

PROCEEDINGS OF THE 2009 COAL OPERATORS' CONFERENCE

UNIVERSITY OF WOLLONGONG, NSW

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FOREWORD

The organizing committee of the Coal 2009 is pleased to hold, once again, the 9th Coal Operators' Conference at Wollongong. This year the conference is held at the UOW newly constructed Innovation campus, and has broadened the organizing committee membership to include new members from across the industry. Despite the current economic situation, the conference attracted a record number of papers. We are very pleased to have participants from China, India and Iran, Kazakhstan, and Turkey. This is a healthy sign of the increasing awareness of the Coal Operators' Conference in Australia and beyond.

A total of 40 papers are in the conference proceedings from different field of mining operations, ranging from longwall mining, ground control, mine gases and outburst control, spontaneous combustion, mine dust control and others. The interest in the importance of the conference is reflected on the quality of papers presented and this conference is now being established as a popular venue for reporting on new technologies introduced to the industry for the betterment of mine production, productivity and safety.

Many companies and organisations have in the past generously sponsored coal operators' conference series, and this year several companies are, once again, providing sponsorship to Coal 2009. The sponsors support is very much welcomed and is a vital factor in keeping the conference registration at an affordable rate.

We would like to express our sincere thanks to:

- The organizing committee members for their diligence and hard work in making this conference a success,
- The authors of the papers, who have taken considerable time and effort in the preparation of their papers to the required standards
- The reviewers of the papers, which at times has not been an easy task, but ensured the high standards of the papers being maintained,
- Peter Vrahas and Robyn Dawson of the Uni-Centre of the University of Wollongong for the management and registration of the conference.
- Leonie McIntyre of the Faculty of Engineering, University of Wollongong for type setting the conference proceedings,
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- Staff of The Wollongong University Printery for printing the conference proceedings and to Anthony Petre for designing the proceedings covers.

Naj Aziz (*Conference Convener and Editor*)

Jan Nemcik, (*Co-editor*)

PREFACE

The Australasian Institute of Mining and Metallurgy aims to “maximise the opportunities for professionals in the minerals sector, and to promote the value of the industry to the wider community”. A number of ways whereby this can be achieved include;

- Leading and upholding industry codes of conduct, reporting and standards.
- Providing stimulating opportunities for continuing professional development and recognition.
- Recognising, rewarding and promoting best practice throughout the industry.
- Presenting opportunities for networking and social interaction amongst professionals.

Coal 2009 as an industry based conference combining the support of both the Wollongong University and the Illawarra Branch of Aus IMM, meets all these objectives.

The theme of “Improving Fundamental Practices” has endured throughout all previous Coal series conferences held in Wollongong, dating back to 1998 and supports the belief that there could always be better, safer, more productive and sustainable ways of doing what we do. With the recent world economic conditions the need to be proactive and adaptive is all the more important.

The need to communicate with each other, compare experiences, identify problems as well as successes, are all part of the interchange of ideas necessary for our professional development. The diversity of papers presented in Coal 2009 exemplifies the wide range of interests, areas of study and the complexity of issues that confront the current coal mining industry.

Special thanks go to the organising committee, the authors and our sponsors. They all play a vital role in the success of this conference and with their ongoing support and assistance we hope to continue with similar conferences into the future.

As the Chairman of the Illawarra Branch for the Australasian Institute of Mining and Metallurgy I welcome you all to the Illawarra and to Coal 2009.

Dr Chris Harvey
Chairman, Illawarra Branch

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HOW CAN WE HELP AND IMPROVE

Maarten Velzeboer¹

The world has changed in a matter of six months. Last year the sky was the limit, today there are many companies living from hand to mouth. Bungee jumping is a recreational activity and now senior management is forced to enjoy the ride as part of day to day business.

Even the Australian mining scene is affected and we are again a “cyclical” industry. This is a worry, as the economic returns pay our wages and generate the investments necessary to maintain a safe and vigorous industry. Over time as the easier deposits are being depleted, the technical and geo-environmental challenges call for greater efforts. Not only in the shape of mine operational initiatives, but also services, equipment and novel applications from other disciplines.

Fortunately during the good times, the physical and intellectual investments in Australia have been high and some of the good “ideas” and initiatives have made it into the production environment. During the period Australia has become a major source of Mining Manpower to the world. Australian expertise has become the backbone of many of the new projects. This has been made possible by the strong and vibrant domestic industry, educational system and service/equipment manufacturing capability.

The current downturn, I like to describe as a breathing space, should allow us to reflect, take stock and determine what is important for the future. Chasing ones tail all the time, may be dynamic, but does this yield BEST results? I am certainly not suggesting that we do nothing, and wait for things to improve. We should use the opportunity to position ourselves for the future upturn, in what form and when this will happen we do not know. What we do know is that the world will be different. As a person who is based and operates in a non Australian environment I am grateful for your contribution on a global scale. It hurts when you also dominate on the sports field, but that at least gives us something different to talk about.

Thank you for allowing me to address you on this occasion. Wollongong and the Australian mining community in particular, bring back fond memories. Also sad ones. I was doing my underground time for my ticket, at Appin when they had the explosion. I did early work on “gassiness” at South Bulli, with lots of help from both Allan Hargraves and Ripu Lama. Both gas pioneers. I am grateful for my early career in Australia, now I am back to learn from the Australian industry, to assist our overseas operations.

I would like to use our ArcelorMittal operations as an example of some of the basic challenges and as a mirror as to some of the future needs of the underground coal industry. In this context I am not only referring to the technical requirements, but increasingly there is a social dimension and responsibility to all we do. This social aspect not only covers environment and being a good neighbour, but also has a strong human focus. Mining always has been a “people” business. I am not only referring to SAFETY but also employee development and community awareness.

ArcelorMittal are the leading steel producer and operate on a global scale. We also strive to become significantly self sufficient in relation to our main inputs. Outside the obvious iron ore, coking coal is a major component in making steel. ArcelorMittal currently operates several surface and underground mines in West Virginia (US), three underground coal mines in Kuzbass, Siberia (Russia) and the Karaganda coalfield in Kazakhstan. Currently the total local service companies are part of the ArcelorMittal establishment. The mines operate multi annual production capacity is some 20 odd Mt. The Karaganda coalfield consists of 8 underground shaft mines, with an annual capacity of 12 Mt. Each mine currently produces some 1.5-2 Mt (ROM) with an average complement of 2,000 people. In addition many of the seams, with at least one thick one of 5-7 m, some of which have significant dips. The key technical issues are gas, outbursts, spontaneous combustion and past operational history.

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This last aspect relates to the Soviet era where a prescriptive form of management was practiced linked with a strong "blame" culture. This usually results in the accident victim being considered the root cause of the incident. That all is not well is reflected in the safety statistics, which are characterised by major events resulting in multiple deaths.

For some time now, ArcelorMittal have been working on a focused programme to break the historic safety performance. Not only have we been concentrating on the human behaviour aspects, but also on a programme of modernisation. The basic technical issues associated with high gas levels, outbursts, improved ventilation practices and roof/rib support have been reviewed and are being addressed.

Let me explain:

- 1) The gas levels of some of the main seams are very high by anybodies standards. Recent measurements confirmed up to 25 m³/t for the D6 seam, which is also very prone to outbursts as the bottom section of the 6 m seam contains a shear zone of soft and very fine coal. It is interesting to note that 25 m³/t level is very similar to what the Russian exploration predicted some 40 years ago.

Our measurements also show that the top section of the seam may act as a significant gas reservoir, which possibly feeds the violent gas and fine coal release from the shear zone. In addition the permeability of the seam is low, which makes effective predrainage difficult and so far ineffective. Work on the size distribution and shape particles from the different seam sections may allow us to introduce a permeability enhancing methodology. It is also expected that by understanding the outburst mechanism, preventative measures and techniques can be developed to create a safe longwall panel development environment.

- 2) Currently stone drives are first driven below the seam, from which the future gate road positions are degassed, before in seam development is started. Even in such a controlled environment, regular discharges of several tonnes of fine coal and over 1,000 m³ of CH₄ still happen, when drilling the pre-drain holes.
- 3) During production gas make during mining is a major problem, as is management of the goaf gas. With background values at the face of 0.8 % CH₄, there is little room for any additional release during production. Recent trials of cross measure drilling and gas extraction are proving very positive, allowing a more controlled gas management to take place. Goaf edge values of 0.3 % and purity considerably in excess of the explosive limits within the vacuum range are some of the physical achievements. Success in this area will not only allow a better gas capture in the extraction infrastructure, but also enable the ventilation system to be simplified to manageable and operationally comprehensible proportions.

Two papers of my Kazakhstan colleagues describe the measures currently being adopted to allow safe operation in a high gas level environment. The mines are governed by local mine legislation, which finds its origin in the Soviet Legislation. Many of the rules and regulations are technically sound for the operating environment. Credit is due to the Kazakhstan inspectorate and the unions, for being open minded as to different approaches to methods of work, and the introduction of "western" techniques.

An example is the use of stonedust. Not an issue in Australia, but what you are used to, is not a standard practice in the very wet Donbass and Kuzbass coalfields. The Karaganda field is dry, so the wet standards may not necessarily be readily applicable. The liberal use of stonedust would be one.

I have always regarded the Inspectorate as a friend, who is competent and can advise and act as part of his legal duties. This, in my view, is a major advantage Australia has over the American system, where the recording of violations appears to be a kPa. I believe that the issues and problems are too complex to play a cat and mouse game. By all means the Inspectorate needs to be the stick, but the door also has to be open allow progress to be made.

- 4) In any longwall mining situation, meeting the development rates are the key. With all the stone drivage, this is a major operational challenge for us in Kazakhstan. We still use steel arches supplemented by roof bolts. When questioned, the argument is sometimes voiced that arches

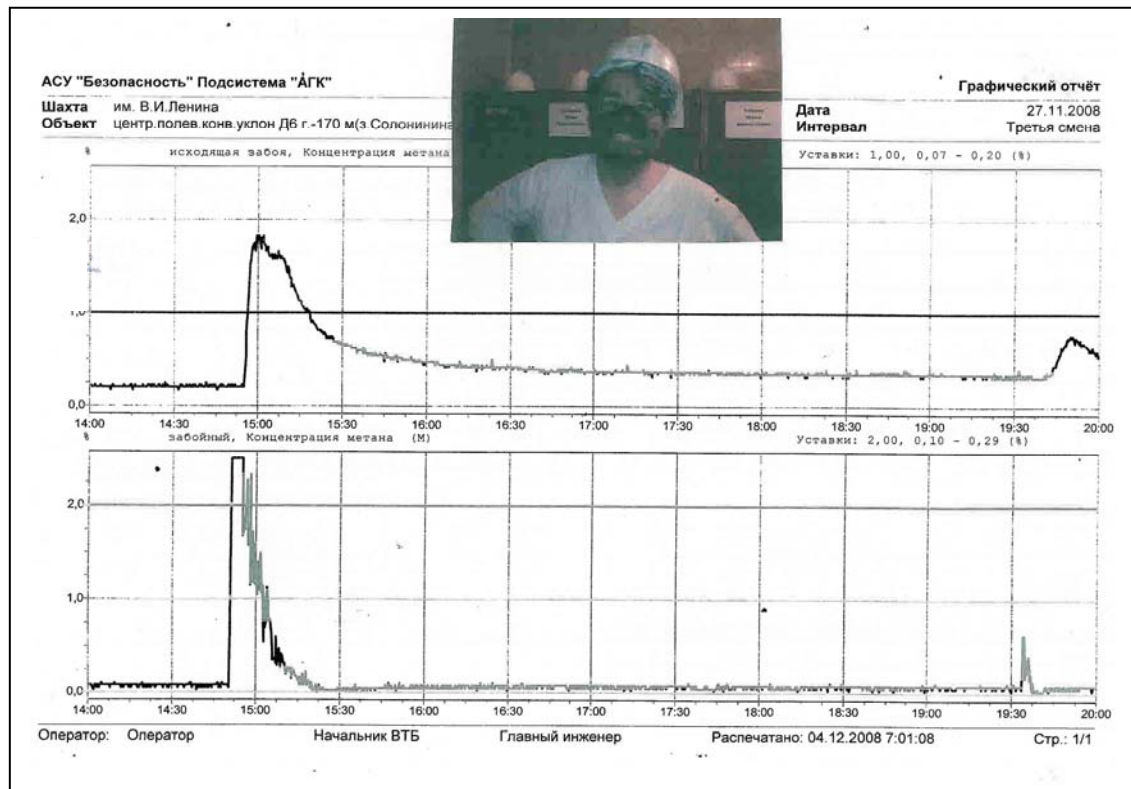
are easier to put up than bolts. In a geotechnical benign environment there appears little justification for arches just to hang the pipes on. With some basic computer simulation the case for more bolting may readily be made.

In the "west" we take computer based mine planning for granted. It is difficult for mine management to make "best practice" decisions when relying on manual planning systems. Unfortunately few of the software tools are Russian language friendly. It is happening, which will assist in promoting decision making and result accountability.

