRCT to assist with decision making, further information management and document control were:

- Re-entry control questions for the assessment of explosibility risk;
- Explosibility graph (MS Excel) for assessment of explosibility, trending and rate of change at each sample point;
- Management of actions required, priority and status as set by the IMT;
- Current situation reporting and recording;
- Authority to enter document.

The assessment of explosibility risk

When the MRCT assessed the risk of explosibility to rescuers the following conditions had to be met and acknowledged:

1. Any analysis system of a potentially explosive atmosphere must consider all flammable gases present in the atmosphere and determine:
   - The flammable gas content expressed as a percentage of the Lower Explosive Limit (LEL) present;
   - The oxygen content expressed as a percentage of the Oxygen Nose Point (ONP) present.

2. All samples, from the same location, for both LEL and ONP must be plotted and trended together against time on the same graph. The same graph should have the capability to plot barometric pressure trends against time.

3. When determining the explosibility risk level the following controls must be observed:
   - Prior to entry the assessment of relevant knowledge for mine re-entry must be completed and all information deemed to be adequate to determine explosibility risk level;
   - Atmospheric conditions must be continuously monitored by rescuers underground and monitored and trended on the surface by a competent person at all times while people are underground;
   - When it cannot be confirmed there are no potential ignition sources which may come into contact with a potentially explosive or explosive environment it must be taken that an ignition source exists;
   - The level of assessment must be based on the highest level of risk within the mine not just the area which the rescue team is going to enter;
   - These control limits may only be modified when a full re-entry RA is undertaken for the specific emergency situation. The original risk assessment must be reviewed as part of this process;
   - Currency of the data must be taken into account when assessing gas results e.g. Tube bundle lag times and the time bag samples were taken;
   - Sufficient time must be allowed for rescuers to exit the mine or atmosphere before the environment enters higher levels of risk.
MRAS has been designed to allow decision makers to assess and address these requirements in electronic format and produce a PDF report for acknowledgement with signature by the decision maker. The software also allows decision makers to directly access the explosibility analyses graph designed during this project (Figure 3).

![Figure 3 - Explosibility assessment tool](image1)

In an effort to equip decision makers with tools to assist in applying these criteria a relatively simple Microsoft Excel spreadsheet was set up to trend both the LEL and ONP with the ability to add trend lines. Figure 4 shows a plot of data collected for a sample location where it appears that the LEL will exceed the trigger levels before the ONP drops below its trigger. By using the trending forward feature in Microsoft Excel it is possible to predict when this would as shown in Figure 4.

![Figure 4 - Trend data with barometric pressure trend](image2)

Often gas results are influenced by barometric pressure so decision makers are aided if the plots include barometric pressure trends occur as shown in Figure 4 (Brady, et al., 2012).
The spreadsheet created features a summary page (Figure 5) that collates the data and displays latest results, trigger levels and TARP actions for each point. Also included is the overall status and trigger level which is the highest risk level of all points.

![Summary page](image)

**Figure 5 - Summary page**

### Authority to enter

Where it was deemed the risk to be acceptable for rescuers to enter the mine an “Authority to Enter” was completed in MRAS with a PDF report produced for and signed off by the Incident controller and NZMRS appointed person. This document formed part of the mine re-entry action plan along with other relevant information which included:

- Up to date mine plans;
- Current status;
- Relevant and current environmental readings and trends;
- Status of mine ventilation and mine services;
- Barometric pressure and trend;
- Known and unknown additional environmental conditions such as expected visibility and temperature, roadway conditions, water hazards and strata hazards.

The objective of an authority to enter was to ensure the instruction for mines rescue teams to enter the mine is unmistakable, deliberate and well informed.

Additional to the core functionality of information management, a structured decision making process for the assessment of explosibility risk and authority to enter, the MRAS software tool was also utilised for management of the status of general actions required of each functional area and also the maintenance of a record of the current situation throughout the operation.

The required actions put into the software appear in chronological order and allow the user to allocate actions to the responsible functional area with a due date and time and priority. The action on completion can be closed out with a date and time when completed along with any comments as shown in Figure 6.

A useful reporting function on the actions screen allows for a report to be generated showing only the incomplete actions grouped within the responsible functional area and in order of priority as shown in, Figure 7. Additionally the current situation throughout the operation could be maintained as regularly as required within a dedicated function within the MRAS software and the ability to generate an accompanying report.
Figure 6 - General actions management tool

Figure 7 - Incomplete actions report
CONCLUSIONS AND DISCUSSIONS

The researchers of ACARP project C19010 believe that due to the successful application of MRAS during the Pike River Mine re-entry operation (and Queensland level one emergency exercise) along with the level of interest and positive feedback from industry the outcomes of this project can be considered successful.

The standard of which the existing proof of concept model has been developed provides a solid foundation for realistic opportunities of commercialisation and implementation into a mine safety management system.

Although this ACARP project has come to a close the importance of the outcomes has been clearly recognised by industry with further support and momentum beyond this project now being provided by industry stakeholders with an aim of successful implementation.

Elements of industry have also recognised MRAS functionality, logic and processes have potential to greatly assist with the information management and decision making process for smaller more common incidents at a mine. E.g. Re-entry into a mine (or part of a mine) after an orderly evacuation when a mine site TARP has been exceeded.

When an incident occurs at a mine, decision makers must demonstrate proper diligence and take reasonable precautions. When a decision maker permits a rescuer to re-enter or remain in a mine after an incident, that person must ensure the risk to rescuers entering the mine after an incident is within acceptable limits and as low as reasonably achievable. The decision makers’ authorisation to re-enter the mine must be unmistakable, deliberate and well informed.

The researchers believe that the results of this ACARP project have significant potential to assist decision maker's to discharge their obligations during an emergency by assisting them to make well informed risk based decisions by taking reasonable precautions and demonstrating proper diligence.

REFERENCES


TRUCK-SHOVEL FLEET CYCLE OPTIMISATION USING GPS COLLISION AVOIDANCE SYSTEM

Benjamin D Knights\textsuperscript{1}, Mehmet S Kizil\textsuperscript{1} and Warren Seib\textsuperscript{2}

ABSTRACT: Truck-Shovel operations in surface mines involve high costs. Fleet management systems can provide a tool to improve fleet availability, utilisation and productivity, thereby reducing those costs. However, these systems are expensive to install. Stop-Watch time and motion studies provide a cheaper alternative. They can be undertaken on any segment of the haul cycle to provide accurate timing data, as well as observations on operator performance but are very time consuming and do not provide continuous monitoring of a fleet.

This paper provides an analysis of an alternative option; using a GPS collision avoidance system for truck-shovel fleet cycle optimisation. A case study was undertaken based on an operating mine in South-east Queensland using a commercially available GPS collision avoidance system. The approach was to use the GPS collision avoidance system to collect the truck positioning, speed, and timing data, which is automatically recorded as part of its normal function then to apply this information in a conventional time and motion study. This was combined with production loading data to provide some additional performance indicators. The methodology for using in a truck-shovel fleet cycle optimisation is discussed and the results from the case study are presented. Finally, the applicability of this approach is evaluated.

INTRODUCTION

Truck-shovel cycle optimisations are commonly performed to increase productivity, reduce costs and generally improve the profitability of the mobile assets at a mine. This task is most often performed by a fleet management system. These systems incorporate timing and positioning data, payload data, maintenance information, operator details and other pertinent information. They can perform detailed optimisation analysis with very little input from other systems or operators; however they are very expensive. Stop-watch studies provide a much cheaper alternative to a fleet management analysis. Using a stop-watch, accurate timing data and many field observations can be obtained for little to no upfront cost. However these studies can require extra inputs from other systems and are time consuming. Furthermore, such studies are conducted over a limited field and time of observation, not the full production system.

This paper looks at an alternative option to these two types of optimisation analysis; using a GPS Collision Avoidance System (CAS). These systems provide a data recording function that is used as part of their incident play-back feature. Data includes positioning, timing, speed, heading, number of satellites and alarms. A case-study into the effectiveness of such systems was undertaken at an operating coal mine in south-east Queensland. Presented in this paper are the results of that study, including the methodology and an evaluation of their applicability.

TIME AND MOTION STUDIES

The focus of a time and motion study is to provide feedback on a system’s performance by highlighting good and bad practices within the system that can then lead to overall improvements. The process is usually undertaken multiple times during the life of that system to continue the improvement of operational practices.

Methodology overview

Niebel (1988) generally accepted as the father of time and motion studies stated that undertaking a time and motion study should generally follow these steps:

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1. Get the facts;
2. Present the facts;
3. Make an analysis;
4. Develop the ideal method;
5. Present the method;
6. Install the method;
7. Develop a job analysis;
8. Establish time standards; and
9. Follow up the method.

Arzi (1997), who continued on from Niebel's work, compiled a set of steps that was more robust and applicable to this investigation, those being:

1. Selection of project and objectives definition;
2. Definition of measures for evaluating the design in view of the objectives;
3. Determination of the project limitation and freedom of action;
4. Data gathering and presentation;
5. Analysis of the data;
6. Development and presentation of alternative methods;
7. Evaluating alternative methods and selection of the best one;
8. Detailed design and presentation of the selected method;
9. Implementation of the designed methods (work place and method); and
10. Following up the method.

Both of these procedures cover five major components:

1. Definition;
2. Data collection;
3. Data analysis;
4. Evaluation; and
5. Implementation.

Another key component highlighted is the requirement to continually re-evaluate the processes; however this is not an actual step in the time and motion study itself.

CASE STUDY METHODOLOGY

Definition

A time and motion study should be well defined to have clear goals and maintain the focus of the investigation. The investigation undertaken, focused on the haul cycle of a selected fleet of coal trucks at one open cut mine. The haul cycle was broken down into separate tasks:

- Spot time at loader and dump;
- Travel time loaded and empty; and
- Dumping and loading times.
The delays in the cycle were also analysed, where possible. As this was a test case-study to analyse the capabilities of the system, the evaluation of the analysed data was left open, undertaking investigations where data limitations permitted.

**Data collection**

The data was collected on a fleet of five 136 t off-highway rear dump coal haul trucks. The data covered the period of time between 10:00 am on the 10/06/11 and 10:00 am on the 11/06/11, a twenty four hour period including three different shifts. During this time the trucks were operating from two different pits ("southern" and "central" pits) loading coal to the coal bins and ROM stockpile. Payload and maintenance information was also collected to assist in the evaluation step of the study.

**Data analysis**

The data analysis step is where the time study is undertaken. This is done by calculating the average time to perform a specific task, in this case a section of a haul cycle. This can be done by using the mean or mode of the sample and is called a time standard. Once the time standard for a task has been found, it can be used as a rating factor to highlight good and bad performances which are later evaluated.

For this analysis time standards for each task were calculated using Google Earth to interpret the downloaded file's traces. These standards times were then entered into and analysed using Microsoft© Excel™. The mean, median, mode, standard deviation and range were all found and a histogram plot of the distribution was produced to provide better understanding of the samples.

**Evaluation**

As outlined in the case-study definition, the scope of the evaluation was left completely open. Many evaluation methods and techniques were explored. Some of these options were ultimately rejected as data limitations prevented them from being viable. Some of those rejected were:

- Availability;
- Productivity; and
- Telemetry analysis.

Those that were used included: utilisation, TKPH analysis; and motion study analysis.

The utilisation, in general, is defined as the percentage of uptime hours that the truck was operating (Paraszczak, 2005). In this case uptime hours would refer to the available hours. The availability and therefore the available hours could not be calculated definitively, but as the maintenance reports indicated no scheduled maintenance or downtime, it was assumed that availability was 100%.

Tonne Kilometres Per Hour (TKPH) is a non-standard unit of measure for the load bearing capacity of a truck tyre (TAM, 2008). It can also be used as a form of productivity indicator when calculated for an entire truck rather than a single tyre. It accounts for not only the payload and time travelled, but also the distance travelled, therefore providing a more accurate result when evaluating loaded travel times over slightly different distances.

A motion study evaluates the different motions that are required to complete a task to identify why a certain task has been performed better or worse than the average. This can be as simple as spotting the differences or as complex as defining why a certain order of motions works better than another. In the case of a haul cycle this would focus on truck positioning and manoeuvring by the operator.

**Implementation**

The implementation step is undertaken after the analysis and evaluation is complete. It involves setting in place ways to ensure the good practices identified are used and bad ones are eliminated. As this is only a test case-study and the implementation is not impacted upon by the means in which the time and motion is undertaken, this step was not undertaken.
RESULTS

Time standards

The time standards were calculated for each section of the haul cycle as outlined and delays in the system were also highlighted. The time statistics were broken down into an average and optimal case. The optimal case is different in that it had significance or explainable outliers removed from the data set.

The travel time empty readings were split in two; one set for the central pit and one set for the southern pit. The statistics for the central pit are shown in Table 1 and a histogram of the sample is shown in Figure 1. The standard time for both the average and optimum case were identical for the central pit as no outliers were removed. The statistics for the southern pit are shown in Table 2 and a histogram of the optimum sample is shown in Figure 2. Six entries were removed from the optimum case in the southern pit due to the truck either travelling through central pit en-route to the southern pit or taking a detour through other areas of the mine.