WHAT DO WE ALL WANT TO ACHIEVE FROM THE CONFERENCE

Lets talk and exchange experiences, learn for one another and together push back the geohazards which are only getting more significant as we go deeper and demand higher productivities from what we are doing.

I encourage the free interchange of ideas, theories and experiences at the operational level. All of us in the Industry have a vested interest in promoting safe working and achieving manageable conditions of coal extraction. No Company has the right to keep this to themselves for short term commercial gain. The stakeholders are not only the shareholders and employees, but also the community, which no longer tolerates unduly hazardous working conditions anywhere.



GEOTECHNICAL EVALUATION OF ROOF CONDITIONS AT CRINUM MINE BASED ON GEOPHYSICAL LOG INTERPRETATION

Peter Hatherly¹, Terry Medhurst², Genxi Ye³, and Dan Payne⁴

ABSTRACT: At the underground coal operators conference held in 2008, Payne described the experiences of crinum mine in characterising the weak roof strata at the mine. To a large extent, primary and secondary roof support strategies are based on UCS values determined from sonic logs. Consideration is also made of lithological units that can be identified on natural gamma logs. At crinum, UCS values in the roof strata tend to fall in the range 3-30 MPa.

Through ACARP funded research, a new method for evaluating geotechnical conditions known as the Geophysical Strata Rating (GSR) has been developed. The GSR is based on the interpretation of sonic, density and natural gamma logs and is designed to provide a measure of strata properties on a linear scale similar to that used in the Coal Mine Roof Rating (CMRR). GSR values are largely based on sonic velocity measurements and some degree of similarity therefore exists with the UCS determinations at Crinum. A comparison between the conventional UCS results at Crinum Mine and the new GSR determinations is made. The basis for relationships between sonic velocity and UCS is also discussed. Compositional factors and the range of depths over which a relationship is applied are important. The GSR takes these factors into consideration and provides an alternative and robust approach to estimating rock properties.

INTRODUCTION

Payne (2008) describes the geotechnical experience of Crinum mine for the period 1997 to 2007 during which time 14 longwall panels were mined and 40 million tonnes of coal was extracted. In that paper, the use of sonic logs obtained from exploration boreholes to predict UCS values was described. It is based on an empirical correlation between sonic log measurements and UCS measurements on over 150 core samples. Sections and maps showing the variations in the estimated UCS are produced for the floor and roof of the working (Lilyvale) seam. Primary and secondary support decisions are made using these and they form an essential part of the mine's on-going strata management and hazard plans. Roof units with strengths less than 10 MPa represent the major roof control issue at Crinum.

Ward (2007) provides guidelines for the determination of the UCS from the geophysical logs. The main roof sequence consists of sandstones and siltstones and four main units (plus sub units) can be recognised on natural gamma logs. For each of these units, representative sonic transit times are scaled off the geophysical logs and converted to UCS. The roof sections are produced over a 12 m section along roadways and show the location of the stratigraphic units and their UCS values. Plan maps of average UCS are produced for immediate roof sections of 0.5 m and 2 m thicknesses. Crinum Mine has thus established an effective and successful method for the prediction and management of their strata conditions.

In a separate development, Medhurst and Hatherly (2005), Hatherly et al. (2007) and Hatherly et al. (2008) have introduced the Geophysical Strata Rating (GSR). The GSR is a means of empirically estimating the quality of rock masses. It is based mainly on sonic log data but unlike the usual approach of converting sonic logs to UCS through site-based relationship, the GSR is designed to work in all forms of clastic sedimentary rocks. It compensates for the variations in sonic velocity caused by factors such as changes in rock composition, changes in depth and the presence of bedding surfaces and fractures. Through this consideration of both the intact rock and the defects, the GSR has a

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geomechanical basis similar to the conventional rock mass rating schemes. The GSR rating is on a linear scale of up to 100 and can therefore be compared to the Coal Mine Roof Rating (CMRR).

One of the advantages of the GSR is the generality of the formulation. Another is the fact that it does not require manual assessment of geophysical logs. A computer based analysis is required but this process is not onerous and it provides useful geological information in its own right.

Current research into the GSR is directed towards (i) comparing the performance of the GSR with other geotechnical procedures, (ii) extending the rating to include coal, and (iii) providing a method for establishing 3D geotechnical models from borehole geophysical logs and seismic survey data (if available). Crinum Mine provides an ideal site for a comparison between GSR and a proven method involving UCS estimates. In order to provide a basis for the rationale behind the formulation of the GSR and the reasons for relationships between sonic velocity and UCS, some fundamental issues concerning sonic velocity is discussed.

SONIC VELOCITY AND UCS

From the theory of elastic wave propagation, it is known that in homogeneous rocks, the velocity, V_p , of the P-wave that is measured by sonic logging is given by:

$$V_p = \sqrt{\frac{K + 4/3\mu}{\rho}} \quad (1)$$

Where K is the bulk modulus (incompressibility), μ is the shear modulus, and ρ is the density.

In reality, most rocks are not homogeneous on account of compositional variations and defects. Another factor in sedimentary rocks is the existence of anisotropy due to bedding - velocity depends on whether the measurements are taken across the beds or along them. The elastic parameters K and μ , and the density are therefore variable. In order to properly interpret sonic logs, it is necessary to understand the influence of the various causes of inhomogeneity and anisotropy.

In fresh igneous rocks where the porosity is low and the crystals have similar elastic properties, the main cause of inhomogeneities are fractures and joints. These have a major influence on the velocity. Barton (2006) refers to Sjögren et al. (1979), who correlated fracture frequency and RQD with measurements of V_p from shallow seismic refraction surveys in Norwegian igneous and metamorphic rocks. This correlation is shown Figure 1. As is also reported by Barton, Deere et al. (1967) found a relationship between RQD and the square of the ratio of V_p measured in the field (which is affected by fractures) and V_p measured in the laboratory on intact samples.

In the case of sedimentary rocks compositional variations also affect the homogeneity. From petroleum exploration it is known that V_p is affected by composition, porosity and pressure. In these circumstances it is not possible to determine exact expressions for V_p , but laboratory studies have allowed development of empirical relationships. Equation 2 is an extremely useful expression obtained by Eberhart-Phillips et al. (1989), which includes terms for fractional porosity (ϕ), fractional shale content (V_{shale}) and effective pressure, p_e (confining pressure minus the pore pressure).

$$V_p = 5.77 - 6.94\phi - 1.73\sqrt{V_{shale}} + 0.446(p_e - e^{-16.7p_e}) \quad (2)$$

Velocity here is measured in km/s and the effective pressure is measured in kilobars.

When considering the sonic/UCS relationship, Equation 1, shows that there is a theoretical relationships between V_p and modulus provided inhomogeneities and anisotropy are not an issue. There is no such relationship between V_p and UCS. However, because it is generally observed that stiffer rocks with higher modulus are stronger and because velocity is related to modulus, empirical estimates of UCS can be made from V_p .

In the context of the use made of the empirical relationships between UCS and V_p in Australian coal mining, it can therefore be seen that sonic/UCS relationships need to be established for a selection of rock types where the compositional variations that affect the velocity also affect the modulus and the range of depths is limited. It would appear that this is the case at Crinum Mine because sonic logging

has proved to be a very useful tool for estimating the UCS of key intervals surrounding the working seam.

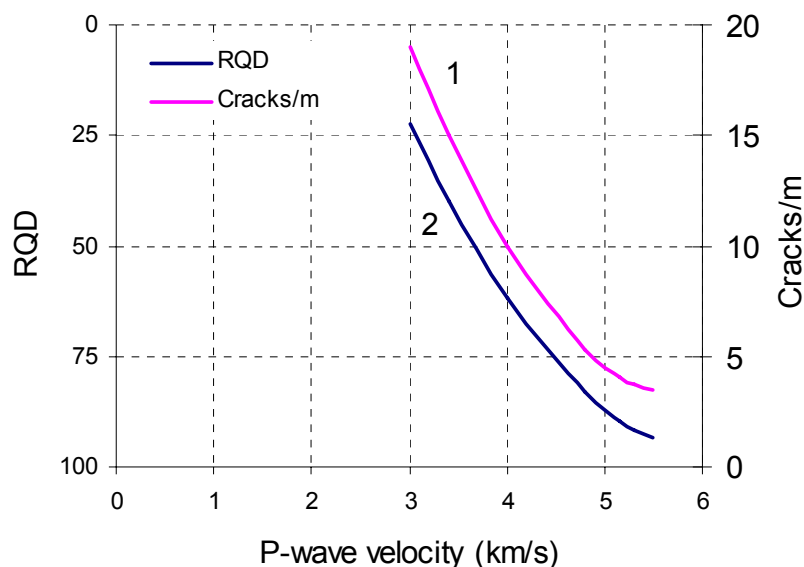


Figure 1 - Correlations of fracture frequency (1) and RQD (2) with P-wave velocity for mainly crystalline and metamorphic rocks from Scandinavia (redrawn from Sjøgren et al., 1979)

THE GEOPHYSICAL STRATA RATING

The full definition of the GSR is given in Hatherly et al (2008). In addition to sonic log values, it requires an interpretation of the geophysical logs to provide an assessment of the porosity and clay contents of the strata of interest. The log interpretation procedure follows the standard procedures described by Hatherly et al (2003). The velocity values are also corrected for the effects of depth.

For a GSR measurement, scores are provided for the following:

- rock strength (score between 0 and 55, depending on velocity)
- rock cohesion (score between 10 and 25, depending on velocity, clay content and porosity)
- porosity (score between 0 and -15, when clay content is less than 35% and depending on porosity)
- shaliness (score between 0 and -10, when clay content is greater than 65% and depending on porosity)
- presence of lithological (bed) boundaries (score between 0 and 10, with an inverse dependence on the down-hole variability of the clay measurements)
- presence of defects (score between 0 and 10, with an inverse dependence on the down-hole variability in the sum of the first 4 scores).

These terms have geomechanical significance. The first four can be added together to give an estimate of the quality of individual beds – the initial GSR (GSRI). The effects of the variability due to defects and changes in the bedding are captured by the last two terms. However, given the association between velocity and fracturing indicated in Figure 1, the score for the GSRI is also affected by defects in the form of fractures and fine bedding.

One aspect of the log interpretation procedure that is unconventional, concerns the use of Equation (2) to check the values of the porosity, clay content and velocity/depth gradient. To determine these parameters, the standard geophysical log interpretation procedures require estimation of the rock grain density, the natural gamma response in pure sandstones and the natural gamma response in pure clays. If these, so-called endpoints have been accurately estimated then substitution into Equation (2) of the calculated porosities and clay contents, together with an estimated effective pressure gradient

should lead to a calculated velocity that matches that observed in the sonic log. If the calculated and observed velocities do not agree, different endpoints are needed.

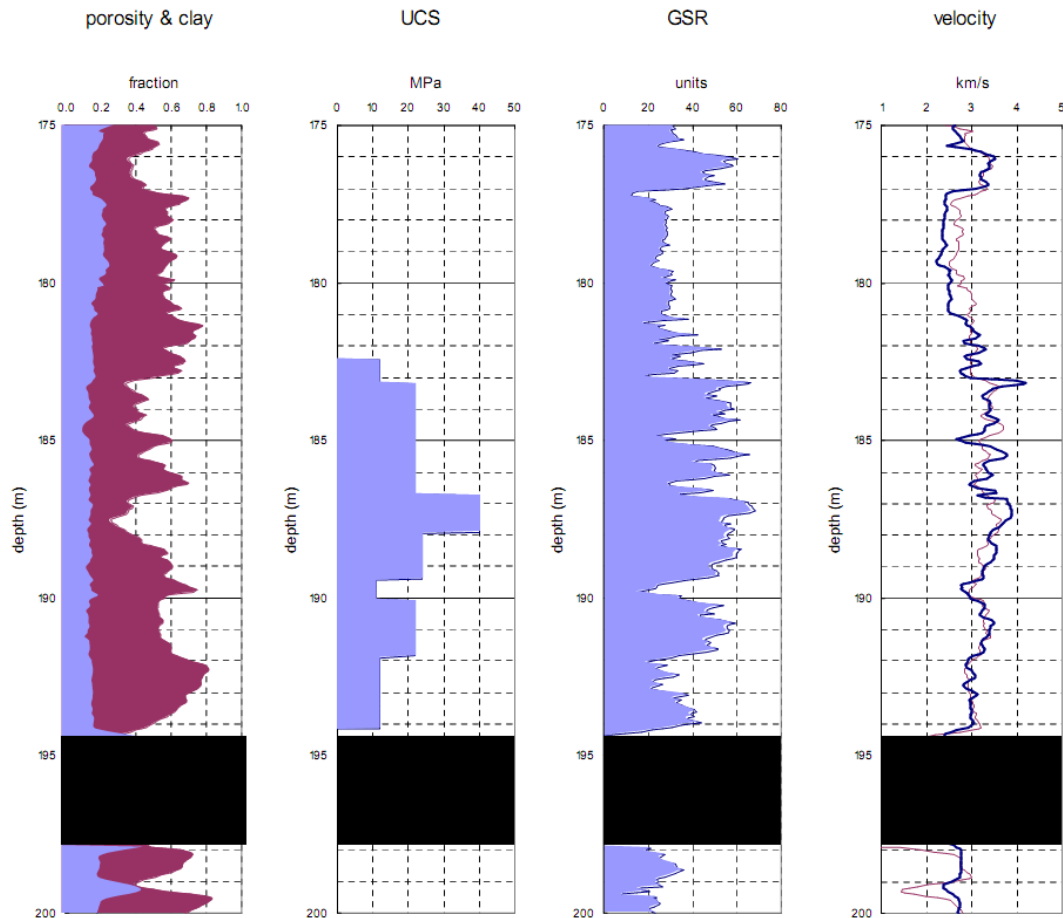


Figure 2 - Geophysical log analysis for Crinum borehole 05335. From left to right, (i) porosity (blue) and clay (maroon) values determined from density and natural gamma logs respectively, (ii) UCS values obtained using the conventional approach, (iii) GSR values and (iv) velocity logs – the thick blue line shows the measured sonic velocity. The thin maroon line is the velocity calculated using Equation 2. The higher clay values correspond to the more silty units.