Table 1 - Central pit travel time empty standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average/Optimum Case (s)</th>
</tr>
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<tbody>
<tr>
<td>Mean</td>
<td>331.86</td>
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<tr>
<td>Median</td>
<td>330</td>
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<tr>
<td>Mode</td>
<td>330</td>
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<tr>
<td>Standard Deviation</td>
<td>36.15</td>
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<td>Range</td>
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Table 2 - Southern pit travel time empty standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average Case (s)</th>
<th>Optimum Case (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>478.70</td>
<td>466.55</td>
</tr>
<tr>
<td>Median</td>
<td>461</td>
<td>457</td>
</tr>
<tr>
<td>Mode</td>
<td>470</td>
<td>423</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>61.35</td>
<td>42.94</td>
</tr>
<tr>
<td>Range</td>
<td>396 - 716</td>
<td>396 - 611</td>
</tr>
</tbody>
</table>

The travel time loaded readings were also split in two. The statistics for the central pit are shown in Table 3 and a histogram of the sample is shown in Figure 3. The standard times for both the average and optimum cases were identical for the central pit as no outliers were removed. The statistics for southern pit are shown in Table 4 and a histogram of the optimum sample is shown in Figure 4. Two entries were removed from the optimum case in the southern pit due to the truck taking a detour through other areas of the mine.
Figure 3 - Histogram of the central pit loaded travel times

Figure 4 - Histogram of the southern pit loaded travel times

Table 3 - Central pit travel time loaded standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average/Optimum Case (s)</th>
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<tbody>
<tr>
<td>Mean</td>
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<td>Median</td>
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<tr>
<td>Mode</td>
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<tr>
<td>Standard Deviation</td>
<td>28.29</td>
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<td>Range</td>
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</table>

Table 4 - Southern pit travel time loaded standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average Case (s)</th>
<th>Optimum Case (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>600.08</td>
<td>596.13</td>
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<tr>
<td>Median</td>
<td>589</td>
<td>588</td>
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<tr>
<td>Mode</td>
<td>551</td>
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</tr>
<tr>
<td>Standard Deviation</td>
<td>50.04</td>
<td>41.41</td>
</tr>
<tr>
<td>Range</td>
<td>528 - 817</td>
<td>528 - 706</td>
</tr>
</tbody>
</table>

Dumping time refers to both dumping at the ROM stockpile and at the coal bin. The statistics for dumping time are shown in Table 5 and a histogram of the optimum sample is shown in Figure 5. Five entries were removed from the optimum case due to the truck waiting on the coal bin to open before dumping. The waiting and dumping time could not be separated due to the truck being stationary the entire time so the results were removed.

Table 5 - Histogram of the dumping times

Table 5 - Dumping time standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average Case (s)</th>
<th>Optimum Case (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>29.78</td>
<td>27.90</td>
</tr>
<tr>
<td>Median</td>
<td>27</td>
<td>27</td>
</tr>
<tr>
<td>Mode</td>
<td>25</td>
<td>25</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>11.17</td>
<td>5.29</td>
</tr>
<tr>
<td>Range</td>
<td>19 - 104</td>
<td>19 - 47</td>
</tr>
</tbody>
</table>
The statistics for loading time are shown in Table 6 and a histogram of the optimum sample is shown in Figure 6. Seven entries were removed from the optimum case: four entries due to the truck being half loaded at one location, then moving and getting completely loaded at another; and three due to an operator change after loading was completed.

Table 6 - Loading time standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average Case (s)</th>
<th>Optimum Case (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>227.53</td>
<td>216.94</td>
</tr>
<tr>
<td>Median</td>
<td>219</td>
<td>219</td>
</tr>
<tr>
<td>Mode</td>
<td>243</td>
<td>243</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>108.60</td>
<td>41.7</td>
</tr>
<tr>
<td>Range</td>
<td>74 - 1136</td>
<td>127 - 348</td>
</tr>
</tbody>
</table>

The statistics for spot time at loader are shown in Table 7 and a histogram of the optimum sample is shown in Figure 7. Three points were removed from the optimal case. One point was removed due to the truck positioning itself ready to reverse during its queuing time, and the other two points were removed due to pausing several times during reversing waiting for the loader to be in position.

Table 7 - Spot time at loader standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average Case (s)</th>
<th>Optimum Case (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>28.67</td>
<td>28.35</td>
</tr>
<tr>
<td>Median</td>
<td>27</td>
<td>27</td>
</tr>
<tr>
<td>Mode</td>
<td>27</td>
<td>27</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>6.52</td>
<td>5.38</td>
</tr>
<tr>
<td>Range</td>
<td>14 - 58</td>
<td>19-50</td>
</tr>
</tbody>
</table>

The statistics for spot time at dump are shown in Table 8 and a histogram of the sample is shown in Figure 8. This sample set had many outliers removed, 29 in total. Two of these were removed due to multiple attempts to position the truck at the ROM stockpile. The remaining 27 were removed due to the truck waiting for the coal bin to be open/available for dumping. For this case an attempt was made to separate out the waiting time from the spot time. This proved difficult, however, as when the truck is
forced to wait it must come to a stop and then accelerate again. This additional stop adds extra time during spotting through acceleration and deceleration that cannot be calculated by simply removing the stationary time. Consequently these samples were removed.

![Spot Time at Dump Histogram](image)

**Figure 8 - Histogram of the spot times at dump**

**Table 8 - Spot time at dump standard time statistics**

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average Case (s)</th>
<th>Optimum Case (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>64.28</td>
<td>30.25</td>
</tr>
<tr>
<td>Median</td>
<td>31</td>
<td>29</td>
</tr>
<tr>
<td>Mode</td>
<td>29</td>
<td>29</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>86.02</td>
<td>5.52</td>
</tr>
<tr>
<td>Range</td>
<td>18-496</td>
<td>18-48</td>
</tr>
</tbody>
</table>

It can be seen in some of the samples that removal of outliers has a significant effect on the mean and standard deviation. In the loading and dumping time samples the standard deviation is halved and in spot time at dump it is reduced by more than 90%. The mean in the spot time at dump sample is also reduced by half. Those outliers that have a significant statistical impact on the time standards would also be contributing to large amounts of lost production time.

The histograms all show a similar distribution pattern; a regular or left skewed bell curve. The left skewed distributions are for those samples that have comparatively lower means (spot and dumping times). This is due to it being difficult to significantly go under the mean, whereas it is quite easy to exceed it. The loading time sample is somewhat different from the others. It is regular in shape but is also slightly bi-modal. This could be due to the loader not using a consistent number of bucket loads to fill each truck.

These results provide the basis for comparison of good and bad results, both within the set and in the future. They also provide a good starting data for planning and scheduling purposes.

**Delays**

There were two major sources of delays that could be calculated; queue time at loader and parked up time. The statistics for queue times are shown in Table 9 and a histogram of the sample is shown in Figure 9. As these are delays they are all counted in the average case.

![Queue Time at Loader Histogram](image)

**Figure 9 - Histogram of the queue times at loader**
Table 9 - Queue time at loader standard time statistics

<table>
<thead>
<tr>
<th>Statistic</th>
<th>Average Case (s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>199.12</td>
</tr>
<tr>
<td>Median</td>
<td>135</td>
</tr>
<tr>
<td>Mode</td>
<td>0</td>
</tr>
<tr>
<td>Standard Deviation</td>
<td>198.92</td>
</tr>
<tr>
<td>Range</td>
<td>0-936</td>
</tr>
</tbody>
</table>

The spread on this sample is not a bell curve, which is to be expected as most results are zero and cannot go negative. The most significant part of this sample is the mean. At 200 s per cycle this adds a large amount of lost time. In total 7 h and 11 min out of the 120 h of recording time was lost due to queuing, which is just under 6%. This is significant lost production.

A significant amount of parked up time was found throughout the recording period, on all trucks. The causes of these parks are not known but would mostly be caused by shift changes, crib breaks, lack of operator or planned downtime. However, all parked up time was assumed to be caused by operational delays as no maintenance downtime was found. Therefore, it was assumed that availability was 100%. There was a total of 51 h of lost time found due to park ups; this gave a utilisation of 57.5% based on the 120 h of recording. This is quite low and creates a large loss of production. This is however a relatively small sample set for the calculation of utilisation of a fleet and may not be a complete representation of the entire fleet.

Payload analysis

A payload analysis was performed to see what affects, if any, the variation of payload weights would have on loading times and loaded travel times. The payload data was directly compared to the loading times to see if there was a correlation between the two. None was found.

To evaluate the loaded travel times, the average speeds were plotted against payload carried for that trip. The scatter plot is a graphical means of showing the TKPH results whereby the higher results fall higher and to the right on the plot and lower results fall lower and to the left on the plot. It would be expected that if there was a correlation between payload and average speed that in would show up on this plot. The graph of the results can be seen in Figure 10. Also included on the plot is a breakdown of which truck and shift the recording took place and by extension when a different operator was on the machine. The three numeral code refers to a different truck number and the letters A, B and C refer to the first day shift, night shift and second day shift respectively.

The plot shows little to no correlation at all; so the conclusion would be that payload does not affect average speeds and thereby loaded travel times. By including the various operators a different correlation can be identified. The data points for each separate operator are somewhat grouped together indicating that a major factor in the loaded travel time performance may be the operator performance. There are however a number of outliers within sample 312A, 312B and 314B. It was found that these outliers all occurred during or close to shift changes and crib breaks. This could mean a different operator was on the truck for a hot seat change-out giving more weight to the theory that operator performance is a driving factor.

![Figure 10 - Southern pit TKPH analysis scatter plot](image-url)
Motion study

Motion study principles were applied to analyse the method by which operators spotted their trucks at both the loader and dump face. The analysis was first performed on the sample for truck 312 on the night shift as this had the largest sample size. A number of both very good and bad performances were found from the time standard calculations and compared to see if there were any significant differences in the method. Figure 11 shows two such cases at the coal bin and Figure 12 shows three cases from the loading face.

Each blue string represents a different attempt at spotting. The highlighted traces represent some of the best and worst cases from this sample. There were several key differences found between the red and green samples at the coal bin. The first was, the green case travels much closer to the coal bin before reversing than the red case, reducing the distance travelled in reverse. This is faster as the truck travels slower in reverse. The second was that the green case travels along a wider arc when approaching. This may seem like a slower option but on analysis it was found that taking the wider arc is quicker because the truck is able to travel at faster speeds. The third difference can’t actually be seen from the snapshots but was evident when replayed at real speed. In the red case the truck remained stationary for a few seconds longer when transitioning from forwards to reverse.

All three of these differences also applied to the cases at the loading face. However there was a fourth difference between the red and green case at the loader face. When the truck reversed in the red case it had to slow and redirect on approach to the loader. This added an additional three seconds, and was the only difference between the red and orange cases at the loader face.

Case-study performance review

As part of the review of the performance of the GPS CAS, it was compared to other methods of performing time and motion studies.

A stopwatch study is the simplest method of completing a time and motion study. It involves manually collecting timing data with a stopwatch and recording any other observations as they happen, either while on an operating truck or observing from a distance. The CAS had a number of benefits over this method. The biggest advantage is the CAS automatically records data and it records more information than can be obtained from simple field observations. It also removes almost all the time spent in the field. There is one disadvantage in that the collected data needs to be formatted before it can be analysed. However, the added time from this step is less than half the extra time required to collect data from a stopwatch.

From the analysis stage onwards both studies would be the same. Both would require external input for payload information or other telemetry data. The final difference is the price of the two studies. In terms of time cost, the CAS is better, but in terms of upfront costs there is no comparison. A stopwatch could cost around ten dollars whereas a GPS CAS costs hundreds of thousands of dollars. The cost component is somewhat irrelevant as many mines already have CAS’s installed for safety reasons, and
with the push for ever safer working environments it is likely that systems like these may become mandatory in the near future.

The second comparison is to a Fleet Management System (FMS) study. FMS's include systems like Caterpillar© VIMS and Leica© JigSaw360™. They encompass a large amount of hardware and software specifically designed to monitor and report on many aspects of operation. This gives this type of study an advantage in the amount and type of information that can be collected. They also have the advantage that most data is automatically recorded and reported in a form that can be readily analysed. Compared to the CAS this is a significant time saver. It comes at a cost, however, with these FMS's costing up to tens of millions of dollars depending on fleet sizes.

If costs are an issue and a CAS is already available for use, then in would provide a viable alternative to the time and motion study needs of a mine compared to stopwatch and FMS studies.

CONCLUSIONS

The test case-study produced a number of promising results. Time standards were calculated and distribution plots were produced for each segment of the haul cycle for use in planning purposes. No correlation was found between payloads and loading times, however operator performance was found to be a key driving factor behind the average speed readings for travelling loaded. Finally, four key factors were found to affect the spot times in the case study:

- The radius of curvature of the arc of approach;
- The distance from the dump/loader before reversing;
- The amount of time spent transitioning from forwards to reverse; and
- Maintaining good vision of the dump/loader while reversing to maintain speed in positioning.

On review of the test case-study it was found that the GPS CAS could provide a cheaper but more time consuming alternative to a FMS and would provide a much better all-round alternative than a stopwatch study.

REFERENCES


A RISK RATING SYSTEM FOR ANGLO AMERICAN’S OPEN CUT COAL MINES IN AUSTRALIA

John Hoelle¹ and Ismet Canbulat²

ABSTRACT: There are a number of risk rating systems used at the Anglo American Metallurgical Coal’s (AAMC) open cut coal mines in Australia. These systems are mainly mine site specific, geological based and the calculated risks are not comparable. Therefore, a uniform risk rating system, called OpenRisk is currently being evaluated and implemented for an unbiased, standard and quantifiable assessment of the risk from highwall and lowwall failures. This system is a semi-quantitative risk rating systems and takes into account the relative differences in the importance of hazards as experienced at each mine site as a result of different combinations of geotechnical factors and mining conditions. The system is based on critical geotechnical and other parameters that have been identified by site mining engineers, geologists and geotechnical engineers.

The primary advantage of this risk rating system is that all open cut mines in the AAMC operations use a near identical system, which enables the user to compare the risk with other mines. The system can be adjusted to meet local mine specific requirements.

The implementation of this system (a computer program that automatically calculates the risk) has been made as practical and as easy to use as possible. This program can be used by personnel from other mining disciplines not directly related to geotechnical engineering.

INTRODUCTION

Anglo American Metallurgical Coal operates five open cut operations located in Central Queensland and New South Wales (NSW). There is an increasing emphasis on safety and reliability at these operations. In addition, the Anglo American vision is to achieve “Zero Harm” through the effective management of safety at all businesses units and operations. In order to accomplish this vision, AAMC has implemented pro-active ground control management systems for a safe and effective production of open cut and underground reserves.

AAMC’s pro-active ground control management involves an understanding of the impacts of the geotechnical environment on likely ground behaviour and consists of various elements (Canbulat, 2010; Hoelle, 2010). The safety record of these mines has been remarkable over the years. However, there have been a few recent highwall failures, which caused loss of production and could have resulted in injury to personnel. In order to prevent these unexpected failures, AAMC has initiated a project to evaluate and implement a risk rating system, called OpenRisk, that was developed by Canbulat et al. (2004) for Anglo Coal South Africa. The input parameters and the controls used in the program have been modified to local conditions in order to ensure that the results are representative of the environment the open cuts operate in Australia. The ultimate aim of this implementation is to minimise the risk to personnel and machinery by identifying the risks and by recommending a set of generic controls. A summary of risk rating system and the modifications implemented in Australia is presented.

APPROACH IN RISK RATING SYSTEM

OpenRisk has two distinct components, namely, controllable parameters and uncontrollable parameters. An advantage of this is that the ground conditions (uncontrollable parameters) and the responses to those conditions (controllable parameters) can be rated separately. There are compelling reasons for these to be rated separately. For example, perfect ground conditions can be turned into a high-risk environment by applying inappropriate mining practice, or very hazardous ground conditions can be turned into low risk environment by applying good mining practice. Such separation in the ratings ensures that:

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² Principal Geotechnical Engineer, Undergrounds
• uncontrollable parameters are the true reflection of ground conditions present and cannot be changed;
• controllable parameters are the true reflection of the responses to those conditions and can be changed to reduce the overall risk.

The two ratings are, however, combined to produce an overall risk. The influence of changing a controllable factor on overall risk can be assessed using this methodology, thus evaluating the efficacy of modifications implemented.

An important consideration in OpenRisk is that uncontrollable parameters represent the magnitude of the inherent risks and is therefore called the Geotechnical Risk Rating. The controllable parameters represent the risk control factors applied in the open cut and are therefore called the Performance Rating. The combination of these two rating values represents the overall risk in the panel and is termed the Overall Risk Rating, Figure 1.

The parameters that form the basis of the risk rating system are drawn from systems previously used in Anglo American and hence they are based on local experience and knowledge. These parameters have been modified for the AAMC open cut operations.

![Flowchart used in the development of the risk rating system](image)

**ADVANTAGES OF THIS DUAL-RATING SYSTEM**

The advantages of this dual-rating system can be summarised as follows:

• easy to apply;
• does not require extensive training;
• the system provides an unbiased, standard quantified assessment of the risk from falls of ground, as the human factor is eliminated from the rating system;
• the likelihood of failure and stability can be assessed;
• consequences/severity of failure can be assessed;
• the risk or change in risk can be monitored over a period of time or face advance;
• controls/responses can be determined to reduce the risk;
• the performance of a crew can be determined over a period of time;
• the likelihood of failure can be assessed by implementation of controls; and
• if required, the ratings can be plotted on mine plans in real time.
PARAMETERS USED IN THE RISK RATING SYSTEM

OpenRisk methodology is common to all mines. It nevertheless allows for differences in the parameters rated and their weightings, according to mine specific experience and requirements. For example, while the effect of water may be a significant factor on the stability of highwalls in a certain mine, because of dry ground conditions, its effect on overall rating may be insignificant in another mine. Therefore, the weightings of each parameter are determined by the experienced mining personnel and geotechnical engineers from the various mines. The probability factor for each parameter is however kept constant.

Geotechnical risk rating parameters

In OpenRisk the geotechnical risk rating parameters are divided into four distinct categories, namely, geology, water, spontaneous combustion and dragline. The adjusted parameters and the probability factors used in the risk rating are presented in Table 1. This table shows that a total of 18 parameters are used in the geotechnical risk rating. Although all parameters are common to the systems used on all mines, manipulation of some of the parameters may however, be different for different mines.

<table>
<thead>
<tr>
<th>Table 1 - Parameters used in the geotechnical risk rating system</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.1</td>
</tr>
<tr>
<td>Depth of weathering</td>
</tr>
<tr>
<td>0 - 5 m</td>
</tr>
<tr>
<td>5 - 10 m</td>
</tr>
<tr>
<td>&gt; 10 m</td>
</tr>
<tr>
<td>1.2</td>
</tr>
<tr>
<td>Discontinuities</td>
</tr>
<tr>
<td>None</td>
</tr>
<tr>
<td>1 (simple)</td>
</tr>
<tr>
<td>2.2</td>
</tr>
<tr>
<td>Direction of discontinuities</td>
</tr>
<tr>
<td>Not applicable</td>
</tr>
<tr>
<td>2.3</td>
</tr>
<tr>
<td>Same direction (&lt;30 deg.)</td>
</tr>
<tr>
<td>Different direction (&gt;30 deg.)</td>
</tr>
<tr>
<td>1.4</td>
</tr>
<tr>
<td>Dipping structure / bedding</td>
</tr>
<tr>
<td>Flat/dipping into the face</td>
</tr>
<tr>
<td>Dipping into the cut</td>
</tr>
<tr>
<td>1.5</td>
</tr>
<tr>
<td>Clay material in bedding</td>
</tr>
<tr>
<td>NO</td>
</tr>
<tr>
<td>YES</td>
</tr>
<tr>
<td>2.6</td>
</tr>
<tr>
<td>1.6</td>
</tr>
<tr>
<td>Length of structure</td>
</tr>
<tr>
<td>0 - 1 m</td>
</tr>
<tr>
<td>1 - 5 m</td>
</tr>
<tr>
<td>&gt; 5 m</td>
</tr>
<tr>
<td>1.7</td>
</tr>
<tr>
<td>Presence of floor rolls and dipping seam</td>
</tr>
<tr>
<td>NO</td>
</tr>
<tr>
<td>YES</td>
</tr>
<tr>
<td>1.8</td>
</tr>
<tr>
<td>Major dykes/faults/burnt coal</td>
</tr>
<tr>
<td>NO</td>
</tr>
<tr>
<td>YES</td>
</tr>
<tr>
<td>YES</td>
</tr>
</tbody>
</table>

Discussions with geotechnical engineers and mining personnel revealed that certain mines require certain parameters in their rating system, while other mines do not require those parameters. For this reason, a “not applicable” option is also introduced in OpenRisk. In such cases, the parameter is taken out of the calculations.

PERFORMANCE RISK RATING PARAMETERS

Performance parameters are divided into three distinct categories, namely, geometry, mining and blasting. The parameters used in the risk rating are presented in Table 2. A total of 17 parameters are used in the performance risk rating. Similar to geotechnical risk rating, a “not applicable” option is also introduced in the performance rating for three parameters.
Table 2 - Parameters used in the performance risk rating system

<table>
<thead>
<tr>
<th>1) GEOMETRY</th>
<th>1.8 Noses present</th>
<th>1.9 Loose blocks at crest</th>
</tr>
</thead>
<tbody>
<tr>
<td>Batter back soft/weathered material</td>
<td>1</td>
<td>NO</td>
</tr>
<tr>
<td>Not Applicable</td>
<td>1</td>
<td>YES</td>
</tr>
<tr>
<td>Yes / minimum 50 deg.</td>
<td>10</td>
<td>20</td>
</tr>
<tr>
<td>No / more than 50 deg.</td>
<td>20</td>
<td>NO</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Height of highwall</th>
<th>2.1 Undercutting spoils</th>
</tr>
</thead>
<tbody>
<tr>
<td>0 - 35 m</td>
<td>1</td>
</tr>
<tr>
<td>35 - 50 m</td>
<td>5</td>
</tr>
<tr>
<td>50 - 70 m</td>
<td>10</td>
</tr>
<tr>
<td>&gt; 70 m</td>
<td>20</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Angle of highwall</th>
<th>2.2 Undercutting highwall</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt; 65 deg.</td>
<td>1</td>
</tr>
<tr>
<td>65 - 75 deg.</td>
<td>5</td>
</tr>
<tr>
<td>&gt; 75 deg.</td>
<td>10</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Top bench width</th>
<th>2.3 Spois in water</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt; 10 m</td>
<td>1</td>
</tr>
<tr>
<td>0 - 10 m</td>
<td>5</td>
</tr>
<tr>
<td>No bench</td>
<td>10</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Spois on the highwall</th>
<th>2.4 Spoiling of weathered material at toe of spoils</th>
</tr>
</thead>
<tbody>
<tr>
<td>Not applicable</td>
<td>0</td>
</tr>
<tr>
<td>&lt;15 m high/&gt;10 m from crest</td>
<td>1</td>
</tr>
<tr>
<td>&lt;15 m high/&gt;5 m from crest</td>
<td>3</td>
</tr>
<tr>
<td>&gt;15 m high/&gt;10 m from crest</td>
<td>5</td>
</tr>
<tr>
<td>&gt;15 m high/&gt;5 m from crest</td>
<td>10</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Height of spoils on lowwall</th>
<th>3.1 Blasting method of highwall</th>
</tr>
</thead>
<tbody>
<tr>
<td>Not applicable</td>
<td>Pre-split</td>
</tr>
<tr>
<td>0 - 40 m</td>
<td>Not pre-split</td>
</tr>
<tr>
<td>40 - 90 m</td>
<td>Highwall condition due to blasting</td>
</tr>
<tr>
<td>&gt; 95 m</td>
<td>Frozen coal, overhangs, loose material</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>3.2 Highwall condition due to blasting</th>
</tr>
</thead>
<tbody>
<tr>
<td>Straight HW no loose material</td>
</tr>
<tr>
<td>Straight highwall, some loose material</td>
</tr>
<tr>
<td>Frozen coal, overhangs, loose material</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>1.7 Cut width (deviation from standard)</th>
<th>3.3 Pre-split barrels</th>
</tr>
</thead>
<tbody>
<tr>
<td>Standard within 5 m</td>
<td>Visible</td>
</tr>
<tr>
<td>Not standard (&gt; 5 m deviation)</td>
<td>Visible</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Blast holes</th>
</tr>
</thead>
<tbody>
<tr>
<td>Visible</td>
</tr>
<tr>
<td>Not Visible</td>
</tr>
</tbody>
</table>

WEIGHTINGS OF PARAMETERS

As is known that different parameters do not have the same effect on the overall panel rating system. The presence of one parameter may have a significant effect on the risk, while another parameter can have only a minor effect. It is therefore necessary to determine the weightings for each parameter in the rating system to ensure safe workings; each parameter is rated against the potential severity of the consequence. The weights of each parameter used in the geotechnical and performance ratings are presented in Table 3 and Table 4, respectively. In these Tables, “1” represents the lower severity, while “20” represents the highest severity.

CONTROLS

Introduction of controls, which are the actions to be taken for a given condition or risk level, can be implemented separately in the rating systems for different mines. These controls are automated to ensure that the risks can be negated almost immediately. However, the controls can also be introduced by the user in “Special Instruction” text boxes.

Preliminary lists of controls for different parameters in the geotechnical and performance ratings are shown in Table 5 and 6 respectively. It is however recommended that the controls should be reviewed and updated regularly to ensure the latest geotechnical engineering and local knowledge is available to the user.