If agreement is not possible, then it is possible that there are problems with the geophysical logs such as the density log being out of calibration, or the sonic log being inaccurate due to cycle skipping. Another possibility is that the geology is unusual. For example, the sandstones may have elevated natural gamma responses. If the problem is with the natural gamma logs or anomalous gamma radiation, then it may be possible to estimate the clay contents using alternative log combinations (neutron/density and resistivity/density). If the problem is with the density or sonic logs, then determination of the GSR will be unavoidably compromised. The process of calculating velocities can thus be seen as a QC step that can be used to improve the interpretation and to recognise problems in the data.

Equation (2) has another use because it allows the calculation of sonic velocities in situations where sonic logs are not available – e.g. above the standing water level in boreholes. Substitution of values for the porosity, clay content and effective pressure allows calculation of velocity and hence GSR. However caution is required because QC of the log interpretation is not possible.

As a final point, it is noted that there is a parallel between the GSR formulation and the scheme developed by Barton (2002, 2006) where it is proposed that Q-value can be estimated from seismic (sonic) velocity, after compensation for depth and porosity. The GSR is conceptually similar except that it includes the additional consideration of the effect of compositional variation as given by the clay content and required by Equation (2).

GSR ANALYSIS AT CRINUM

To illustrate the performance of the GSR, Figure 2 shows results for a single hole, 05335, over a 25 m section which embraces the Lilyvale seam. Also shown is the UCS analysis obtained by the standard Crinum procedure.

In Figure 2, the clay content has been determined from the natural gamma log and the porosity has been determined from the density log. Using these values and a suitable pressure gradient (15 MPa per km), velocities have been calculated and are shown to be in reasonable agreement with the observed sonic log. The sonic values, porosities and clay contents are therefore deemed to be suitable for calculating GSR values.

Comparing the UCS and clay results, it can be seen how the clay variations have been used to identify the lithological intervals over which the UCS values were determined. Note that in this case, the sonic velocity alone does not provide this same insight.

Examination of the GSR plot, however, conveys a clear sense of the locations of the strong and weak units within this section of the hole and their relationship to the lithologies indicated by the clay values. This illustrates that the GSR is able to respond to variations in rock type as well as the smaller discrete variations detected by the continuous sampling of the geophysical logs.

Figure 3 compares the GSR and the blocked UCS results for the takeoff roadways for longwalls 6 to 12. Lithological boundaries have been interpolated between holes and actual UCS values are given in the circled text. The colour scheme for the UCS section is lithologically based and aims to provide a guide to the geotechnical variability. In general, units coloured red, orange and yellow are weak and those coloured green and blue are strong.

In the case of the GSR results, the gridding and imaging program Surfer¹ has been used to interpolate the data between holes and a colour scheme has been chosen to convey the same meaning as in the UCS section. Red, orange and yellow correspond to low values of GSR. Green and blue are for the higher values. The similarity of the UCS and GSR sections is immediately obvious. Clearly, the GSR results over these longwall take-off roadways could have been used in the same way as the mine has utilised the UCS section.

DISCUSSION AND CONCLUSIONS

In this paper we have discussed the geological influences on seismic velocity and provided reasons for the empirical relationships between UCS and velocity. Strictly speaking, this only occurs if the elastic properties are related to strength and we therefore suggest that sonic/UCS relationships should only be applied in situations where the rock types present are not highly variable and there is no overlap in the UCS values of rocks with differing composition. The range of depths over which the relationship is applied should also be restricted.

The GSR has been designed to compensate for these limitations. The results from Crinum Mine illustrate the application of the GSR as a means of automatically assessing strata conditions. They show geotechnical variations that are similar and arguably an improvement to the UCS values that Crinum Mine has derived on the basis of sonic velocity and lithological considerations. The GSR has thus been demonstrated to be an alternative approach to obtaining geotechnical data.

Amongst the benefits of using the GSR are its objective nature and the automatic approach it provides for analysing geophysical borehole data. The method can be applied at any coal mine site and it eliminates the need to establish mine-based sonic/UCS relationships. Results can also be used in conjunction with other classification schemes. It is not difficult to include more data when they become available and to extend the range of interest if required.

¹ <http://www.goldensoftware.com/products/surfer/surfer.shtml>.

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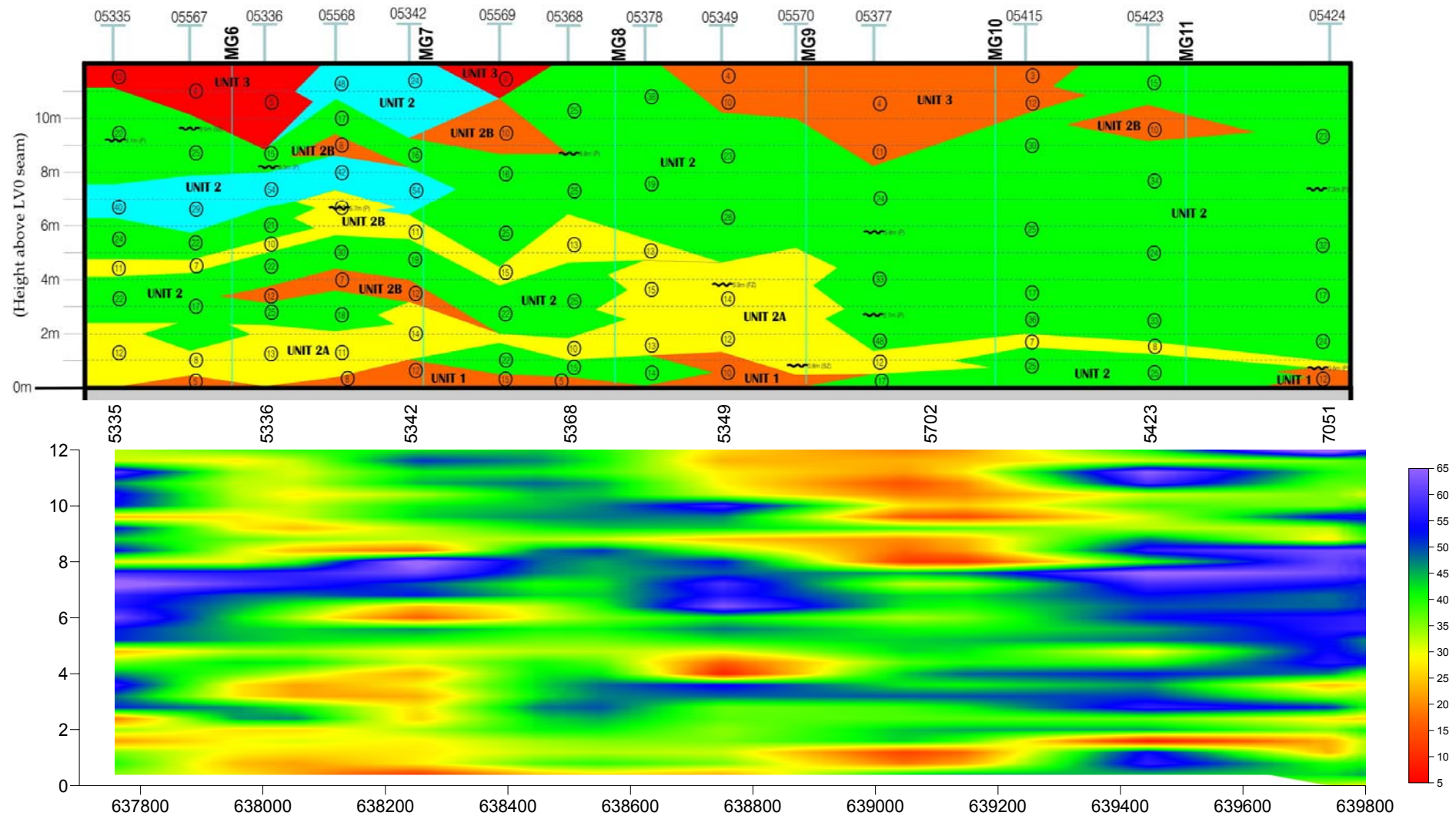


Figure 3 - Comparison of UCS (top) and GSR (bottom) values over a 12 m roof section across 2 km of longwall takeoff roads. The boreholes utilised for each are indicated at the top of each section.

DEVELOPMENT OF A PRE-DRIVEN RECOVERY EVALUATION PROGRAM FOR LONGWALL OPERATIONS

David Wichlacz¹, Tim Britten² and Basil Beamish¹

ABSTRACT: Many longwall coordinators are examining the use of pre-driven recovery roadways. This method, if performed successfully can improve the overall efficiency and safety of moving longwall equipment from panel to panel. However, it is difficult to assess the feasibility of using pre-driven recovery unless extensive research is carried out or a consultant is used to analyse the particular situation. A number of previous case studies have been analysed to discover which parameters have the greatest influence on the success of pre-driven recovery. Floor strength, Coal Mine Roof Rating (CMRR), extraction depth, Roof Density Index (RDI), standing support and mining rate were the main parameters impacting on the successful implementation of pre-driven recovery roadways. These parameters have been incorporated into a program that was developed to assess the feasibility of using pre-driven recovery roadways. The Pre-driven Recovery Evaluation Program (PREP) is simple to operate and it will enable new longwall mining operations as well as current operations to quickly determine the suitability of the method to their site.

INTRODUCTION

Pre-driven recovery is a very important aspect of longwall mining. Pre-driven recovery rooms are used to safely remove longwall equipment once extraction of a panel has been completed. This method, if performed successfully can improve the overall efficiency and safety of shifting longwall equipment from panel to panel. Pre-driven recovery can significantly reduce the longwall downtime due to panel change, and therefore considerably improve the profit margin of the operation.

The method of pre-driven recovery is slowly replacing the conventional method and to-date over 100 full or partial pre-driven recovery roads have been used in the US, Australian and South African underground coal mines (Thomas, 2008). The majority of these cases have proven to be extremely successful and improved the overall efficiency and safety of the operation as they were performed under appropriate strata conditions using the correct ground support. However, it is difficult to evaluate the feasibility of using pre-driven recovery unless carried out by strata specialists.

Analysis of past case studies has been used to discover the main parameters that influence the success of pre-driven roadways. These parameters have been incorporated into a program to assess the feasibility of using pre-driven recovery for a given situation and hence improve the overall safety and efficiency of longwall equipment recovery.

CURRENT LONGWALL RECOVERY METHODS

The moving of longwall equipment from one panel to the next is a critical efficiency issue to any longwall operation. The two main longwall recovery methods that have been practiced to date are the conventional and pre-driven recovery methods.

Conventional Method

Currently the average move time for longwall operations is around 20 days, while move times as long as 30 days are recorded. The move time can vary depending on face width, panel length (distance of move), experience of mine personnel, and amount of equipment installed on the new face prior to start up of the actual move (Bauer *et al.* 1989). The preparation for the conventional recovery method usually occurs at 15 m from the extraction point. The roof of the mine is supported by installing bolts and wire mesh along the longwall face at the end of each panel advance (Bauer *et al.* 1989). The bolts are installed either by hand-held drilling equipment or specialised single boom bolters designed specifically for this application (Tadolini, Zhang and Peng, 2002).

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Figure 1 illustrates a sectional view of the conventional recovery method.

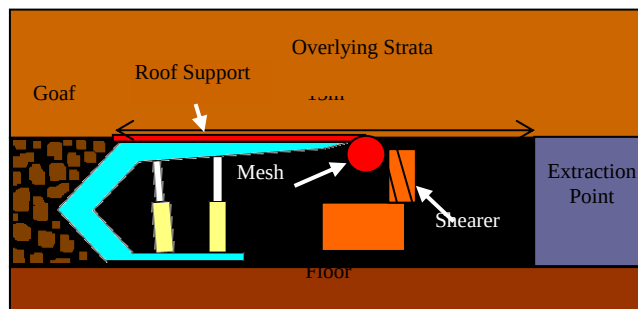


Figure 1 - Schematic of conventional recovery method

Pre-driven Recovery Roadways

As an alternative to the conventional method, mines have investigated and utilised pre-driven recovery rooms for longwall face moves. In this method an entry is developed and supported ahead of time so that the required combinations of standing and internal support can be installed prior to the longwall face approaching the extraction point.

The roadway is created using a continuous miner and generally has a width of around 5 m, however the width can range up to 12 m depending on the size of the equipment being recovered and ground conditions (Science Communication Services, 1990). The longwall is then able to extract the remaining fender at full speed before holing into the recovery roadway. Figure 2 represents a cross-sectional view of the pre-driven recovery method.

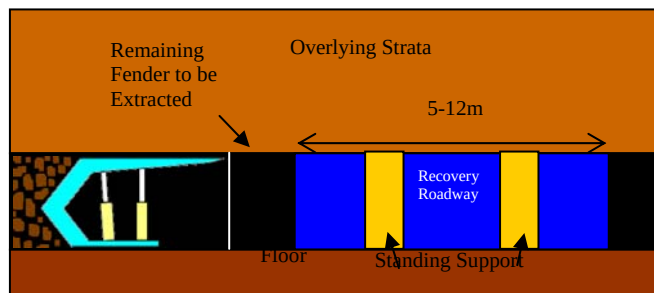


Figure 2 - Schematic of pre-driven roadway recovery

Strata behaviour during pre-driven recovery

It was discovered that once the unconfined fender has yielded, it will not carry any appreciable load owing to its poor post failure strength (Science Communication Services, 1990). At this stage, the supports and barrier pillar edge bear the load of a 16 m long cantilever and the roof strata behaves as though the face was at the edge of the barrier pillar (Figure 3). Consequently, the strata above the longwall face undergo various degrees of tensile failure ahead of the face line. As the remaining fender yields, the tensile strains are transferred to the barrier pillar edge and are considerably increased. At this point, if the cantilevered roof cannot support itself and fails, the shields will be carrying the majority of the load (Science Communication Services, 1990).

To improve the success of the recovery, it is therefore recommended that the longwall is checked and given a major service thirty metres before the pre-driven road is to be holed. From this point onwards the longwall is operated continuously until the recovery road is holed. The reason for this is to attempt to keep the front abutment pressure moving across the roadway into the solid outbye pillar. This is considered to be a vital element in the approach.

Suitable ground conditions

McCowan and Hornby (1989) found that the use of full length pre-driven recovery roads under laminate or mudstone/siltstone roof strata was a high risk operation from which the derived benefit could not be justified. It was found that if the cantilever fails through the weak intact rock or an inherent geological feature, then loads will be produced that exceed the support capacities. Therefore, the use

of full length pre-driven recovery roads is not recommended under a laminate or mudstone roof without adequate passive support (McCowan and Hornby 1989).

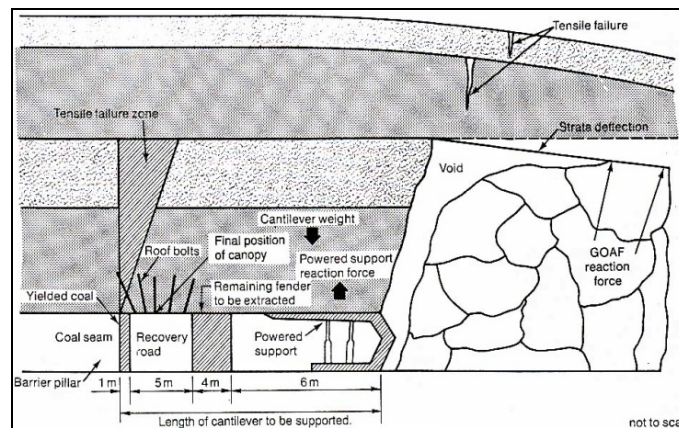


Figure 1 - The structures and forces involved in roof failure associated with a full-length, pre-driven recovery road (Science Communication Services, 1990)

From research conducted by ACIRL (Science Communication Services, 1990) it was discovered that laminated strata have a lower shear modulus than more massive strata, which in turn results in more flexibility within the strata. This increased bending results in greater loading on the fender, leading to greater increased face and recovery road-rib side yielding. The reason for this is that the laminated roof is less able to support itself along with the overburden. As the overlying stratum is unable to support itself, higher capacity supports are required for softer or laminated roof types (Science Communication Services, 1990). Therefore it is reasonable to suggest that laminate or mudstone/siltstone roof strata are unsuitable ground conditions for pre-driven recovery roads unless adequate passive support is installed.

Selection of support design

Peng (2007) found that the safest support design is to use a combination of standing supports and roof bolting. Both the roof bolting and standing support systems can be designed to independently support the room, or combined as a system, thereby utilising their individual advantages. Experiences have shown that with proper design this system can ensure a successful outcome (Peng, 2007).

The combination of the internal bolting system and the standing concrete supports are critical for the successful and safe recovery of the longwall equipment. A combined support system that is too soft or too stiff can result in excessive recovery room closures or brittle failures of the concrete crib systems (Tadolini, 2003). Figure 4 shows the installation of pumpable cribs. Note that a bag containing a softer material has been placed between the crib and the roof to allow for the cribs to be slightly compressed.

It is however possible to successfully recover equipment from a pre-driven roadway without the use of standing support. This has been proven at the US Steel, Mine 50 where traditional methods involving standing support are not feasible due to the difficulties of the plough face mining through these types of support (Smyth *et al*, 1998). Smyth *et al*. (1998) stated that recent development of new cable support systems provides an option for cut through entry support. Mine 50 and Jennmar Corporation personnel have worked together to design and apply the cable systems in the cut-through entries and full face recovery room to eliminate standing support. To date, a number of full face recovery rooms and cut-through in-panel entries have been successfully mined (Smyth *et al*. 1998)

DATA ANALYSIS

In recent years, the National Institute for Occupational Safety and Health (NIOSH) compiled a comprehensive international database of past case histories of parameters associated with pre-driven recovery roadways (Oyler *et al*, 1998). An analysis of the NIOSH data was conducted to identify the various mining parameters that lead to the overall success of pre-driven recovery roadways.



Figure 4 - Installation of pumpable cribs (Peng, 2007)

Effect of mining parameters on pre-driven recovery roadways

The mining parameters that were analysed included:

- Floor strength
- Depth of extraction
- Coal mine roof rating (CMRR)
- Seam height
- Panel width
- Room length
- Shield capacity
- Roof density index (RDI)
- Standing support
- Advance rate

The outcome of the pre-driven roadways were categorised into three groups. These were:

- Successful outcome
- Failure due to face break or roof fall
- Failure due to major overburden weighting

Each parameter was individually graphed to observe the impact that it had on the success of this mining method. From the data analysis it was found that floor strength, depth of extraction, CMRR, RDI, standing support and advance rate were parameters that had the most influence on the success of pre-driven roadways.

Floor Strength

A higher percentage of weighting failures occurred in mines with soft floor conditions as seen in Figure 5. In some conditions where the fender pillar is thin and likely to punch into the floor, the potential for failure may be increased due to soft floor conditions. However in some successful soft floor cases, the soft floor conditions were credited with delaying the fender yield and therefore contributing to the success of the recovery room.

Depth of Extraction

A wide range of cover depths from case histories were included in the analysis. However there was no strong relationship found between depth of extraction and major failure due to overburden weighting (Figure 6). Generally, it can be seen from Figure 6 that failures due to face breaks or roof falls were somewhat more likely to occur at greater depths. This is most likely due to the increase in horizontal stresses on the surrounding strata of the pre-driven recovery roadway in the deeper mines. Also due to the increase in horizontal stress, deeper mines typically install higher densities of roof reinforcement to help compensate for these stresses.

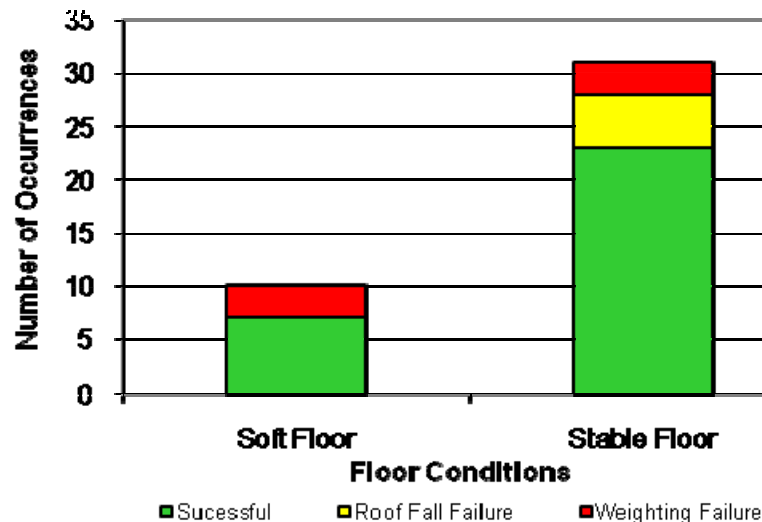


Figure 5 - Histogram showing impact of floor conditions on recovery roadway outcomes

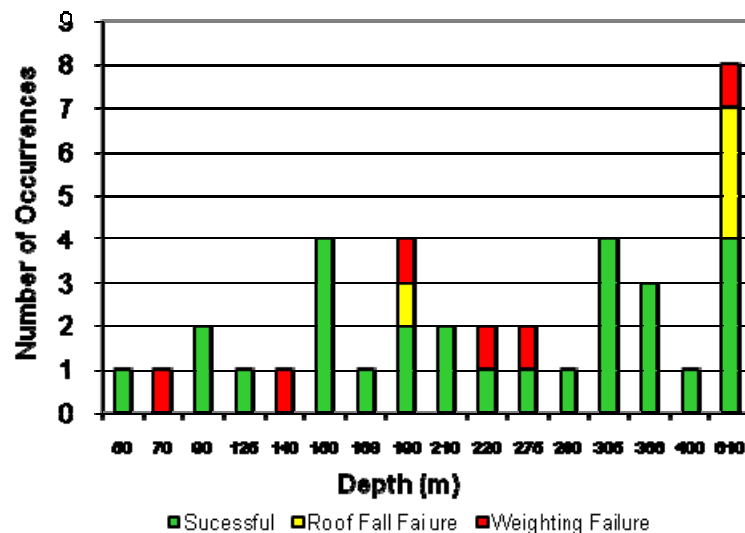


Figure 6 - Histogram showing impact of depth of cover on recovery roadway outcomes

Coal Mine Roof Rating

A very strong correlation was found between CMRR and weighting failures as showed in Figure 7. All of the six weighting failures occurred where the roof was relatively weak (CMRR < 50). This provides an indication that if the CMRR is less than 50 then unless the roof is heavily supported; weighting failures are likely to occur. However there is less evidence of roof falls being related to roof strength.

Mining Rate

A slow mining rate seemed to be a strong predictor of both types of failure. It can be seen in Figure 8 that even though the same amount of failures occurred a higher percentage of failures were associated with slow mining rates compared to normal mining rates.

This is why it is extremely important that the longwall is stopped around 30m before the fender pillar and is fully serviced. This ensures that breakdowns are less likely to occur which will reduce the advance rate of the longwall face during this critical stage of operation.

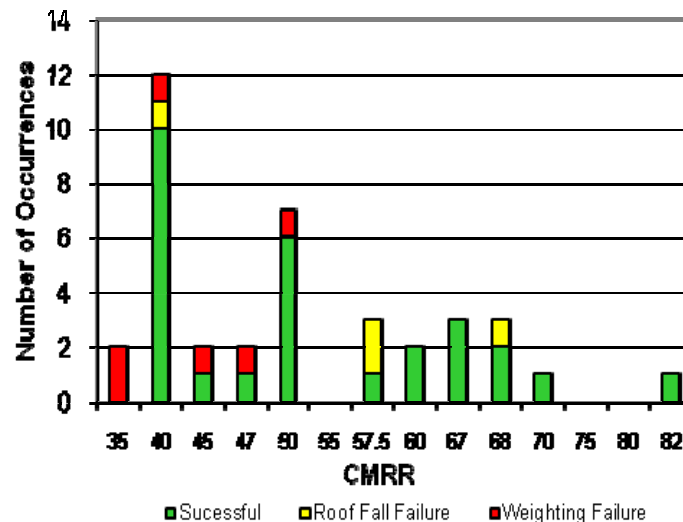


Figure 7 - Histogram showing impact of CMRR on recovery roadway outcomes

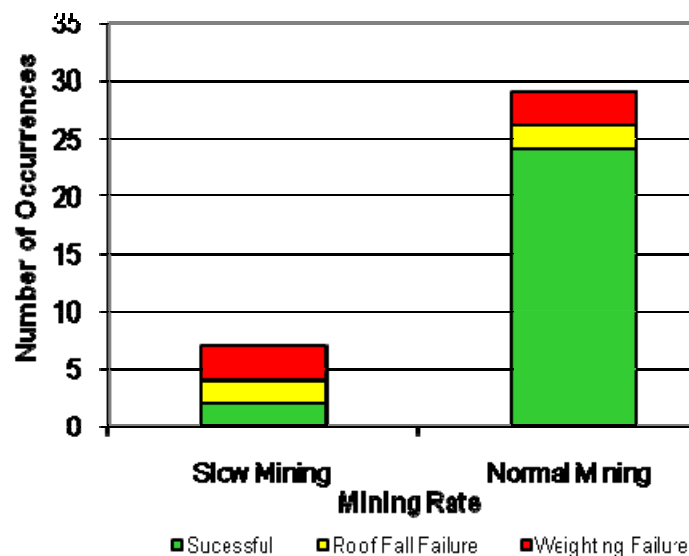


Figure 8 - Histogram showing impact of mining rate on recovery roadway outcomes

Roof Reinforcement

Roof reinforcement includes all intrinsic support elements such as roof bolts, cable bolts, and trusses. The reinforcement is quantitatively measured by determining the load capacity of each element per unit area of roof supported by the element and multiplied by the length of the element.