CALCULATION OF LIKELIHOOD OF FAILURE AND SEVERITY

The following mathematical models are used in calculation of probability of failure (LoF) and severity (Sev) for both geotechnical and performance ratings:

\[ \text{LoF} = \frac{\sum_{i=1}^{n} SPF_i}{\sum_{i=1}^{n} MPF_i} \] (1)
Where:

\[ SPF_i = \text{Selected probability factor for each parameter}; \]
\[ MPF_i = \text{Maximum of probability factor of each parameter}; \]
\[ W_i = \text{Weighting of each parameter}. \]

Table 3 - Weightings of parameters used in the geotechnical risk rating

<table>
<thead>
<tr>
<th>Parameter Description</th>
<th>Weighting</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.1 Depth of weathering</td>
<td>5</td>
</tr>
<tr>
<td>1.2 Discontinuities</td>
<td>20</td>
</tr>
<tr>
<td>1.3 Direction of discontinuities</td>
<td>20</td>
</tr>
<tr>
<td>1.4 Dipping structure / bedding</td>
<td>20</td>
</tr>
<tr>
<td>1.5 Clay material in bedding</td>
<td>1</td>
</tr>
<tr>
<td>1.6 Length of structure</td>
<td>20</td>
</tr>
<tr>
<td>1.7 Presence of floor rolls and dipping seam</td>
<td>1</td>
</tr>
<tr>
<td>1.8 Major dykes/faults/burnt coal</td>
<td>10</td>
</tr>
<tr>
<td>1.9 Cracks on highwall/benches within 10 m of crest</td>
<td>20</td>
</tr>
<tr>
<td>1.10 Highwall condition</td>
<td>10</td>
</tr>
<tr>
<td>2.1 Water coming out of face bedding or structure</td>
<td>10</td>
</tr>
<tr>
<td>2.2 Is there water accumulation at toe of slope</td>
<td>1</td>
</tr>
<tr>
<td>2.3 Is there water on top of highwall/benches within 30m of crest</td>
<td>1</td>
</tr>
<tr>
<td>2.4 Rain</td>
<td>5</td>
</tr>
<tr>
<td>2.5 Head of water</td>
<td>1</td>
</tr>
<tr>
<td>3.1 Is the toe of highwall burning</td>
<td>1</td>
</tr>
<tr>
<td>3.2 Is the toe of lowwall/spoil or any layer burning</td>
<td>1</td>
</tr>
</tbody>
</table>

Table 4 - Weightings of parameters used in the performance rating

<table>
<thead>
<tr>
<th>Parameter Description</th>
<th>Weighting</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.1 Batter back soft/weathered material</td>
<td>20</td>
</tr>
<tr>
<td>1.2 Height of highwall</td>
<td>10</td>
</tr>
<tr>
<td>1.3 Angle of highwall</td>
<td>10</td>
</tr>
<tr>
<td>1.4 Top bench width</td>
<td>10</td>
</tr>
<tr>
<td>1.5 Spoils on the highwall</td>
<td>1</td>
</tr>
<tr>
<td>1.6 Height of spoils on lowwall</td>
<td>5</td>
</tr>
<tr>
<td>1.7 Cut width (deviation from standard)</td>
<td>1</td>
</tr>
<tr>
<td>1.8 Noses present</td>
<td>10</td>
</tr>
<tr>
<td>1.9 Loose blocks at crest</td>
<td>10</td>
</tr>
<tr>
<td>2.1 Undercutting spoils</td>
<td>20</td>
</tr>
<tr>
<td>2.2 Undercutting highwall</td>
<td>20</td>
</tr>
<tr>
<td>2.3 Spoils in water</td>
<td>1</td>
</tr>
<tr>
<td>2.4 Spoiling of weathered material at toe of spoils</td>
<td>1</td>
</tr>
<tr>
<td>3.1 Blasting method of highwall</td>
<td>1</td>
</tr>
<tr>
<td>3.2 Highwall condition due to blasting</td>
<td>1</td>
</tr>
<tr>
<td>3.3 Pre-split barrels</td>
<td>1</td>
</tr>
<tr>
<td>3.4 Blast holes</td>
<td>10</td>
</tr>
</tbody>
</table>
Table 5 - Controls for the geotechnical rating parameters

<table>
<thead>
<tr>
<th>1) GEOLOGY</th>
<th>Actions/Instructions</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.1 Depth of weathering</td>
<td>Batter, bench to hard, if it is soil batter to 45 deg., if it is weathered material batter to 60 deg. Conduct stability analysis; evaluate pre-strip</td>
</tr>
<tr>
<td>1.2 Discontinuities</td>
<td>Increase awareness of jointing. Conduct kinematic stability analysis;</td>
</tr>
<tr>
<td>1.3 Direction of discontinuities</td>
<td>Increase awareness of joint orientation. Conduct kinematic stability analysis;</td>
</tr>
<tr>
<td>1.4 Dipping structure / bedding</td>
<td>Increase awareness of dip of jointing. Conduct kinematic stability analysis;</td>
</tr>
<tr>
<td>1.5 Clay material in bedding</td>
<td></td>
</tr>
<tr>
<td>1.6 Length of structure</td>
<td></td>
</tr>
<tr>
<td>1.7 Presence of floor rolls and dipping seam</td>
<td>Determine the dip of the strata. Install monitoring.</td>
</tr>
<tr>
<td>1.8 Major dykes/faults/burnt coal</td>
<td></td>
</tr>
<tr>
<td>1.9 Cracks on highwall/benches within 10 m of crest</td>
<td>Notify management and Geotechnical Engineering Department immediately. Install monitoring. Haul routes to be moved. Barricade the area. Ensure no equipment or men at the H/W.</td>
</tr>
<tr>
<td>1.10 Highwall condition</td>
<td>Increase the exclusion zone to 15 m</td>
</tr>
</tbody>
</table>

2) WATER

| 2.1 Water coming out of face bedding or structure | Pump water and monitor the slope. |
| 2.2 Is there water accumulation at toe of slope | Pump water and monitor the slope. |
| 2.3 Is there water on top of highwall/benches within 30 m of crest | Divert water and pump water out. |
| 2.4 Rain | Monitor the slope. Pump standing water, if required. Slope may be affected up to 5 days after rain, therefore keep awareness high. |
| 2.5 Head of water | |

3) SPONCOM

| 3.1 Is the toe of highwall burning | Sand dress the slope. Use water canons. |
| 3.2 Is the toe of lowwall/spoil or any layer burning | Sand dress the slope. Use water canons. |

3) DRAGLINE

| 4.1 | Dragline bench built on |

FINAL RISK RATING AND RISK CATEGORIES

The risk categories for geotechnical and performance ratings as well as overall rating are calculated by using the chart in Figure 2. The probability of failure and the severity are plotted in this figure and the risk areas for low, medium and high are determined with two linear lines. These lines are drawn based on a back analysis of failures and experienced gained from numerous different highwalls in South Africa and Australia. Although, it is not recommended to adjust these lines for different mines, they can be adjusted, using a back analysis of past failures.

![Figure 2 - Overall risk category chart](image-url)
Table 6 - Controls for the performance rating parameters

<table>
<thead>
<tr>
<th>1) GEOMETRY</th>
<th>Actions/Instructions</th>
</tr>
</thead>
<tbody>
<tr>
<td>1.1 Batter back soft/weathered material</td>
<td>High hazard area, batter back, if possible. Mark &amp; barricade off. No people, equipment or machinery under the H/W. Batter to 50-degrees</td>
</tr>
<tr>
<td>1.2 Height of highwall</td>
<td>Increase the exclusion zone to 15 m. Conduct stability analysis</td>
</tr>
<tr>
<td>1.3 Angle of highwall</td>
<td>Ensure dragline digs a straight H/W. Check the blasting practice. Review design parameters. Review mining procedure</td>
</tr>
<tr>
<td>1.4 Top bench width</td>
<td>All crests should have a minimum 10 m bench</td>
</tr>
<tr>
<td>1.5 Spoils on the highwall</td>
<td>All crests should have a minimum 10-metre bench; review design and mining procedure</td>
</tr>
<tr>
<td>1.6 Height of spoils on lowwall</td>
<td>Check dragline spoiling. Check cut width. Ensure spoil is not undercut. Extended bench may be required. Conduct stability analysis</td>
</tr>
<tr>
<td>1.7 Cut width (deviation from standard)</td>
<td>Spooling room may be an issue. Extended bench may be required. Review 3D-Dig. Cut correct pit width. May require coal safety berm at least 20 m wide.</td>
</tr>
<tr>
<td>1.8 Noses present</td>
<td>High-risk area. Install monitoring. Initiate better scaling practices</td>
</tr>
<tr>
<td>1.9 Loose blocks at crest</td>
<td>Make/extend the exclusion zone at the toe of H/W to 15 m and ensure no people, equipment or machinery in the area. Monitor the area. Work under supervision. Scale if possible</td>
</tr>
<tr>
<td>2) MINING</td>
<td></td>
</tr>
<tr>
<td>2.1 Undercutting spoils</td>
<td>Stop undercutting the spoils. Barricade the area. Install monitoring. Review mining procedure</td>
</tr>
<tr>
<td>2.2 Undercutting highwall</td>
<td>Stop undercutting the spoils. Barricade the area. Install monitoring. Review mining procedure</td>
</tr>
<tr>
<td>2.3 Spoils in water</td>
<td>Pump the water. Practice should be that spoil should never be dumped or shot into water.</td>
</tr>
<tr>
<td>2.4 Spoiling of weathered material at toe of spoils</td>
<td>Extended bench may be required. Double handle weathered material or mix with fresh O/B before spoiling. Review mining sequence to minimise placement of weak material at base of spoil</td>
</tr>
<tr>
<td>3) BLASTING</td>
<td></td>
</tr>
<tr>
<td>3.1 Blasting method of highwall</td>
<td>Review blasting design &amp; applicability to conduct pre-split on all highwalls and endwalls</td>
</tr>
<tr>
<td>3.2 Highwall condition due to blasting</td>
<td>Scale if possible. Review blast design and applicability.</td>
</tr>
<tr>
<td>3.3 Pre-split barrels</td>
<td>Review blast design and applicability.</td>
</tr>
<tr>
<td>3.4 Blast holes</td>
<td></td>
</tr>
</tbody>
</table>

CONCLUSIONS

The risk rating system has been used on the Anglo American open cast coal operations in South Africa since June 2004 and is currently being trialled in Australia. Back analyses of the past instabilities indicated that while failing highwalls were rated and found to be high risk, stable highwalls were found to be low risk.

These initial results indicated that the risk system was consistent with reality and could be “trusted” to provide relative assessments of the open pits.

This risk rating system is primarily aimed at reducing the risk on the AAMC’s open cut coal operations and ensuring the rock/slope management strategy, as laid out in the Principal Hazard Management Plans and the Code of Practice. It is envisaged that OpenRisk will empower the employees on the operations to determine the risk and assists them in making quality decisions to determine the appropriate controls for these risks.

REFERENCES


FAULT AND FRACTURE ZONE DETECTION BASED ON SOIL GAS MAPPING AND GAMMA RAY SURVEY AT THE EXTENSION SITE OF AN OPEN PIT COAL MINE

Ye Ma1, Detlef Bringemeier2, Alexander Scheuermann, Tiro Molebatsi and Ling Li

ABSTRACT: Identification of open active faults and fracture zones is a part of exploration study prior to mining operation. However, detailed mapping of geological discontinuities in an otherwise low permeable overburden is rarely carried out in the mining area. To develop a rapid and feasible survey method, a field campaign was conducted to examine different soil gas survey methods along three transects at the Carrington West Wing extension site of a coal mine, Hunter River Valley, NSW, Australia. Coal seam gas together with Uranium-238 (present in the gas-bearing coal seam) increases the soil gas signal which can be detected with suitable soil gas mapping methods. Three techniques associated with four parameters were tested at the field site. A conventional active soil gas sampling method was applied with the samples analysed off-site in the lab by gas chromatography for carbon dioxide and methane concentrations. Radon was measured on site by means of radon detector. It was expected that high soil gas concentration anomalies, if detected, could then be related to the locations of permeable fault/fracture zones. A rapid and simple technique was used to determine the relative counts of Bismuth-214 in the soil surface by employing a gamma ray spectrometer. As a decay product of the $^{222}\text{Rn}$, $^{214}\text{Bi}$ is also expected to exhibit relatively higher activities in the soil over faults and fracture zones.

INTRODUCTION

Open-cut is a commonly applied mining method in the Hunter River Valley and other Australian coal mining regions. By removing the overburden above the coal seam, this surface mining operation creates a significant drainage potential for the surrounding environment, like the adjacent river and the groundwater systems. The NSW Office of Water proposed a new NSW aquifer interference policy to protect the groundwater system, because of the significant growth of the coal and coal seam gas industries (The NSW Office of Water, 2011). Most previous studies focussed on the shallow aquifer investigation, without further study of the fracture flow in the rock overburden in the Hunter River Valley (Mackie, 2010). However, the mining impacts on the adjacent river and groundwater systems are likely to be controlled predominately by preferential flow zones provided by faults, fracture and coal seam cleats (López and Smith, 1995). Therefore, how to characterise hydraulic connectivity of faults/fractures with a river and alluvial aquifers and how to locate these permeable structures are becoming very critical questions for the groundwater risk assessment and mine water management. In this context, the non-invasive soil gas mapping method and gamma ray spectrometer reconnaissance are discussed and examined for the purpose of detecting the location of structures.

Soil gas mapping is based on measurements of the gases contained in the interstitial spaces of the soil above the water table and capillary fringe. Soil gases, including methane, carbon dioxide and radon, are sampled and analysed. Thermal methane is formed as part of the process of coal formation - coalification. It is released as a result of natural erosion or faulting. Because methane is highly mobile, buoyant and almost insoluble in groundwater, the fault system may act as the conduit, allowing methane to migrate to the soil layer. The carbon dioxide in the soil may originate from the mantle degassing, carbonate dissolution, organic material oxidation and plant breathing (Baubron, et al., 2002). Carbon dioxide is heavier than air and less volatile than most gases, it is accumulated in soil layer forming stable, well defined anomalies. As a decay product of uranium-238, radon ($^{222}\text{Rn}$) is present in relatively high concentrations in uranium-rich rock, such as carboniferous mudstone and coal seam. $^{222}\text{Rn}$ is an inert, radioactive gas with a half-life of 3.82 days. Short half-life of $^{222}\text{Rn}$ restrains its transport in subsurface, so that radon gas generated from a deep origin cannot reach the ground surface unless there exists a preferential flow pathway. Although all shallow soil gas concentration is affected by meteorological

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2 Golder Associates, Toowong, QLD 4066, Australia
factors (Hinkle, 1994), meteorological variations do not appear to affect anomalous soil gas concentration measurements during sample collecting within a period of a few days, unless there are rainfall events (Margaret, 1991). Rainfall affects soil gas concentration more than any other parameters at all sites. No samples are recommended to be taken for at least two days after a heavy rain because of the downward flushing of soil gas in pore spaces, which decrease soil gas concentrations. However, a small amount of rainfall increase soil gas concentration, as an impermeable barrier is formed at ground surface and soil gas is trapped (Hinkle, 1994). For the site investigation presented here, a depth of 1 m soil gas sampling is chosen in order to reduce the meteorological effects. Furthermore, soil gas survey was conducted over a short period of a relatively dry weather condition.