This Reinforcement Density Index (RDI) has the units of MPa.m. It can be observed from Figure 9 that heavy roof reinforcement was apparently successful in reducing the incidence of roof fall type failures. However roof reinforcement was not successful in preventing weighting failures.

Standing Support

Figure 10 shows that standing support has a dramatic influence on the success of pre-driven recovery rooms. A characteristic of every one of the weighting failures is the lack of standing support.

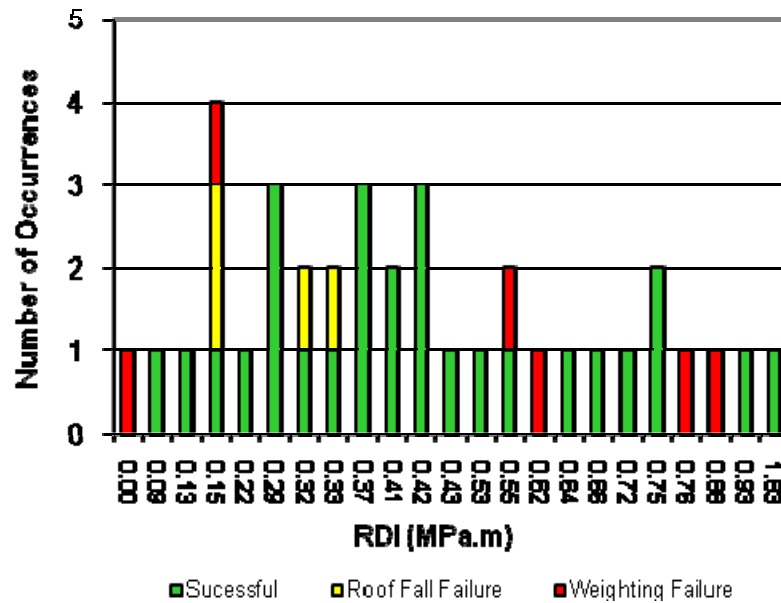


Figure 9 - Histogram showing impact of RDI on recovery roadway outcomes

It has been recorded in two instances where after a severe weighting failure developed in a room without standing support, adjacent rooms were mined successfully with standing support (Oyler *et al*, 1998). From these cases it has been indicated that standing support can be the difference between success and failure of a pre-driven recovery operation.

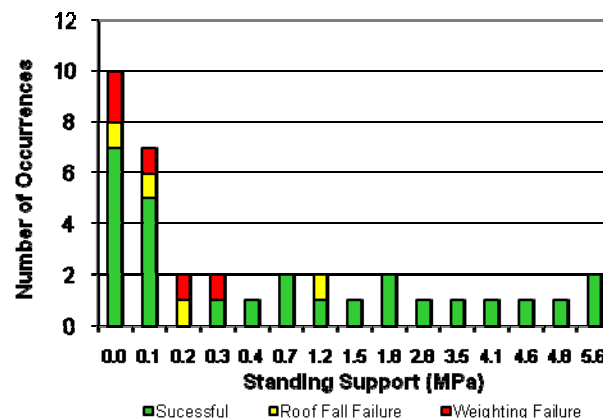


Figure 10 - Histogram showing impact of standing support on recovery roadway outcomes

Multivariate analyses

A multivariate analysis was used to obtain a possible insight into the parameters that most influenced pre-driven roadways and help set particular design guidelines.

CMRR and Standing Support

Weighting failures are closely associated with CMRR and standing support as shown by Figure 11. A highly significant relationship indicates that when the CMRR was greater than 50, little support was necessary. It can be seen from Figure 11 that for a CMRR equal to 40, the successful cases used a standing support density of at least 1.0 MPa. For CMRR values ranging from 45-50, standing support densities as low as 0.5 MPa were sufficient in preventing or controlling weighting failures. However the cost of standing support is small compared to the cost of a weighting failure, and therefore it is recommended that the observations from Figure 11 should not be taken as a recommendation to estimate standing support.

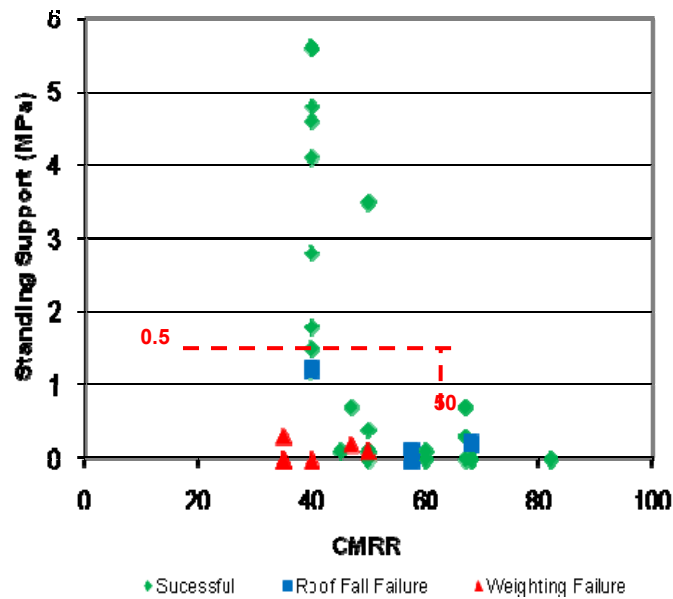


Figure 11 - Combined influence of CMRR and standing support density on recovery roadway outcomes

RDI and Standing Support

From the multivariate data analysis it was discovered that the majority of pre-driven roadway failures were associated with either a low density or no standing support and a low RDI (Figure 12). The majority of the failures were associated with roadways where the standing support density was less than 0.5 MPa. In terms of roof reinforcement, the majority of pre-driven roadway failures were recorded where the RDI was less than 0.8 MPa.m.

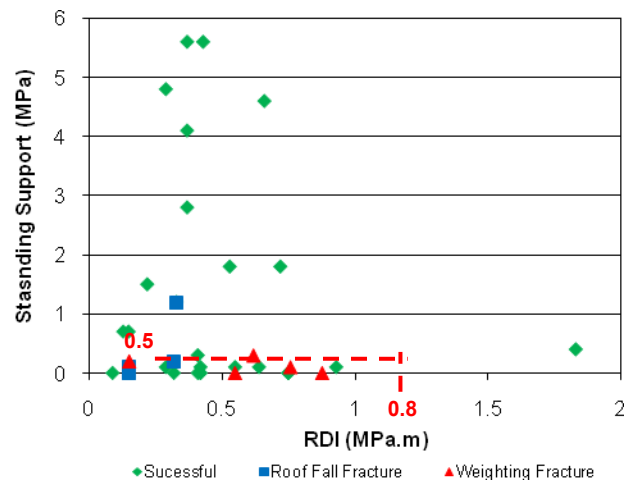


Figure 12 - Combined influence of RDI and standing support on recovery roadway outcomes

Figure 12 indicates if a particular pre-driven roadway has standing support less than 0.5 MPa and a RDI less than 0.8 MPa.m, then the likelihood of failure is dramatically increased. It was recorded that 8 out of the 20 cases (40%) that used a combination of support in this range encountered failures of pre-driven roadways.

From the analysis it can be observed that the majority of successful cases used high densities of standing support to counteract the need for high densities of roof support. On the other hand, high densities of roof support have also been used to offset the need of high densities of standing support.

PRE-DRIVEN RECOVERY EVALUATION PROGRAM

Microsoft Excel was used to develop a program to evaluate the feasibility of using pre-driven recovery based on the most significant parameters identified as having an influence on a successful outcome.

Development of the program

The final version of the Pre-driven Recovery Evaluation Program (PREP) can be seen in Figure 13.

The evaluation program was created using the Developer tab in Microsoft Excel 2007. From the data analysis six of the most influential parameters of pre-driven roadways were selected to be incorporated into the evaluation program. Scroll bars and list boxes were incorporated into the program to allow the user to clearly see what option they have chosen and reduce the chance of selection errors from occurring.

An evaluation of the inputs into the program is provided so that the user can clearly understand the feasibility of their particular pre-driven roadway. The evaluation has four possible outcomes based on the total value from each of the parameters:

- Strongly Recommended (100-75)
- Recommended (75-50)
- Not Recommended (50- 25)
- Strongly Not Recommended (25- 0)

Also a bar graph was incorporated into the program and linked to the overall result to provide a visual rating out of one hundred.

The data for each parameter along with the formulas used to calculate the overall rating was hidden in a second sheet so that the user of the program would not be confused by the data. For each of the parameters a formula was applied to weight the data based on the importance and overall impact that each particular parameter had on the overall success of the pre-driven roadway.

Conditional Formatting was also applied to the program to give the user a visual response to the result of the recommendation. A colour scheme ranging from red (strongly not recommended) to green (strongly recommended) was used as this gave a recognisable and distinct indication of the result.

Clear instructions on how to activate the program are also provided. In order to use the program the developer tab must be activated followed by enabling the macros. The file must then be closed and re-opened before the program can be used.

Figure 13 represents the best case scenario for the pre-driven recovery evaluation program. The best case is given when the following parameters are entered into the program:

- CMRR is high
- Floor strength is Normal (>20MPa)
- Depth of Extraction is shallow
- RDI is high
- Standing Support is high
- Mining Rate is high

Testing the Program

Values from the case histories were entered into the program to ensure that it provided the correct result.

Pre-driven Recovery Evaluation Program

*In order to use the program the Developer Tab must be firstly activated. This can be done by:
File (Office Button) → Excel Options → Tick the Box Show Developer Tab in the Ribbon → OK
Next the Macro Security must be turned off. This can be done by:
Clicking on the Developer Tab → Macro Security → Enable All Macros → OK
The File must be then closed and re-opened for the settings to be activated

Select a value for each of the characteristics below

Floor Strength	Normal	
CMRR	35 90	90
Extraction Depth (m)	50 610	50
RDI (MPa.m)	0 1.83	1.83
Standing Support (MPa)	0 5.6	5.6
Mining Rate (Shears/h)	0.1 1.5	1.5

Strongly Recommended

100

0 10 20 30 40 50 60 70 80 90 100

Figure 13 - Pre-driven Recovery Evaluation Program

Example of a Successful Outcome

The parameters from Table 1 were entered into the Pre-driven Recovery Evaluation Program.

Table 1 - Parameters for a successful outcome

Country	Australia
State	NSW
No. of Rooms	4
Soft Floor	No
Depth (m)	290
CMRR	50
Seam Height (m)	3
Panel Width (m)	200
Room Length (m)	200
Shield Capacity (t)	590
RDI MPa.m	1.83
Standing Support MPa	0.4
Slow Mining	No
Outcome	Successful

Figure 14 shows that the program provided the correct result for the given parameters and it is recommended that pre-driven recovery be used.

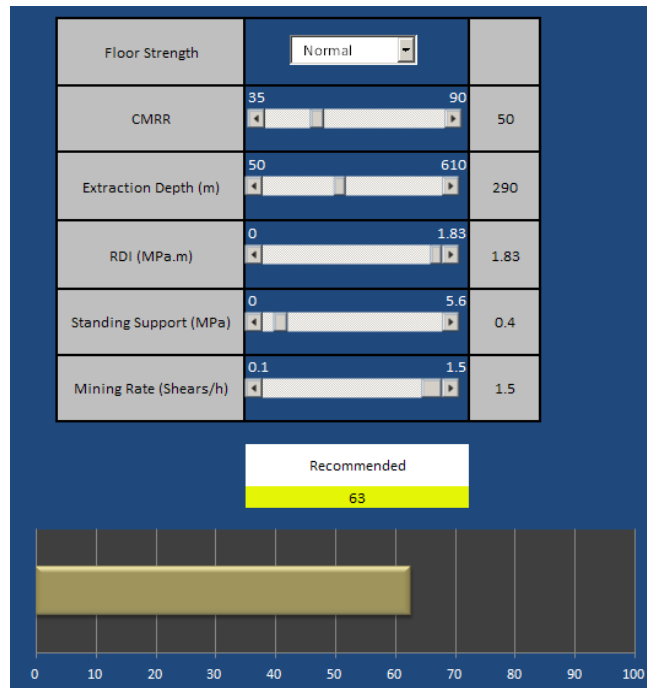


Figure 14 - Results of a successful outcome

Example of Failure Due to Face Break or Roof Fall

The parameters from Table 2 were entered into the Pre-driven Recovery Evaluation Program.