Another technique used for the investigation is the gamma ray spectrometer. $^{214}\text{Bi}$ is the decay product of $^{222}\text{Rn}$. The dominant gamma rays from $^{214}\text{Bi}$ are more intense in number and higher in energy than the other $^{222}\text{Rn}$ decay products. As a strong gamma ray emitter, $^{214}\text{Bi}$ is the first radioelement in uranium-238 decay series emitting gamma ray, which could be detected by ground survey (Griffiths, et al., 2010). $^{214}\text{Bi}$ is also a major indicator for the estimation of uranium concentrations in rock/soil by the gamma ray spectrometry detection method (IAEA, 2003). Because of the positive correlation between radon and $^{214}\text{Bi}$, it is expected that $^{214}\text{Bi}$ also exhibit relatively higher activities in the soil over the potential fault areas (LaBrecque, et al., 2004). Therefore, a gamma ray spectrometer is applied to measure the $^{214}\text{Bi}$ as a simple and rapid technique.

Subsequently, the field investigation was conducted to combine soil gas mapping and gamma ray survey for the detection and characterization of the fault zones.

**STUDY AREA**

The field site is located at 24 km North West of Singleton, NSW, Australia (S32.49491°, E150.95169°). The site consists of low undulating slopes and flat lying area with land surface elevations of about 70 m Australian Height Datum (AHD) over most of the floodplain area. A number of monitoring bores provide valuable information of groundwater hydrology. The site is mainly underlain by unconsolidated paleochannel sediments of gravel, silts and clays. The thickness of the alluvial sediments varies from 10 m to 20 m. Soil gas mapping and gamma ray survey were conducted in areas of three different soil types, as shown in Figure 1. The measurement procedure was tested for different soil conditions.

Figure 1 - Satellite image of the top view of the field site

Red area is covered by duplex loam. The brown part is uniform silty clay. The pink area is underlain by uniform silty clay loam (GSS Environmental, 2010). The 10 dots show the soil sampling wells. The two yellow lines are inferred faults crossing the site. The solid black lines are the three transects where soil gas mapping was carried out. Each solid black line is 100 m long with five soil gas measuring points (25 m interval). The blue line is around 1 000 m long, covering all three individual transects. Gamma ray survey was carried out along the blue line with a 10 m interval.
The site is underlain by Permian coal measure strata comprising among others from top down the Vaux, Brookie and Bayswater seams. The interburden comprises sandstone, siltstone and shale (Mackie, 2010). Two inferred fault structures are striking through the site (Figure 1). Those inferred faults increase the likelihood of methane, carbon dioxide and $^{222}$Rn migrating through permeable fractures to the ground surface.

**Sampling and analysis**

In our investigation, conventional active soil gas sampling methods were adopted. The entire sampling procedure is affected by many factors, which could lead to operational errors, such as ambient air intruding the sampling train and soil gas bypassing flow through the annular gap. Three tentative tests were conducted to aim at lessening operational errors prior to mobilizing in the field (Department of Toxic Substance Control, 2010).

*Equilibrium time test:* Soil gas conditions were disturbed during the probe emplacement. To allow the soil gas to equilibrate to the initial condition, a waiting time between the soil gas probe emplacement and soil gas sampling must be included. This equilibrium waiting time was tested.

*Shut-in test (valves, lines, fitting):* Prior to purging and sampling, shut-in test were conducted to check the leaks from valves, lines and fittings (above ground portion of sampling train). To evaluate the leaks, a vacuum pump was employed to vacuum the closed tube line. If there is any observable loss of vacuum, the sampling train needs to be redesigned or reconnected.

*Purge volume test:* The purpose of the purge volume test was to ensure that the ambient and stagnant air was purged out from the sampling system so that samples collected were of representative soil gas conditions.

The active soil gas sampling procedure was divided into three steps:

*Soil gas probe emplacement:* The probe was placed with the sampling tube into 1 m soil depth (Figure 2). A sealant of hydrated bentonite and air isolation packer were applied to seal off the annular gap around the probe. The hydrated bentonite was prepared by mixing one portion of powder bentonite with four portions of water, giving a gel-like end product.

*Soil gas sampling procedure:* 28 ml soil gas samples were taken by syringe. Soil gas samples were injected into the vacuum Labco exetainer. The volume of the vacuum exetainer is 12 ml. The more volume of gas is injected, the higher exetainer inner pressure is gained. If a leakage occurred, the gas would leak from inside to outside instead of air coming into the exetainer. This procedure prevents ambient air from entering the vacuum exetainer to dilute the soil gas sample. To check the reproducibility, replicate samples were taken for each point. The samples stored in the glass exetainers were analysed off-site in the lab by gas chromatography.

*Helium tracer gas application:* The most common errors of soil gas sampling are due to leakages along the sampling tube where the tube is in contact with the soil (ring gap) (Department of Toxic Substance Control, 2010). A tracer gas method was developed to check the effectiveness of the sealant, which was applied to avoid the breakthrough of air down to the probe, as shown in Figure 2.

---

**Figure 2 - Schematic of the tracer gas method**
The shroud was manufactured with three screw fixing rubbers on top, which were expected to stop bypassing leakage at the connecting points. The volume of the shroud was around 11.5 L. The helium gas cylinder (GOREGAS) was filled by 99.9% helium.

Helium tracer gas testing was conducted in the following steps:

1) Place the enclosure shroud over the probe on the ground;
2) Pull the sampling tube out of the shroud;
3) Seal the bottom shroud with hydrated bentonite;
4) Connect helium gas cylinder to the shroud, and open the relief valve;
5) Slowly inject the tracer gas into the shroud, and monitoring the helium concentration at the port of the relief valve with hand-held helium detector (GasCheck G3);
6) When the helium concentration approximately reach 0.2%, stop the tracer gas injection, and close the relief valve;
7) Take the 28 ml soil gas sample with syringe;
8) Helium concentration is measured by the pin detector at the tail-end of the sampling tube.

Finally, the dilution factor is calculated with the soil gas helium concentration divided by the shroud helium concentration. If the dilution factor is less than 5%, the samples are relatively acceptable (Department of Toxic Substance Control, 2010). Otherwise, the soil gas needs to be re-sampled.

**Radon measurements**

Measurements of $^{222}$Rn concentrations in the soil gases were carried out using the RAD7 portable radon detector (Durridge, USA). It contains a solid-state silicon alpha detector and a built-in pump with a flow rate of around 1 l/min. The inlet filter blocks fine dust particles and radon daughters entering the RAD7 testing chamber. The soil gas measurements are carried out in the sniff mode, which calculates $^{222}$Rn concentrations from the data in window A only. The sniff mode covers the energy range from 5.40 to 6.40 MeV, showing the total counts from 6.00 MeV alpha particles of the $^{218}$Po decay (daughter of $^{222}$Rn).

**$^{214}$Bi measurement**

The $^{214}$Bi measurements were made by the uranium channel of the GR-320 differential gamma ray spectrometer (Terraplus, Canada). The channels and energy ranges are shown in Table 1. The 0.35l NaI detector was placed vertical on the soil surface with a counting period of 100 seconds. Vegetation and small stones were removed before the measurement. The spectra were recorded and the Noise-Adjusted Singular Value Decomposition (NASVD) method was applied to process the spectrum.

<table>
<thead>
<tr>
<th>Window of interest</th>
<th>Potassium</th>
<th>Uranium</th>
<th>Thorium</th>
</tr>
</thead>
<tbody>
<tr>
<td>Nuclide</td>
<td>$^{40}$K</td>
<td>$^{214}$Bi</td>
<td>$^{208}$Ti</td>
</tr>
<tr>
<td>Channel range</td>
<td>111-126</td>
<td>133-149</td>
<td>192-223</td>
</tr>
<tr>
<td>Energy(MeV) range</td>
<td>1.365-1.557</td>
<td>1.647-1.854</td>
<td>2.420-2838</td>
</tr>
</tbody>
</table>

**Results and discussions**

**Results of soil gas mapping and gamma ray survey**

Soil gas concentrations are shown in Table 2. The range of CH$_4$ concentration is relatively low, maybe because of the coal seam methane degassing in/from the adjacent open-cut pit. The average dilution factor of radon is higher than those of the CO$_2$ and CH$_4$, due to more soil gas pumping in the course of radon measurement.
Table 2 - Three transects’ soil gas concentrations for CO₂, CH₄ and Rn. DF1 is the dilution factor for CO₂ and CH₄, DF2 shows the dilution factor of Rn. All presented data have been corrected based on the dilution factor.

<table>
<thead>
<tr>
<th>Location</th>
<th>CO₂ (%)</th>
<th>CH₄ (ppm)</th>
<th>DF1</th>
<th>Rn (kBq/m³)</th>
<th>DF2</th>
</tr>
</thead>
<tbody>
<tr>
<td>A1</td>
<td>0.18</td>
<td>1.43</td>
<td>2.70%</td>
<td>0.04</td>
<td>15%</td>
</tr>
<tr>
<td>A2</td>
<td>0.14</td>
<td>2.38</td>
<td>3.60%</td>
<td>3.86</td>
<td>31%</td>
</tr>
<tr>
<td>A3</td>
<td>0.95</td>
<td>0.68</td>
<td>0.24%</td>
<td>20.58</td>
<td>2.70%</td>
</tr>
<tr>
<td>A4</td>
<td>0.61</td>
<td>2.72</td>
<td>0.71%</td>
<td>0.09</td>
<td>16.60%</td>
</tr>
<tr>
<td>A5</td>
<td>1.58</td>
<td>0.20</td>
<td>0.78%</td>
<td>18.77</td>
<td>0.75%</td>
</tr>
<tr>
<td>B1</td>
<td>1.65</td>
<td>0.19</td>
<td>9.41%</td>
<td>28.14</td>
<td>1.48%</td>
</tr>
<tr>
<td>B2</td>
<td>1.17</td>
<td>0.42</td>
<td>0.76%</td>
<td>26.83</td>
<td>5.90%</td>
</tr>
<tr>
<td>B3</td>
<td>1.35</td>
<td>0.26</td>
<td>1.70%</td>
<td>36.76</td>
<td>4.30%</td>
</tr>
<tr>
<td>B4</td>
<td>1.57</td>
<td>0.25</td>
<td>1.40%</td>
<td>21.40</td>
<td>1.74%</td>
</tr>
<tr>
<td>B5</td>
<td>1.72</td>
<td>0.20</td>
<td>0.65%</td>
<td>27.13</td>
<td>0.40%</td>
</tr>
<tr>
<td>C1</td>
<td>1.64</td>
<td>0.24</td>
<td>1.00%</td>
<td>28.64</td>
<td>1.90%</td>
</tr>
<tr>
<td>C2</td>
<td>2.35</td>
<td>0.26</td>
<td>0.59%</td>
<td>22.71</td>
<td>0.80%</td>
</tr>
<tr>
<td>C3</td>
<td>2.00</td>
<td>0.22</td>
<td>0.45%</td>
<td>21.16</td>
<td>1%</td>
</tr>
<tr>
<td>C4</td>
<td>1.90</td>
<td>0.23</td>
<td>1.59%</td>
<td>21.69</td>
<td>1.55%</td>
</tr>
<tr>
<td>C5</td>
<td>1.86</td>
<td>0.20</td>
<td>0.50%</td>
<td>17.13</td>
<td>0.30%</td>
</tr>
</tbody>
</table>

To examine the corresponding correlation between different parameters, the normalized gamma ray, radon, CO₂ and CH₄ are plotted in Figure 3. ²¹⁴Bi signals in the U window are chosen to represent the gamma ray data. In this application, the statistical threshold for gamma ray anomalous values was fixed at “mean value+1/2 standard deviation”. Above this threshold, no change is applied to raw data, below this threshold, the raw data is set to zero.

Figure 3 - Radon, CO₂, CH₄ and GR-320 measurement values are normalized. Gamma ray data from the U window represents ²¹⁴Bi signals. The squares show the measuring points.

In Figure 3, higher CH₄ concentrations are evident in transect A. However, lower radon and CO₂ concentrations are also shown. In transect B, the radon value is relatively high, and the CO₂ concentration is lower compared with transect C. The maximum CO₂ concentration appears in the transect C. No clear corresponding correlation between the gamma ray, CO₂, CH₄ and radon concentrations can be identified from the results shown in Figure 3. This may reflect the complexity of signals of soil gas and gamma ray at the site. On the other hand, as soil gas follow the permeable...
pathway formed by faults and fracture zones, the measurement of higher soil gas concentration may illustrate the existence of permeable structure, but the relatively small database could not show any anomaly from the soil gas concentration.

Grasty (1987) showed that 98% gamma ray radiation comes from the top 35 cm of the earth crust. However, in this layer the radon concentration is highly variable because of variable barometric pressure, water content and gas permeability. Field experiments show different correlations between $^{214}$Bi concentration and radon concentration in different environments. Vulkan and Shirav (1997) found good correlation between $^{214}$Bi measured by an airborne gamma ray spectrometer and $^{222}$Rn concentration in an arid area, where a soil layer was almost absent. However in the semi-arid and humid areas, the correlation is poor.

Carbon dioxide and light hydrocarbon (such as methane) could easily migrate through more permeable structures. It contributes to higher concentrations of the light hydrocarbon and carbon dioxide in the upper soil layer. Because of this light hydrocarbon accumulation, Price (1986) concluded that hydrocarbon-consuming bacteria (hydrocarbon oxidizing activity) would significantly influence the near surface geochemical environment. As a result, light hydrocarbons are oxidized to carbon dioxide and lead to acidic condition in the soil and pore water. This microbial activity may be another reason for the lower methane background. The acidic environment created by the hydrocarbon degradation could also build up the uranium concentration in the soil layer, which leads to increased gamma ray emission.

**NASV DV method**

To improve the data interpretation, the noise in the gamma ray data should be filtered. The main factors that reduce the assay precision are the statistical nature of radiation, variable background and variable water content in the soil. High soil moisture could block the radiation flux. 10% increase in soil moisture will decrease about the same amount of radiation flux from the soil surface (Minty, 1997). The statistic of radioactive decay in a particular time interval follows the Poisson statistical distribution (Frigerio, 1974). The gamma ray background is originated from the atmospheric radon, cosmic background and fallout materials from nuclear accidents, such as Chernobyl nuclear accident. It does not reflect the geological information and needs to be removed from the observed gamma ray spectra.

A statistical approach was proposed by Hovgaard (1997) to extract signals in the multichannel raw spectra, called Noise-Adjusted Singular Value Decomposition (NASVD). NASVD is a procedure for removing noise from the raw gamma ray spectra using the spectral component analysis method. The observed spectra are scaled to the unit variance in each channel. Then, eigenvectors are calculated and rescaled by multiplying the unit average spectrum. The lower-order components represent signal in the original observed spectra, and the higher-order components represent noise. The noise is removed by reconstructing the spectra from the lower order eigenvectors and amplitudes (Minty and McFadden, 1998). The “cleaned” spectra are then processed using a standard 3-windows method to extract K, U and Th window data.

The first 16 eigenvectors from the NASVD analysis of raw spectra are shown in Figure 4. The coherent spectral shape is shown in the lower order eigenvector, and these are interpreted to represent the signal in the input spectra. The higher-order eigenvectors do not show evidence of coherent spectral shape, and these are mainly noise.

The comparison of the NASVD processed spectra with the raw spectra is shown in Figure 5. In the U window, the processed spectra showed a peak point and a clearer trend, compared with the more oscillatory raw spectra.

The precision of the field assays is expected to be about 0.1% K, 0.4 ppm eU and 0.6 ppm eTh (IAEA, 2003). However, at our field site the mean values were 0.1% K, 0.25 ppm eU and 0.41 ppm eTh. The media values were 0.1% K, 0.2 ppm eU and 0.4 ppm eTh. All these values are around the detection limit of the field survey, which means the background value is very low. The sensitivity level of the detector for the U window is 0.325 counts per second, and nearly all the U window counts were around this sensitivity level. In Figure 6(a), the NASVD processed data in U window do not reflect any trend compared with original spectra result in U window. In Figure 6(b), the error bars are plotted in each measurement point. It can be seen that almost all signals are strongly affected by the noise. The dot black line showing the benchmark passes through the majority of error bars with all the gamma ray data within the noise variation range, indicating that nearly no significant $^{214}$Bi signals were detected.
Figure 4 - NASVD eigenvectors from field test spectra. PC is the principle component.

Figure 5 - The comparison of NASVD processed spectra with original spectra.

Figure 6 - (a) Gamma ray survey result of the U window along the whole transect with the comparison of the NASVD processed data; (b) Gamma ray result of the U window with error bars, and the dot black line shows the benchmark.
CONCLUSIONS

The field experiment has demonstrated the complexity of the soil gas migration and gamma ray emission. The limitation of the soil gas mapping method lies in the weak crustal gas concentration in case of the thick sedimentary cover, such as alluvial soil layer (average 20 m alluvial soil layer cover the rock and coal seam at this site), leading to the spread of gases through the soil layer which broadens the anomalies. Environmental influence in the near surface soil layer may also contribute to weakening the signals. Only careful design of soil gas sampling programmes can increase the probability of detecting faults by soil gas mapping. Regarding the gamma ray survey, the gamma ray background signal seems too weak for detection. Further study is required to understand this complex system, in particular,

- How to differentiate the geogenic and biogenic sources for methane, carbon dioxide?
- How to distinguish the radon origin of deep host rock from that due to uranium mineral in the upper soil layer?

ACKNOWLEDGEMENTS

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BLAST OPTIMISATION WITH *IN SITU* ROCK MASS CHARACTERIZATION BY SEISMIC PROFILING AT AN OPENCAST COAL MINE IN INDIA

More Ramulu , Anand Ganpatrao Sangode and Amalendu Sinha

**ABSTRACT:** Blast optimisation studies were conducted at an opencast coal mine in India for selection of a site specific explosive for different rock types. This seismic refraction survey technique was applied at sandstone benches of a coal mine for rock mass characterisation and blast optimisation by impedance matching of explosives. Field experiments were conducted on seismic profiling to characterise sandstone rock mass on the basis of P-wave velocity (Vp) measurements. The running benches were selected for the experimentation so as to cross check the results of the Vp with the exposed faces of the benches. The instrument used for seismic profiling contains 24 geophones of 14 Hz frequency. The mode of survey was the ‘refraction method’ which could give the Vp profile up to 50-60 m depth and about 100 m stretch. The source of vibration generation was by hammering of specific Sledge hammer. The raw seismic data collected in the field was analysed by a software called ‘Seismic imager’ for generating a Vp profile of the rock strata. The Vp profiles were determined for three benches of the mine, which include weak, medium and hard type of rock mass. The rock impedance was calculated based on the Vp determined by seismic profiling. This data was used for the selection of explosive with desired velocity of detonation and density, so as to match the impedance of the rock mass. The blast performance with the suitable explosives with impedance matching was obviously better than that of impedance mismatching. Trials were also conducted with heavy energy-rich ANFO explosive with mismatched impedance properties and observed better results. The optimisation studies resulted in reduction of back break by 50-75% and reduction of mean fragment size by 15-47%. The paper stresses the need for conducting impedance matching exercise for all the blast sites for blast optimisation and productivity improvement.

**INTRODUCTION**

The mining productivity in open cast mines depends heavily on the degree of fragmentation. Various unit operations like drilling, blasting, loading and transport are influenced by fragmentation and jointly contribute to the overall productivity. It is often observed that practising engineers indiscriminately use explosive charges to improve fragmentation with scant regard to rock formations and explosive properties. This may not be in the best interest of the overall mine productivity. It calls for a study on proper selection of explosive for various rock properties. The best matching for optimum shock wave transmission to the rock occurs when the detonation impedance of explosive is equal to the impedance of the rock material (Atchison, 1964). Impedance is the product of compressional wave velocity and density of the material. Impedance calculation requires the determination of *in situ* P-wave velocity (Vp) and density of rock mass. Therefore the refraction seismic survey technique of seismic profiling was applied for rock mass characterisation of sandstone overburden in this study.

Continuous acquisition of multichannel surface wave data along linear transects has recently shown great promise in detecting shallow voids and tunnels, mapping the bedrock surface, locating remnants of underground mines and delineating fracture systems (Park, *et al.*, 1999). Extending this technology from sporadic sampling to continuous imaging required the incorporation of Multichannel Analysis of Surface waves (MASW) with concepts from the Common Depth Point (CDP) method (Mayne, 1962). Integrating these two methodologies resulted in the generation of a laterally continuous 2-D cross-section of the shear wave velocity field. Cross-sections generated in this fashion contain specific information about the horizontal and vertical continuity and physical properties of shallow materials. Seismic reflection surveys are generally designed to image structural and stratigraphic features with a high degree of resolution and accuracy.

Since shear wave velocity has the greatest impact on the properties of a surface wave, the dispersion curve can be inverted in such a way as to obtain the shear wave velocity as a function of depth (Xia, *et
al., 1999). Barton (2007) says that the phenomena of seismic anisotropy giving lower stiffness perpendicular to layering than in parallel, has been used since the nineteenth century for investigating fractured rock at depth. The same analogy holds good for compressional wave velocity. The objective of this experimental work has been to find out the compressional wave (Vp) structure and from these dispersion curves to obtain the inferences regarding the structural quality of the strata. Ramulu et al (2011) extensively used the seismic refraction technique for determination of Vp of rock mass for blast optimisation by impedance matching of explosives.

This paper deals with the blast optimisation by rock mass characterisation with seismic profiling at various benches of the mine.

REFRACTION SEISMIC SURVEY FOR ROCK MASS CHARACTERISATION

Data acquisition:

The instrument called Geode (Geometrics controllers Inc., USA) was used for acquiring the data for surface wave analysis using refraction seismic survey technique. The sensors used were of 14 Hz frequency and 24 in number. The refraction seismic survey system with various components is shown in Figure 1. The sensors were spread at 1 m spacing and the seismic source was at 5 m distance in all the experiments. A sledge hammer of 4.5 kg (10 lb) weight was used as seismic source. Each site will have specific characteristics effecting data properties. Optimising parameters and equipment is critical to maximising the accuracy, analysis format, and potential of the resultant processed sections. Data acquired for surface wave analysis using the refraction seismic survey technique are generally broadband (i.e., 4 Hz to 64 Hz), with offsets designed and based on target dimensions and depths. Standard Common Mid-point (CMP) roll-along techniques are used in conjunction with 24-channel recording systems. Shot and receiver spacing as well as near and far source offsets depend on number of recording channels and maximum and minimum depth of interest.

![Figure 1 - Various components of the refraction seismic survey system](image)

Ground cover (such as soil, cement, gravel and grass) has no significant influence on the accuracy of the recorded surface wave energy (Miller and Xia, 1999). Generation of surface waves is quite easily accomplished with weight drop style sources, with the particular specifications of the source only limited by the dominant frequency band of interest. For deeper penetration a large and heavy source is optimum. Receivers need to be low frequency (< 8 Hz) and broadband. With cost consideration, the optimum geophone has a natural frequency of around 4.5 Hz and can be outfitted with either flat base plates or short spikes depending on the surface to be surveyed. Recording geometries and frequency ranges of data examples presented here provided optimum data characteristics for examining earth materials in the depth range from about one to over 50 m below ground surface. Many studies have
shown that receiver-ground coupling is critical for high-resolution body wave surveys (Hewitt, 1980). Maximising frequency response and recorded body waves normally requires longer spikes, well seated into competent earth. Coupling experiments at various sites have suggested receivers only require simple ground contact to record broad-spectrum surface wave energy. Little or no improvement is evident in response (frequency vs. amplitude) when geophones are “planted” using spikes, placed on the ground using plates, or held to the ground with sandbags (Miller and Xia, 1999).

FIELD APPLICATION OF THE SEISMIC PROFILING

Mine details

The field experiments on seismic profiling and impedance matching were conducted at one of the Opencast Project (OCP) mines of Coal India Limited, which is situated about 25 km from Nagpur town. The area is characterized with flat topography having elevations ranging from 298.7 m to 304.8 m (980 ft to 1000 ft) above mean sea level. There are five coal seams namely, I, II, III, IV, and V in the leasehold area of the colliery. There is also a problem of optimum blast fragmentation, which may be due to mismatching of rock mass properties and explosive selection.

Geology

The exposures of Lower Gondwana rocks around Tekadi – Silewara – Patansaongi – Bokhara – Khapa - Saoner belt located about 25-30 km from Nagpur. The OCP mine Coalfield is a horse-shoe shaped basin aligned in a NW-SE direction. The coalfield is blanketed by a detrital mantle. The Barakars overlie the Talchirs and underlie the Moturs conformably. They consist of fine, medium and coarse grained sandstone, intercalations of shale and sandstones, sandy shale, carbonaceous shale and coal seams and are around 300 m in thickness. Kamthis is a good aquifer and overlaps directly above Barakars. The dip of the seams is about 1 in 4.5 on the rise side and about 1 in 5 to 1 in 6 on the dip-side. It shows a tendency to further flatten beyond the existing limit of working. The rock properties of all the three benches, where tests were conducted are shown in Table 1. The intact rock P-wave velocity was tested by ultrasonic device as shown in Figure 2.

<table>
<thead>
<tr>
<th>Site</th>
<th>Rock density (kg/m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bench-I</td>
<td>2550</td>
</tr>
<tr>
<td>Bench-II</td>
<td>2600</td>
</tr>
<tr>
<td>Bench-III</td>
<td>2675</td>
</tr>
</tbody>
</table>

Seismic profiling at OCP coal mine

A seismic profiling survey was carried out to characterise sandstone rock mass on the basis of P-wave velocity measurements. The survey was carried out at the surface of sandstone rock mass towards N.E direction of the Mine. The survey was carried out at three locations of the mine covering hard, soft and medium rock mass. The running benches were selected for the experimentation to cross check the results of the P-wave velocity profile (V_p) with the exposed faces of the benches. The experimental
set up is shown in Figure 2. The seismic profiler experimental set up is shown in Figure 3. The rock samples were collected from the middle and bottom layers of bench-I for laboratory testing of P-wave velocity. The faces of exposed benches are shown in Figure 4.