Table 2 - Parameters for failure- face break or roof fall

Country	US
State	MD
No. of Rooms	1
Soft Floor	No
Depth (m)	190
CMRR	40
Seam Height (m)	2.6
Panel Width (m)	229
Room Length (m)	229
Shield Capacity (t)	599
RDI MPa.m	0.33
Standing Support MPa	1.2
Slow Mining	Yes
Outcome	Failure

As shown by Figure 15 the program provided the correct evaluation and gave a result of 'Not Recommended'. This was mainly due to the fact that the majority of the parameters were less than average and the mining rate was slow.

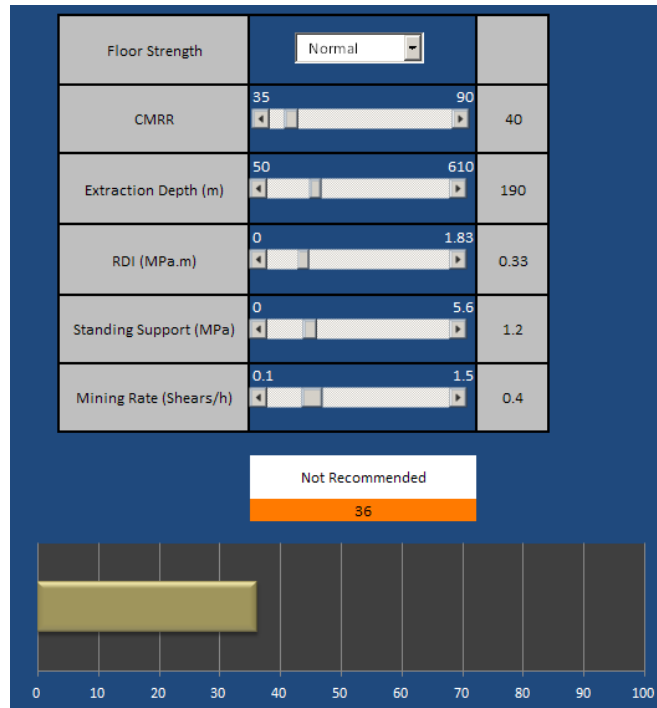


Figure 15 - Results for failure due to face break or roof fall

Example of Failure Due to Major Overburden Weighting

The parameters from Table 3 were entered into the Pre-driven Recovery Evaluation Program.

Table 3 - Parameters for failure - major overburden weighting

Country	S. Africa
State	
No. of Rooms	1
Soft Floor	Yes
Depth (m)	70
CMRR	35
Seam Height (m)	3
Panel Width (m)	200
Room Length (m)	100
Shield Capacity (t)	327
RDI MPa.m	0.55
Standing Support MPa	0
Slow Mining	Yes
Outcome	Failure

Figure 15 shows that the program provided the correct result and gave a result of 'Strongly Not Recommended'. This was mainly due to the mine having a low CMRR and no standing support. It also had a soft floor and a slow mining advance rate.

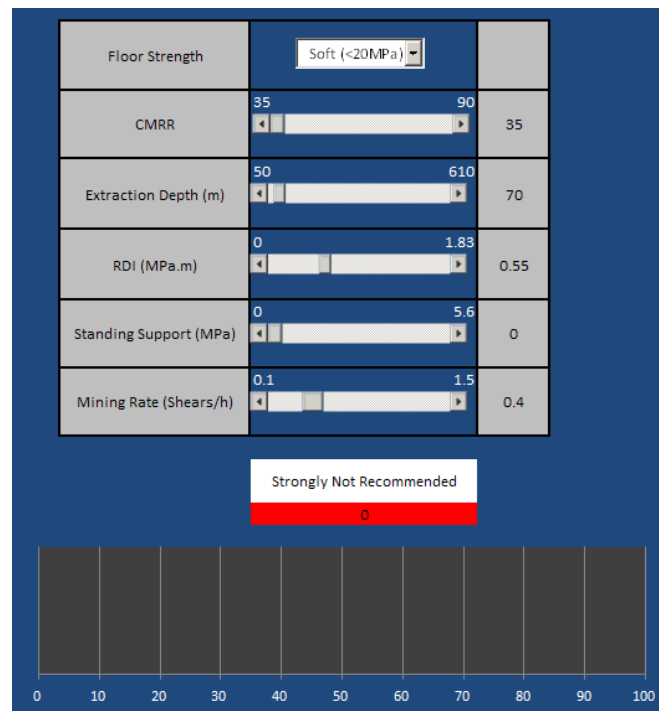


Figure 16 - Results for failure due to major overburden weighting

CONCLUSIONS

Pre-driven longwall recovery rooms can be used to safely recover longwall equipment from the current coal panel. This particular method can also reduce the time needed to extract the longwall equipment as support is applied to a pre-driven room prior to the longwall face reaching the take-off point. Therefore the longwall can maintain a constant advance rate compared to the conventional method where considerable production delays occur due to face preparation.

It has been recommended that the use of full length pre-driven recovery roads under laminate or mudstone/siltstone roof strata was found by research and experience to be a high risk operation from which the derived benefit was not feasible. It was also found that the combination of the internal bolting system and the standing concrete supports are critical for the successful and safe recovery of the longwall equipment.

Although there have been some catastrophic failures in the past from the use of pre-driven roadways, the majority of pre-driven roadway operations have proven to be successful. It was discovered from the data analysis that floor strength, depth of extraction, CMRR, RDI, standing support and advance rate all play a vital role in the success of pre-driven recovery. These parameters along with their individual amount of influence on the final outcome were incorporated into an Excel macro-driven program, the *Pre-driven Recovery Evaluation Program*. This program has been designed to be user friendly and clearly display the final recommendation both numerically and visually as to whether the use of a pre-driven recovery roadway will produce a successful outcome.

The *Pre-driven Recovery Evaluation Program* will provide excellent assistance for new longwall mining operations as well as current operations desiring to change to pre-driven recovery to assess the feasibility of this particular method for longwall equipment recovery. It is however recommended that the program only be used as a guide and if pre-driven recovery is being strongly considered, that a strata specialist be used to fully assess the situation.

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ALTS 2009 – A TEN YEAR JOURNEY

Mark Colwell¹ and Russell Frith²

ABSTRACT: This paper summarises the development and application of the ALTS (Analysis of Longwall Tailgate Serviceability) design methodology for longwall gateroad design associated with Australian collieries. The original ALTS design methodology was presented to the industry via workshops in early 1999 and since that time continued research, updating of the database and direct support from most Australian longwall operations has resulted in the ALTS 2009 software package, which also incorporates ADRS (Analysis and Design of Rib Support). In addition to the chain pillar design component, ALTS 2009 now provides design recommendations for primary and secondary roof and rib support for both the belt road and travel road/tailgate. ALTS and ADRS are empirical techniques which recognise that several geotechnical and design factors affect gateroad performance and in addition that operational and safety issues essentially dictate the level of performance required. These techniques are based on a sound mechanistic understanding of roadway behaviour, are transparent in their content and application and geotechnical engineers can be readily trained in their use.

As part of the review of ACARP's geomechanics-related research in 1991, 52 underground geomechanics-related projects were highly rated in terms of their research quality and industry application. ALTS was one of 11 projects that received the highest rating and yet it took several years for it to gain the widespread acceptance it now enjoys. It is suggested that the principal causes of this delay were; a misguided point of view when relating the science of rock mechanics to engineering and, some ill-informed commentary concerning empirical modelling in general and specifically with respect to ALTS. The myths and some of the mis-information surrounding ALTS are addressed.

INTRODUCTION

In many cases prior to 1998, chain pillars in Australia had been designed utilising a process similar to that used for pillars within bord-and-pillar operations, which applies a Factor of Safety in relation to pillar collapse. As discussed by Colwell et al (1999) this approach was inadequate and there was a clear need for a design method uniquely developed for Australian longwall chain pillars. In 1997 with ACARP (Australian Coal Association Research Program) and colliery support a research program (ACARP Project C6036, Chain Pillar Design – Calibration of ALPS) commenced to develop such a method.

The starting point or basis of that research program was ALPS, i.e. Analysis of Longwall Pillar Stability. The ALPS methodology (Mark, 1990 and Mark et al, 1994) was chosen because of its operational focus, as it uses tailgate performance as the determining chain pillar design criteria rather than simply *inner core* pillar stability which is the sole focus of factor of safety design methods. Furthermore ALPS recognises that several geotechnical and design factors, including (but not limited to) chain pillar stability, affect tailgate performance.

Based on this initial research the original ALTS design methodology was developed (Colwell, 1998 and Colwell, 1999). During the initial ALTS research, it was identified that a compromise between pillar size, primary roof support and secondary roof support is possible and necessary to efficiently achieve satisfactory tailgate conditions.

The original database (1997/8) was of sufficient size to confidently make recommendations for chain pillar size and to provide guidelines in relation to the installed level of primary roof support. However it was only possible to make a subjective assessment in relation to secondary roof support requirements. Funding from individual collieries and mining companies allowed for the expansion of the database in 2000, from which the ALTS II design methodology was developed (Colwell et al, 2003).

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As part of his review of ACARP's geomechanics-related research, Brown (2001) considered 52 underground geomechanics-related projects and individually rated the projects in terms of their research quality and industry application. ALTS was one of only 11 projects that received the highest rating and yet it took several years for it to gain the widespread acceptance it now enjoys. It is suggested that the principal causes of this delay were due to a misguided point of view when relating the science of rock mechanics to engineering and some ill-informed commentary concerning empirical modelling in general and specifically with respect to ALTS.

The development of ALTS II marked a significant leap forward for the Australian coal industry, in that the interaction between roof quality, primary and secondary roof support and chain pillar size had been quantified in terms of satisfactory tailgate performance. With ALTS II the roof support levels could (and should) be assessed in combination with rather than independently of the chain pillar dimensions.

In subsequent years the ALTS database was continually updated and significantly expanded such that it now includes detailed information in relation to both the tailgate (148 cases) and maingate belt road (58 cases). Further funding from individual collieries and mining companies resulted in the 3-year ALTS 2006 Project. A major component of the ALTS 2006 project was to conduct research so as to develop a roof support design capability for the maingate belt road which would then be included as a design module within the ALTS 2009 software package. This paper details those and associated analyses and their impact on ALTS.

LONGWALL LAYOUT AND TERMINOLOGY

To assist with subsequent discussion contained in this paper and terminology used, reference is made to Figure 1, which is a plan schematic of a typical Australian longwall mining layout utilising a two heading gateroad system. Figure 1 depicts a fully extracted longwall panel, one currently being extracted and a third where the gateroads (MG 3 – 'A' and 'B' Headings) are still to be completed to fully delineate the longwall panel and chain pillars. 'A' Heading is generally referred to as the travel road along which men, materials and machinery will travel, while 'B' Heading is called the belt road where the conveyor belt is installed to transport coal from the longwall extraction face.