![Figure 3 - Seismic profiler experimental set up](image)

(a) Hammering point  
(b) Geode and geophones

![Figure 4 - Exposed face of a test site](image)

The mode of survey was the 'refraction method' which could give the $V_p$ profile of 50 to 60 m depth over a length of about 100 m. The source of vibration generation was by hammering by means of a 4.5 kg (10 lb) weight Sledge hammer with 5-10 numbers of blows. The seismic raw data generated in the field was stored in a lap-top computer connected to Geode while surveying. The raw data collected in the field was analysed by a software called 'Seismic imager'. The processed data generated a $V_p$ profile of rock strata up to a depth of 25 m from the surface. The P-wave velocity profiles of the each bench were initially analysed for the composite layers and smooth layering was done afterwards for generalisation of rock mass characterisation. The $V_p$ profiles of Bench-I, which is comparatively soft formation is shown Figure 5. The P-wave velocities of individual layers varied from 240 m/s to 2200 m/s from top to bottom. The poor $V_p$ at the top might be because of fractures generated due to the weathering of the rock mass. The $V_p$ profiles of Bench-II is shown in Figure 6. The P-wave velocities varied from 500m/s to 2314 m/s and the poor $V_p$ at the top might be because of fractures generated due to the production blasting in the past. The $V_p$ profiles of Bench-III is shown in Figure 7. The P-wave velocities in smooth layered analysis were varying from 500 m/s to 2500 m/s from top to bottom and the here also the top layer gives poor $V_p$, which might be because of fractures due to previous blast rounds. The in situ and laboratory P-wave velocities of sandstone rock for all the test sites are given in Table 2.

The P-wave velocities indicate that there is 19-25% increase in laboratory $V_p$ values in comparison to field $V_p$ values at all the test sites. This indicates that the field $V_p$ profiles are realistic measurements by the seismic profiling surveys as reported by Barton (2007).
**Figure 5** - $V_p$ profile of major cluster of layers of sandstone strata at bench-I

**Figure 6** - $V_p$ profile of major cluster of layers of sandstone strata at bench-II

**Figure 7** - $V_p$ profile of major cluster of layers of sandstone strata at bench-III
Table 2 – *In situ* and laboratory P-wave velocities of sandstone rock

<table>
<thead>
<tr>
<th>Site</th>
<th>Field P-wave velocity, m/s</th>
<th>Laboratory P-wave velocity, m/s</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bench-I</td>
<td>2000</td>
<td>2380</td>
</tr>
<tr>
<td>Bench-II</td>
<td>2200</td>
<td>2640</td>
</tr>
<tr>
<td>Bench-III</td>
<td>2500</td>
<td>3125</td>
</tr>
</tbody>
</table>

**OPTIMISATION OF BLAST FRAGMENTATION BY IMPEDANCE MATCHING AT OC MINE**

**Blasting practice at OCP mine**

The prevailing blasting practice at OCP mine is carried out with cartridge explosives of fixed velocity of detonation (VOD) for all the benches, irrespective of various rock properties. The blast results like fragmentation, throw and peak particle velocity of vibration were monitored using high resolution video camera and seismographs. Fragmentation size distribution analysis was carried out by image analysis software called Wipfrag. The blast design parameters and the blast results are given in Table 3 and Table 4 respectively.

**Table 3 - Existing blast design parameter at OCP mine**

<table>
<thead>
<tr>
<th>Blast No.</th>
<th>Location</th>
<th>Drilling pattern</th>
<th>No. of rows</th>
<th>Hole diameter, mm</th>
<th>Bench height, m</th>
<th>Hole depth, m</th>
<th>Burden, m</th>
<th>Spacing, m</th>
<th>Delay used, ms</th>
<th>Charge/hole, kg</th>
<th>VOD of explosive, m/s</th>
<th>Density of explosive, kg/m³</th>
<th>Specific charge, kg/m³</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>OB bench-I</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>2</td>
<td>OB bench-I</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>3</td>
<td>OB bench-I</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>4</td>
<td>OB bench-II</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
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<td>1100</td>
<td>0.6</td>
</tr>
<tr>
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<td>4</td>
<td>150</td>
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<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
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<td>1100</td>
<td>0.6</td>
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<tr>
<td>6</td>
<td>OB bench-II</td>
<td>Staggered</td>
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<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
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<td>1100</td>
<td>0.6</td>
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<tr>
<td>7</td>
<td>OB bench-II</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
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<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
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<tr>
<td>8</td>
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<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
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<tr>
<td>9</td>
<td>OB bench-II</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
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<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>10</td>
<td>OB bench-III</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>11</td>
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<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>12</td>
<td>OB bench-III</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>13</td>
<td>OB bench-III</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>14</td>
<td>OB bench-III</td>
<td>Staggered</td>
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<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
<tr>
<td>15</td>
<td>OB bench-III</td>
<td>Staggered</td>
<td>4</td>
<td>150</td>
<td>6.5</td>
<td>7</td>
<td>4.5</td>
<td>3</td>
<td>25</td>
<td>50</td>
<td>3000</td>
<td>1100</td>
<td>0.6</td>
</tr>
</tbody>
</table>

**Table 4 - Blast results with existing impedance values**

<table>
<thead>
<tr>
<th>Blast No.</th>
<th>Location</th>
<th>Throw, m</th>
<th>Back break, m</th>
<th>Mean Fragment size, m</th>
<th>PPV at 55m distance, mm/s</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Bench-I</td>
<td>10</td>
<td>1.0</td>
<td>0.46</td>
<td>15.8</td>
</tr>
<tr>
<td>2</td>
<td>Bench-I</td>
<td>8</td>
<td>2.0</td>
<td>0.38</td>
<td>17.2</td>
</tr>
<tr>
<td>3</td>
<td>Bench-I</td>
<td>9</td>
<td>1.0</td>
<td>0.52</td>
<td>16.6</td>
</tr>
<tr>
<td>4</td>
<td>Bench-II</td>
<td>8</td>
<td>3.0</td>
<td>0.41</td>
<td>17.2</td>
</tr>
<tr>
<td>5</td>
<td>Bench-II</td>
<td>10</td>
<td>3.0</td>
<td>0.36</td>
<td>15.9</td>
</tr>
<tr>
<td>6</td>
<td>Bench-II</td>
<td>9</td>
<td>2.0</td>
<td>0.42</td>
<td>16.3</td>
</tr>
<tr>
<td>7</td>
<td>Bench-III</td>
<td>8</td>
<td>2.5</td>
<td>0.45</td>
<td>15.5</td>
</tr>
<tr>
<td>8</td>
<td>Bench-III</td>
<td>10</td>
<td>1.0</td>
<td>0.53</td>
<td>17.3</td>
</tr>
<tr>
<td>9</td>
<td>Bench-III</td>
<td>10</td>
<td>2.0</td>
<td>0.51</td>
<td>16.0</td>
</tr>
</tbody>
</table>

**Blast optimisation by impedance matching of shock energy**

The best matching for optimum shock wave transmission to the rock occurs when the detonation impedance of the explosive is equal to the impedance of the rock material. According to the theory of impedance matching, the explosive impedance should be as nearer to the rock impedance as possible.
to couple the explosive induced stress waves through the rock mass. The impedance matching expression is given below (Persson, et al., 1994).

\[ \rho_e C_d = Z_r \rho_r C_p \]

Where:
- \( \rho_e \) = explosive density;
- \( C_d \) = VOD of explosive;
- \( \rho_r \) = rock density;
- \( C_p \) = P-wave velocity; and
- \( Z_r \) = impedance ratio.

It is very clear from the rock mass properties of the sandstone that the compressional wave velocity is varying from 2000-2500 m/s from Bench-I to Bench-III, however there is no change in the explosive properties, especially VOD. Substituting the values of rock and explosive parameters given in Tables 3 and Table 4, in the Equation (1), the impedance ratio \((Z_r)\) values were calculated as 0.66, 0.58 and 0.49 for Bench-I, Bench-II and Bench-III, respectively. These \( Z_r \) values are considered as poor from the impedance matching point of view (Persson, et al., 1994). This indicates that the explosive which was used for blast fragmentation is relatively suitable for Bench-I, but not for Bench-II and Bench-III.

Based on the impedance values of various rock masses at all the three benches, the best possible explosive impedance was calculated and shown in Table 5. As there was some technical limitations on the increase of density of explosive beyond 1100 kg/m\(^3\), only VOD values were adjusted as 3400 m/s, 3700 m/s and 4100 m/s for Bench-I, Bench-II and Bench-III, respectively. This combination of explosives resulted in the \( Z_r \) value of above 0.7 for all the benches. The modified VOD values were applied in all the three test sites and the blast results with modified explosive parameters i.e. impedance matching are given in Table 5.

### Table 5 - Blast results with modified VOD values

<table>
<thead>
<tr>
<th>Blast No.</th>
<th>Location</th>
<th>Throw, m</th>
<th>Back break, m</th>
<th>Mean Fragment size, m</th>
<th>PPV at 55m distance, mm/s</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Bench-I</td>
<td>6</td>
<td>0.50</td>
<td>0.39</td>
<td>11.3</td>
</tr>
<tr>
<td>2</td>
<td>Bench-I</td>
<td>7</td>
<td>1.00</td>
<td>0.31</td>
<td>12.4</td>
</tr>
<tr>
<td>3</td>
<td>Bench-I</td>
<td>9</td>
<td>1.00</td>
<td>0.41</td>
<td>14.3</td>
</tr>
<tr>
<td>4</td>
<td>Bench-II</td>
<td>9</td>
<td>0.75</td>
<td>0.30</td>
<td>12.5</td>
</tr>
<tr>
<td>5</td>
<td>Bench-II</td>
<td>6</td>
<td>1.25</td>
<td>0.32</td>
<td>12.2</td>
</tr>
<tr>
<td>6</td>
<td>Bench-II</td>
<td>9</td>
<td>1.00</td>
<td>0.31</td>
<td>14.5</td>
</tr>
<tr>
<td>7</td>
<td>Bench-II</td>
<td>9</td>
<td>0.50</td>
<td>0.28</td>
<td>11.5</td>
</tr>
<tr>
<td>8</td>
<td>Bench-III</td>
<td>7</td>
<td>0.75</td>
<td>0.31</td>
<td>9.6</td>
</tr>
<tr>
<td>9</td>
<td>Bench-III</td>
<td>6</td>
<td>0.50</td>
<td>0.27</td>
<td>10.7</td>
</tr>
</tbody>
</table>

The improvements in blast performance due to impedance matching are given in Table 6. The results clearly indicate that the selection of proper explosives with impedance matching to the rock impedance result in improving the blast fragmentation, reducing the throw and reducing blast vibrations. The overall throw in the modified blast rounds was reduced by about 25%. The back break was reduced by about 50% at Bench-I and upto 75% at both Bench-II and Bench-III. The mean fragment size of blast fragmentation was reduced by 15-21% at Bench-I, 11-26% at Bench-II and it was 37-47% reduction at Bench-III. Earlier works on the relation between VOD and damage by Singh and Xavier (2005) also indicate that the high VOD explosives produce less damage for the reason that generally the high VOD explosives yield higher shock energy and less gas energy.

**Blast optimisation by considering heave energy**

As the rock mass to be blasted is sandstone which is not so hard, a low brisance explosive such as ANFO was proposed for blasting. Considering the density of ANFO as 800 kg/m\(^3\) and VOD as 4100 m/s, the impedance ratio \((Z_r)\) values were calculated as 0.66, 0.6 and 0.52 for Bench-I, Bench-II and Bench-III, respectively. These \( Z_r \) values are considered as poor from the point of view of shock energy impedance matching. In spite of the poor \( Z_r \) values, ANFO was proposed to be used for sandstone benches and the blast performance was monitored. The representative blast fragmentation images captured for both the explosives are shown in Figure 8. The fragmentation analysis was done by the digital image analysis technique by using Wipfrag software. The sieve analysis of the fragmentation
was performed to both ANFO and emulsion explosives. The mean fragment size, which is the representative size of the average fragmentation size was 0.23 m with Emulsion explosive and 0.16 with ANFO explosive, which is about 30% improvement. Use of ANFO explosive also reduced the vibration intensity by 15-20% in comparison to the vibration induced by Emulsion explosive. The comparative results are shown in Figures 9 and 10. From the sieve analysis results it is very clear that the ANFO with poor impedance matching resulted in better fragmentation than the emulsion explosives with very good impedance matching. This indicates that the heave energy component of an explosive plays a vital role in fragmentation than the shock energy for the rock formations like sand stone. This might be because of the reason that a meager amount of shock energy is sufficient for forming crack network in soft to medium hard rocks. But there should be enough heave energy to extend the cracks for fragmentation.

Table 6 - Improvements in blast performance due to impedance matching

<table>
<thead>
<tr>
<th>Blast No.</th>
<th>Location</th>
<th>Percentage reduction in throw, m</th>
<th>Percentage reduction in back break, m</th>
<th>Percentage reduction in Mean Fragment size, m</th>
<th>Percentage reduction in PPV at 55m distance, mm/s</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
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<td>40</td>
<td>50</td>
<td>15.22</td>
<td>28.48</td>
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<td>12.5</td>
<td>50</td>
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<td>27.91</td>
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<tr>
<td>3</td>
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<td>0</td>
<td>0</td>
<td>21.15</td>
<td>13.86</td>
</tr>
<tr>
<td>4</td>
<td>Bench-II</td>
<td>-12.5</td>
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<td>5</td>
<td>Bench-II</td>
<td>40</td>
<td>58</td>
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<td>Bench-II</td>
<td>0</td>
<td>50</td>
<td>26.19</td>
<td>11.04</td>
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<tr>
<td>7</td>
<td>Bench-III</td>
<td>-12.5</td>
<td>80</td>
<td>37.78</td>
<td>25.81</td>
</tr>
<tr>
<td>8</td>
<td>Bench-III</td>
<td>30</td>
<td>25</td>
<td>41.51</td>
<td>44.51</td>
</tr>
<tr>
<td>9</td>
<td>Bench-III</td>
<td>40</td>
<td>75</td>
<td>47.06</td>
<td>33.13</td>
</tr>
</tbody>
</table>

(a) Fragmentation with Emulsion              (b) Fragmentation with ANFO

Figure 8 - Representative images of blast fragmentation with Emulsion and ANFO explosive

Figure 9 - Fragment size distribution of muckpiles of test blasts with Emulsion explosive
CONCLUSIONS

The improvements in blast performance due to impedance matching were substantial in terms of blast fragmentation and vibration as well as damage control. The overall throw in the modified blast rounds was reduced by about 25%. The back break was reduced by about 50% at Bench-I and up to 75% at both Bench-II and Bench-III. The mean fragment size of blast fragmentation was reduced by 15-21% and Bench-I, 11-26% at Bench-II and it was 37-47% at Bench-III. The vibration intensity was also reduced by 14 to 45% with increase of impedance matching of explosives. The blast results shown in this study, clearly indicate that the selection of proper explosives with impedance matching to the rock impedance result in improving the blast fragmentation, reducing the throw and reducing of blast vibrations. The study also reveals that the heave energy factor plays a more vital role than the impedance matching of the shock energy for fragmentation of rock formation like sandstone. Test blasts with ANFO explosive with mismatched impedance properties resulted in 30% improvement of fragmentation and reduced the vibration intensity by 15 to 20% in comparison to the Emulsion explosive. Therefore, impedance matching as well as heave energy utilisation should be given adequate importance while selecting of explosives for improving blasting productivity and safety.

ACKNOWLEDGEMENTS

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QUALITY ASSURANCE PROGRAMS FOR MANAGEMENT EXCELLENCE. ARE THEY RELEVANT TO THE COAL INDUSTRY?

Allison Golsby

ABSTRACT: Quality assurance programs were introduced into industry in 1973. Over time these systems have evolved to what is available to management today.

How can quality assurance programs be used to ensure management excellence and are they relevant to the Coal Mining Industry?

INTRODUCTION

The term ‘Mine Manager’ is often all about ‘mining’ and too little about ‘management’. The technical demands of mining needed to safely extract resources demand personnel with the appropriate level of education and training. The requirements for statutory roles such as Ventilation Officer, Deputies, Undermanagers and Mine Managers include the assessment of their competencies, where their technical skills and expertise are benchmarked against standards. Quality Assurance is the name for the gap analysis that measures the degree of compliance with the standards for management excellence.

The benefits of utilising management excellence in an organisation include:

- Effective prioritisation of improvement efforts to deliver maximum benefits;
- Process efficiency and effectiveness through reduced waste and variation;
- Empowered and motivated workforce with increased retention;
- Increased productivity and reduced operational costs;
- Focus on customer service delivering superior perception of value;
- Sustainable performance by increasing stakeholder value.

WHAT IS QUALITY ASSURANCE? AND WHAT DOES MANAGEMENT EXCELLENCE MEAN?

‘There is an old saying that demonstrates the importance of systems and structures: “What is important, gets measured - what is measured, gets rewarded - what is rewarded, gets done!” Of course, people and companies that get things done get ahead’ (Dowling, 2001).

There is an Australian Standard Quality Assurance Program for Management Excellence that benchmarks management systems. The Australian Business Excellence Framework (GB.002-2011) was developed with the objective of describing the principles and practices that create high performing organisations. The criteria can be used by organisations to assess their performance and drive continuous and sustainable improvement in their leadership and management systems. Excellence in management is achieved when management systems measure up to, or exceed the standard. Too often management systems fall short, but too few in mining are even doing that gap analysis.

Excellence in management is not just about designing efficient systems for driving the management process. It is not just about ensuring that a Permit or Authority to Mine is available when mining is due to commence in an area and that it has been signed off by all those needing to do so. In that case, excellence in management is about ensuring that the management process to generate the Permit or Authority to Mine was commenced in time, proceeded through the management chain in a smooth manner and arrived fully completed where and when it was needed. Moreover its path through all of...
the various steps in the chain of command needs to be smooth, not accompanied by any last minute, just in time, pushing and shoving.

People skills are an integral part of management excellence. The paper, or at the more technically advanced mines, the emails and electronic reports need to be processed as efficiently as the resource, if mining is to achieve its goal of optimising resources. But people need to be managed just as effectively as the paper, the emails or the physical resource. Therefore, a leadership competency framework like the one shown in Figure 1, is needed as bases for ensuring managers have these minimum required skills.

**Figure 1 - Leadership competency framework, for business excellence (ACELG 2011)**

Many mine managers hearing the expression ‘optimising resources’ may feel that means getting more coal out of the ground more quickly. However, if the people, many of whom work in the back office are not managed effectively, the coal will not be cut because:

- the Special Mining Vehicles (SMV) Driftrunner is not operative;
- it’s raining and there are no pumps;
- the transformer was moved and no one was told;
- the roof bolts were delivered to outbye;
- the resins were left outside in the weather and have gone off;
- the bolting dollies were not purchased;
- the mesh sheets are the wrong size;
- the bucket for the loader has gone missing;
- the spare cable in the panel has been removed;
- the roof bolts are too short;
- the cable winder is missing;
- the belt rubber is too wide for the conveyer frame;
- only miner cables, but no shuttle car cables are available;
- the stone dust was stored in a puddle of water;
- the emergency pod has been moved and no one was told;
- the miner is ready to flit and the ventilation stoppings are not ready;
- the scheduled belt move did not happen;
- the deputy for this shift is not on duty;
- there is no ticketed miner driver;
- the scheduled survey was not completed;
- that drill hole is full of water;
- a sequence plan is not on site; and
- ventilation has stopped.

The Mining Industry has historically promoted the highly technically adept personnel into management roles, as they ‘do such a good job on the tools’. From this promotion, personnel are moved out of their area of technical expertise into an area they know nothing about. Figure 5 shows the utilisation of a technical person’s time. New to the role and without the appropriate management tools, managers are left to struggle through the people management aspects (Figure 3) without any support via technical skills. The industry has recognised this issue. There is now more of a succession planning process in place to prepare personnel for the leap into management. There are now credible courses such as the ‘Frontline Management Framework’ aimed at assisting these very technically adept personnel to develop management skills.

The Frontline Management qualification reflects the role of individuals who take the first line of management in a wide range of organisational and industry contexts. They may have existing qualifications and technical skills in any given vocation or profession, yet require skills or recognition in supervisory functions. Typically they would report to a manager. At this level frontline managers provide leadership and guidance to others and take responsibility for the effective functioning and performance of the team and its work outcomes.

The next step is to develop the supervisors into a more people oriented role. Figure 2 shows the utilisation of a manager’s time. Management skills have not readily been accepted as a necessary skill at the mine manager level.

![Manager's Time](image)

Figure 2 - Utilisation of a manager’s time (Golsby, 2011)
IS COAL MINE MANAGEMENT TOO REACTIVE, RATHER THAN PROACTIVE?
CAN WE DO IT BETTER?

One Mine Manager might define their people skills as, ‘I don’t have to be polite and talk to the employees. I have surrounded myself with good HR people so I can behave anyway I like, and if anyone doesn’t like to hear me carry on then they don’t need to talk to me’. Is this an effective Manager? Will this Manager be able to resolve their people management issues in a timely and effective manner if they are not interested in their own projection of themselves?

Supervisor's Time

Figure 3 - Utilisation of a supervisor’s time (Golsby, 2011)

Yet, the benefits of using business excellence, far outweighs the investment in developing and administering business excellence systems.

The Australian Business Excellence Framework aims to create an environment of continuous improvement, by:

- Providing the foundation with which to develop the organisation’s focus on sustainable performance;
- Offering a solid structure for integrating all improvement initiatives and organisational decision-making;
- Helping to achieve organisational goals and deliver increasing customer and stakeholder perception of value;
- Providing a performance benchmarking program;
- Identification and communication of priority areas for attention and action to improve organisational performance;
- A regular and systematic process of information, analysis and comparison of performance against agreed organisation performance criteria.

The other noticeable benefits include improved safety performance and increased returns to the bottom line (Table 1).

Table 1 - 1990-2006 Australian Business Excellence Award Winners vs. Standard and Poor’s Accumulated Index (SAI Global 2007)

<table>
<thead>
<tr>
<th>Award recipients</th>
<th>TOTAL A$ Investment 31 October 1990</th>
<th>TOTAL A$ Value 10 March 2003</th>
<th>TOTAL A$ Value 30 June 2006</th>
<th>THPR % Since 2003</th>
<th>THPR % Since 1990</th>
</tr>
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<tbody>
<tr>
<td>Standard and Poor's All Ordinaries</td>
<td>$12,192</td>
<td>$44,113</td>
<td>$118,477</td>
<td>168.58%</td>
<td>871.76%</td>
</tr>
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</table>
The Australian Business Excellence Awards Winners (as shown in Table 1) outperformed the All Ordinary Index, 3.5 to 1.

Leading Australian based organisations use the business excellence framework to:

- Improve management and leadership practices;
- Assess the performance of their leadership and management systems;
- Build those results into strategic planning processes; and
- Benchmark where their organisation stands in terms of the marketplace and competitors.

The business excellence framework provides to the industry, an umbrella under which a number of business initiatives can be integrated to form one coherent, cohesive organisational systems model.

**HOW CAN QUALITY ASSURANCE PROGRAMS BE USED TO ENSURE MANAGEMENT EXCELLENCE? ARE THEY RELEVANT TO THE COAL MINING INDUSTRY?**

There has been a change in thinking over time, from a customer only focus in the previous Australian Business Excellence Framework (SAI, 2007) category of ‘Customer & Market Focus’ to ‘Customers & Other Stakeholders’ (in GB.002-2011 as shown in Figure 4), broadening the category to include all stakeholders.

‘Corporate culture can work for you or against you. A major role of management within any company is to lead change within the organisation. It is those individuals and companies who embrace change and can lead change by moulding their corporate culture to the changing business circumstance that will survive and prosper. By coupling a cultural influence strategy to a cultural strategy, change can be a very positive and rewarding experience. With experience and positive results, these organisations obtain a special capability to react quickly to their customers and create extraordinary value. On the other hand, those organisations or individuals that cannot adapt or lead change will, over a period of time, fail, and others will capture their resources and inherent value’ (Dowling, 2001).

Without striving for continuous improvement and without using quality assurance programs, management excellence cannot be achieved. Some of the quality assurance programs used in conjunction with business excellence, are:

- ISO 9000 series;
- Six Sigma;
- Balanced Scorecard;
• Enterprise Resource Planning;
• Triple Bottom Line reporting;
• Corporate Governance; and
• Risk Management.

These systems are recognised in industry. As with any system, they are only as good as the practitioners, standards, procedures, competency of all participants, documentation, communication and record management in play.

Risk management as a quality assurance program referenced in legislation, has a suite of Australian Standards (ISO31000, ISO9000, ISO14001 and AS/NZS4804) to guide users. There is also nationally accredited training for facilitation and the use of risk management tools and the risk management assessments are audited by regulatory groups. All of these measures continue to encourage improvement in the use of and the outcomes of each risk management system.

![Technical Personnel 's Time](Figure 5 - Utilisation of a technical personnel's time (Golsby 2011))

WE ARE ALL PARTICIPANTS IN THE MINING INDUSTRY. WE ARE PROUD OF WHAT WE PRODUCE. WE ALL KNOW THE TERMS ‘FIT FOR PURPOSE’ AND ‘RIGHT FIRST TIME’.

One of our fine members of the mining profession once had the honour of becoming the President of the United States of America. His name was Herbert Hoover. He was given credit for making a very profound statement as the keynote speaker at one of the American Institute of Mining Engineers’ annual meetings during the 1950s. Allegedly, he said “our mining industry manages to survive in spite of ourselves”! (Riggs, 2001).

Management Excellence is a path of continuous improvement. Questions that can help with organisational continuous improvement include:

• How do we define, recognise, assess and promote it?
• How might the business excellence framework need to change?
• What other tools are out there?
• What can other sectors teach the mining industry about excellence?
• What are typical elements of the organisation’s continuous improvement program?
• What are the legislative frameworks that jurisdictions use to promote it?
• How can the business excellence framework work in more efficiently with existing legislative requirements?
• Is there a conflict between quality control and innovation?
Do concepts of public and customer value differ from those of Business Excellence?

How can issues of governance and community be incorporated into the business excellence framework?

How do we best support and advance excellence in the industry?

The goal for coal mining is to make Mine Managers as effective at ‘management’ as they may be at ‘mining’.

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