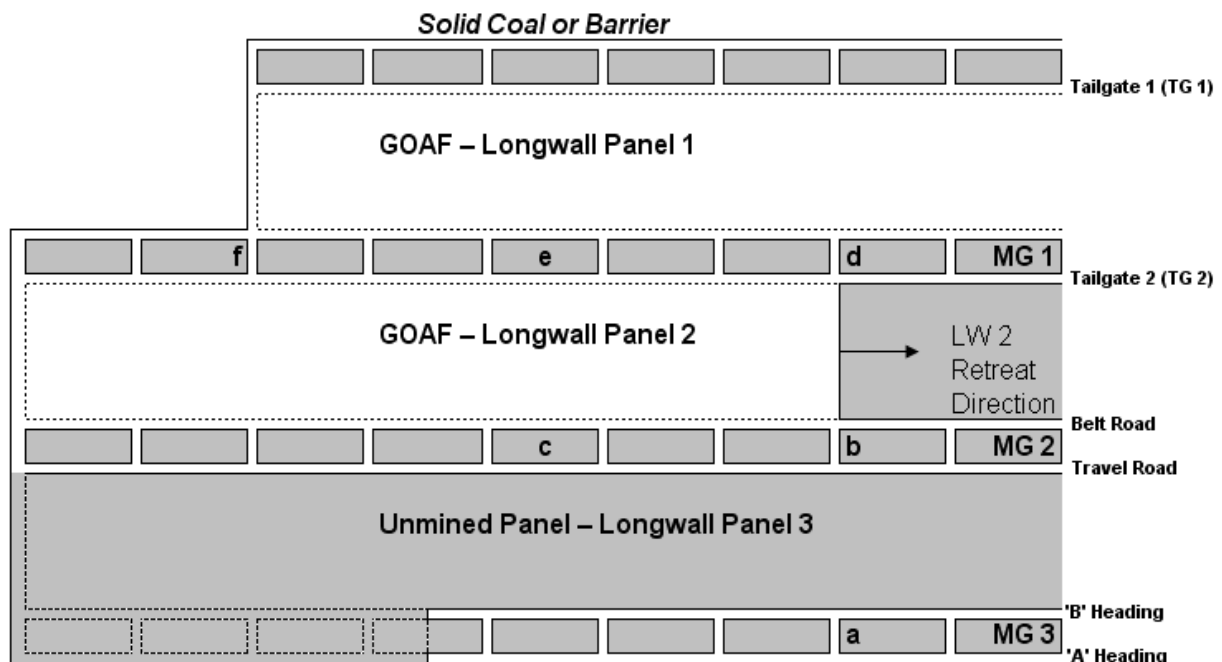


Figure 1 - Typical Australian longwall layout

In a series of longwall panels, 'A' Heading typically serves two roles, firstly as the travel road of the current longwall panel and secondly as the tailgate of the next. For example, the travel road of Longwall Panel 2 (LW 2, refer Figure 1) will become the tailgate of LW 3. Therefore this travel road/tailgate is subject to a series of changing geotechnical environments, moving from development

(Position a) to the passage of the 1st adjacent longwall face (Positions b and c respectively) and finally being subject to the approach of the second adjacent longwall face up to the tailgate intersection (Position d) with the travelling longwall face.

With reference to Figure 1 it can be seen that the chain pillars are also subject to a series of changing loading environments with the following terminology being used to describe each stage of the chain pillar loading cycle:

- Position a – Development loading
- Position b – Maingate belt road or front abutment loading
- Position c – Maingate (MG) loading
- Position d – Tailgate (TG) loading
- Position e – Double goaf (DG) loading

MAINGATE BELT ROAD ROOF SUPPORT ANALYSES

The maingate belt road database comprises 58 cases representing 33 longwall operations where the Coal Mine Roof Rating (CMRR) ranges from 25 to approximately 80 and the cover depth ranges from 100m to 510m. The analyses clearly indicated that the principal geotechnical drivers which, in combination, essentially dictate the level of roof support required to maintain a satisfactory level of roof performance during longwall extraction include:

1. The structural integrity of the immediate roof (as measured by the CMRR) and,
2. The magnitude of (or at least a reliable estimate or indicator of) the horizontal stress acting across the roadway roof adjacent to the 'travelling' intersection with the longwall face (refer Position b – Figure 1).

Calculating the Resultant Horizontal Stress (σ_{R-Dev} & σ_{R-MGB})

The following information is required to calculate/estimate the horizontal stress acting perpendicular to the direction of driveage (i.e. σ_{R-Dev} , MPa) and subsequently that stress acting across the roof of the belt road adjacent to the intersection with the longwall face during retreat extraction (i.e. σ_{R-MGB} , MPa):

- Longwall retreat direction (LW(θ), degrees from true north)
- Major horizontal stress direction (σ_H orientation – degrees from true north)
- Magnitude of the major horizontal stress (σ_H , MPa)
- Magnitude of the minor horizontal stress (σ_h , MPa)

The angle between the longwall retreat direction and the major horizontal stress direction is designated as " β - Beta" (refer Figure 2). Note: the minor horizontal stress direction is taken to be at 90° to the major horizontal stress direction.

The resultant horizontal stress acting perpendicular to the direction of driveage (i.e. σ_{R-Dev}) is calculated using equation 1 (refer Page 92, Hoek & Brown, 1980) which is derived from Mohr's Circles:

$$\sigma_{R-Dev} = [0.5 * (\sigma_H + \sigma_h) - 0.5 * (\sigma_H - \sigma_h) * \cos(2\beta)] \quad \text{MPa} \quad (1)$$

The change and increase in horizontal stress in the roof that occurs about the belt road intersection with the longwall face during retreat extraction (i.e. refer Position B – Figure 1) is often referred to as *Maingate Stress Notching*. The magnitude of the resultant stress (in MPa) is denoted as σ_{R-MGB} , and was estimated based on the research findings of Gale and Matthews (1992), Mark et al (1998) and Su and Hasenfus (1995) to estimate σ_{R-MGB} .

Gale and Matthews (1992) linked a Stress Concentration Factor (SCF) to the angle between the longwall retreat direction and the stress direction (i.e. the angle " β " – refer Figure 2). This relationship is detailed in Figure 3 such that the SCF is used as a multiple of the magnitude of the *in situ* horizontal stress to estimate the resultant stress acting across the roof about the belt road intersection with the longwall face. When the angle (β) between the direction of longwall retreat and the major horizontal

stress (σ_H) is between 0° and 90° then the belt road is subject to a concentration of the major horizontal stress such that $\sigma_{R-MGB} = SCF * \sigma_H$.

The maingate is technically within a zone of major horizontal stress relief when $90^\circ < \beta < 180^\circ$, however in this situation the SCF would need to be applied to the minor horizontal stress (σ_h) to assist in calculating a resultant horizontal stress magnitude (σ_{R-MGB}) for design purposes. Therefore it is necessary to have reliable/realistic estimates for the magnitude and direction of both the major and minor horizontal stresses. Where possible the estimates used for σ_H and σ_h should be that which best represent the immediate roof strata. The database was formulated and analysed in this manner

A similar relationship to that displayed in Figure 3 was found by Su and Hasenfus (1995) using three-dimensional finite element modelling. The research findings of Su and Hasenfus (1995) were also utilised by Mark et al (1998) and incorporated by NIOSH in their software program, Analysis of Horizontal Stress in Mining (AHSM). In this instance the angle (β) between the direction of longwall retreat and σ_H (from 0° to 180°) is plotted against a percentage (%) of the maximum possible stress concentration.

It was found that the maximum (or 100% of the maximum) stress concentration occurred when $\beta \approx 70^\circ$ (similar to that by found Gale and Matthews, 1992 – refer Figure 3) and when $\beta \approx 160^\circ$ the stress concentration is a minimum, which is expressed or plotted as 0% (Mark et al, 1998). In terms of the horizontal stress magnitude acting across the roof this is not possible i.e. 0 MPa. While there may be 100% relief of the major horizontal stress there will still be a concentration of the minor horizontal stress as previously explained.

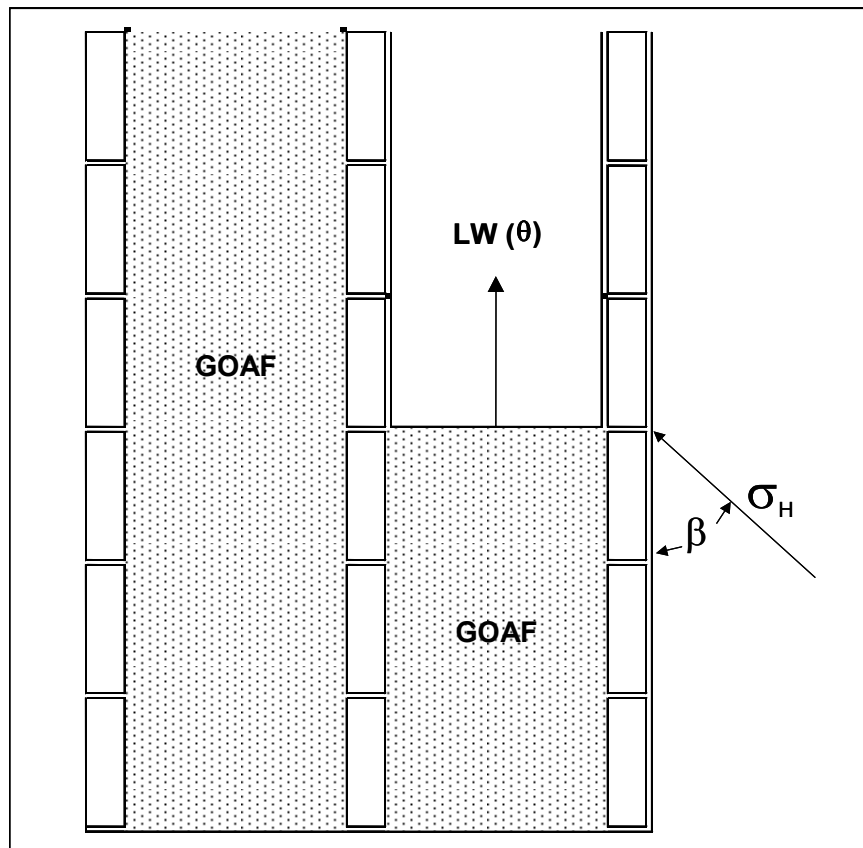


Figure 2 - The angle β used to determine the values of σ_{R-Dev} and σ_{R-MGB}

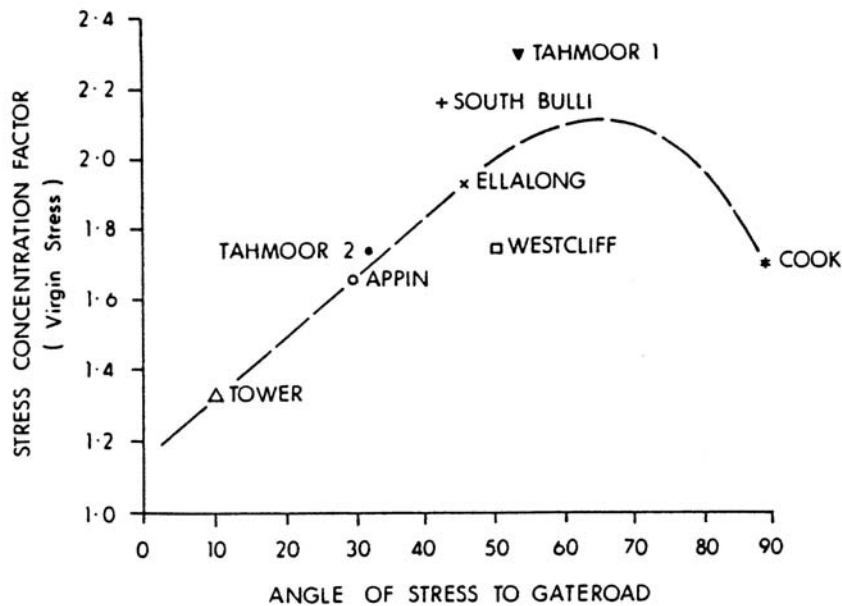


Figure 3 - Relationship between stress concentration factor and angle of gateroad to stress direction (after Gale and Matthews 1992)

Utilising the research findings of Gale and Matthews (1992), Mark et al (1998) and Su and Hasenfus (1995), Figure 4 was developed in terms of the SCF associated with the major horizontal stress (σ_H), which is now denoted as SCF_H , to estimate the maximum stress in terms of σ_H . When $90^\circ < \beta < 180^\circ$ then a concentration of the minor horizontal stress (σ_h) occurs. The SCF associated with the minor horizontal stress is denoted as SCF_h . Figure 4 can be used to interpret SCF_h ; for example if $\beta = 150^\circ$ then the angle between the direction of longwall retreat and the minor horizontal stress would be 30° and therefore $SCF_h \approx 1.7$, while $SCF_H \approx 1.05$.

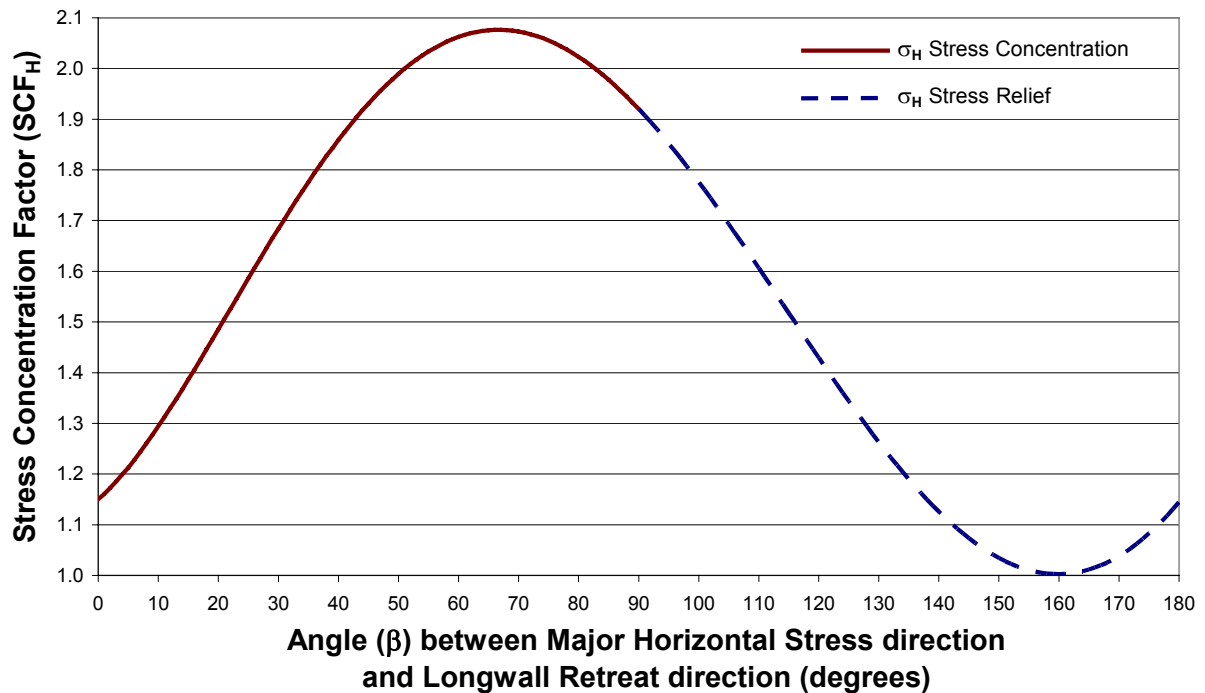
Based on Figure 4 and the above discussion the following logic is utilised in calculating σ_{R-MGB} .

1. When $0^\circ < \beta < 90^\circ$ then $\sigma_{R-MGB} = SCF_H * \sigma_H$ (2)
2. When $90^\circ < \beta < 180^\circ$ then $\sigma_{R-MGB} = \text{Max} [(SCF_H * \sigma_H) \& (SCF_h * \sigma_h)]$ (3)

Roof Support Analyses

The initial series of analyses associated with the maingate belt road database plotted the total roof support level measured by the Ground Support (GRSUP) rating - see Appendix A) against the CMRR for both headings and intersections. It should be noted that "Headings" initially refers to the sections of the belt road, either travel road or tailgate (refer Figure 1) between cut-through intersections, while "Intersections" refers to the sections of the gateroad that intersect with the cut-throughs. It was found during the course of the research that most collieries (as a part of their Support Rules) increase roof support levels within the intersections and for certain distances either side of the cut-through edge along the heading (i.e. inbye and outbye of the intersections). This practice is consistent with both the geotechnical environment and operational factors.

For example with respect to the belt road Thomas & Wagner (2006) state that "during longwall retreat the magnitude of horizontal stress notching in a maingate belt road will increase on the inbye side of a cut-through and reduce on the outbye side of a cut-through. This phenomenon is related to the tendency for the horizontal stress to concentrate between the longwall goaf and the cut-through (termed "stress pinching") and the subsequent ability of the cut-through to relieve the horizontal stress about the gate road when the face retreats outbye of the cut-through".

Figure 4 - SCF_H versus β

Furthermore the size or “effective area” of an intersection can be due to operational issues (i.e. how the intersections are formed), what level of standing support (if any) is installed at the mouth of the cut-through during longwall retreat and whether pillar corners are lost post development or as a result of increased vertical load associated with longwall retreat. These issues would also impact on support densities within and about intersections (for both the maingate and tailgate). Due to space constraints associated with a conference paper it is only the “Headings” analyses that are presented.

Based on linear regression analyses; a strong exponential relationship was found between the installed level of roof support (GRSUP refer Appendix A) and the structural integrity of the bolted mine roof interval (CMRR). Figure 5 displays the database for headings as well as the exponential trendline relationship and upper and lower boundaries that encompass the vast bulk (approximately 95%) of the data.

However further analyses, utilising the statistical technique of multiple regression revealed that in addition to the structural integrity of the roof (as measured by the CMRR), σ_R -MGB also had a major impact on the resultant GRSUP utilised by the collieries. Based on the multiple regression analyses the following relationship was found with respect to headings:

$$\text{LN (GRSUP}_{\text{Headings}}) = 5.3604 - 0.0415 \text{ CMRR} + 0.0201 \sigma_R\text{-MGB} \quad (4)$$

or

$$\text{GRSUP}_{\text{Headings}} = 212.81 \times e^{-0.0415 \text{ CMRR}} \times e^{0.0201 \sigma_R\text{-MGB}} \quad (5)$$

With the inclusion of σ_R -MGB the correlation (in terms of GRSUP v's CMRR & σ_R -MGB) increases significantly and is an exceptionally high 0.89. Based on the relationships associated with equations 4 & 5; a roof support monogram can be produced to “visually” demonstrate the combined impact of the CMRR and σ_R -MGB on GRSUP. Figure 6 clearly illustrates that the GRSUP v's CMRR relationships for varying stress levels acting across the roof (i.e. σ_R -MGB) fit seamlessly within the upper and lower boundaries of the database. The maximum σ_R -MGB associated with the maingate belt road database is approximately 45 MPa.

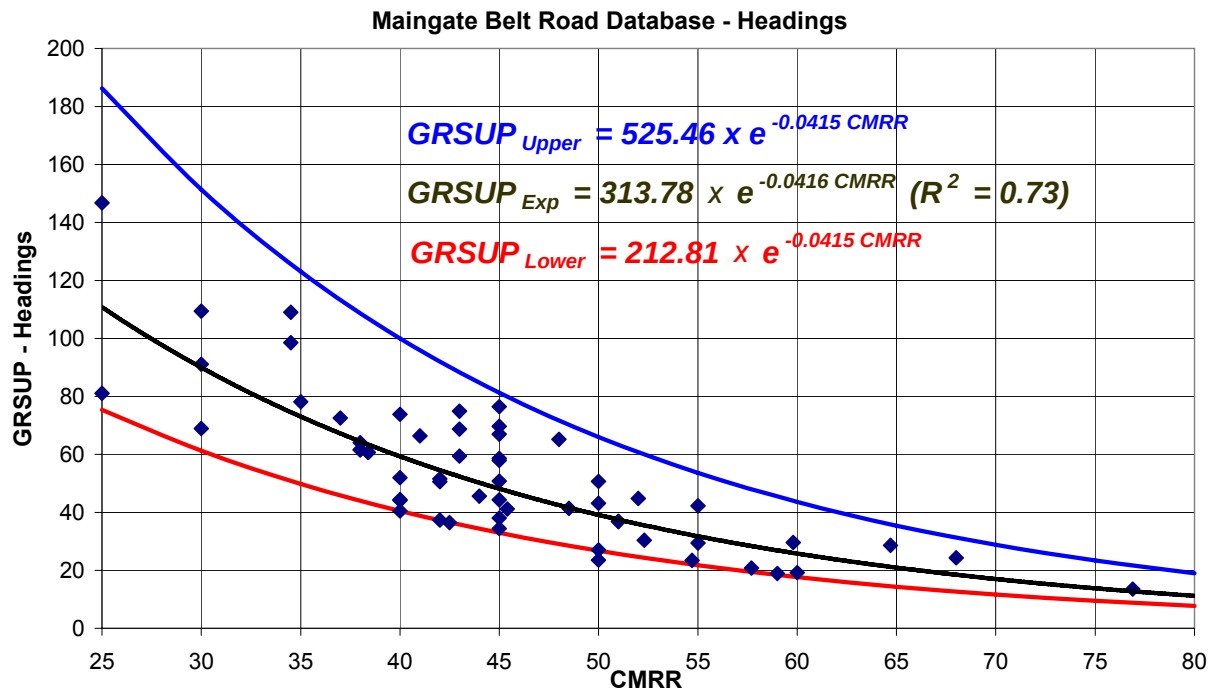
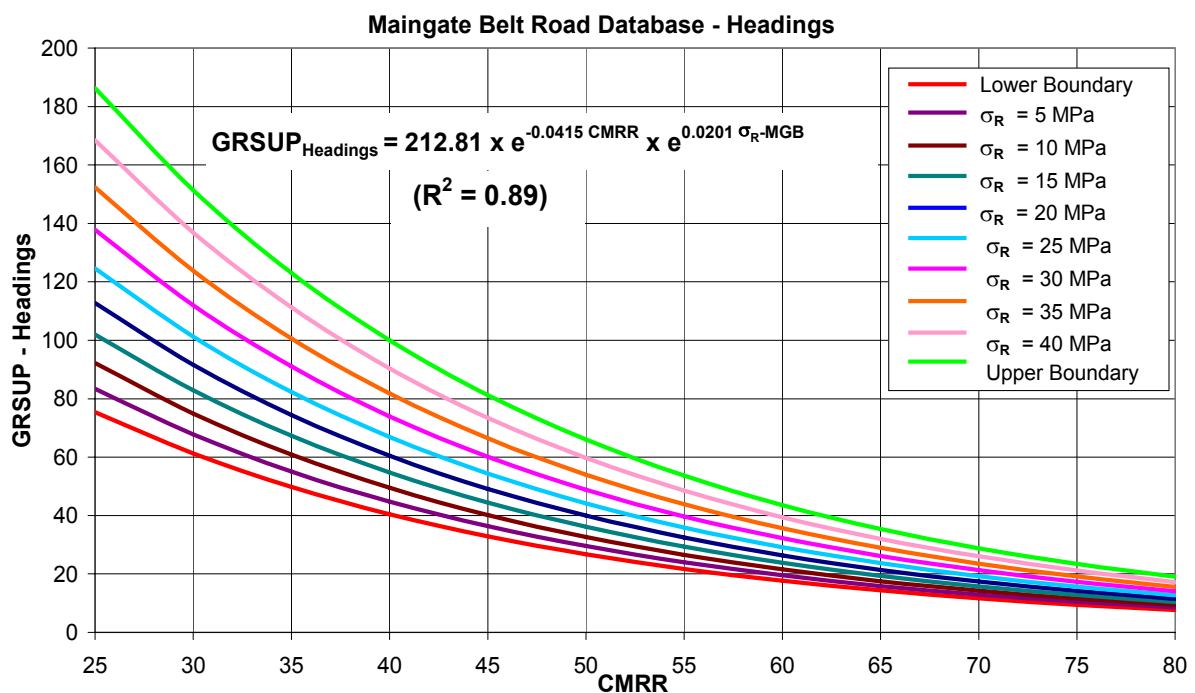


Figure 5 - GRSUP v's CMRR maingate belt road headings

Figure 6 - GRSUP v's (CMRR and σ_R -MGB) – maingate belt road headings

Supplementary roof support analyses

When utilising empirical models for geotechnical design, the size (i.e. number of cases) and extent (i.e. number of different coalfields/collieries) of the database is an important factor to consider with respect to the confident application of the statistical relationships.

For example Salamon et al (1996) describe the Australian pillar database of 19 collapsed and 16 unfailed cases as a “*relatively small database*”, however the resultant UNSW pillar strength equation (Galvin et al, 1999) derived from the Australian database can be confidently used for design as

Salamon et al (1996) combined the Australian database with the much larger South African database (142 cases) and clearly demonstrated, *"that the strength estimates derived from the combined database approximate well both the Australian and South African strengths computed from the individual estimators"*. In practical terms what this means is that the large South African database essentially *underpins* our confident use of the *UNSW rectangular pillar strength formula* for the design of pillars which fall within the *limits of the geotechnical parameters* associated with the combined database.

With respect to the above analyses, pillar failure was defined as *collapse* of the pillar not simply *pillar yield*. An outcome of this type (i.e. pillar collapse) when a pillar is designed to be stable would not be acceptable within any country's underground coal industry and therefore is a *black & white* outcome.

Unfortunately with respect to roof support design it not generally practical to combine another coal industry's roof support database with an Australian roof support database as the tolerable level of risk, in terms of roof instability, will vary from country to country.

For example the roof support design methodology developed by NIOSH for the United States underground coal industry, ARBS (Analysis of Roof Bolt Systems – Mark et al, 2001) defines *Failure* as, *"more than 1.5 reportable roof falls per 3048 m (10,000 ft) of drivage"*. These are roof falls as a result of roadway development and therefore do not include gateroad roof falls associated with longwall retreat. This definition of roof failure (or level of roof falls) would be totally unacceptable in the Australian underground coal industry and this discussion highlights that a country's tolerable level of risk is a critical factor in the level (and type of support) utilised and in developing a roof support design methodology.

The maingate belt road database of 58 cases (reported here) were all considered successful by the respective colliery in the sense that the colliery reported that there had been no production delays or safety concerns and certainly no roof falls or remedial roof support measures required. In terms of size, the maingate belt road database would be considered medium size with respect to worldwide databases utilised for geotechnical design in relation to underground coal mining. Therefore the question is, *"can the maingate belt road database be supplemented or tested to increase our confidence in the application of equation 5 for the geotechnical design of roof support associated with the maingate belt road?"*

The primary roof support database developed via the various ALTS research projects comprises 109 cases (representing 38 collieries; being 36 longwall and 2 bord & pillar). The analyses associated with the primary roof support database found that the principal geotechnical drivers which, in combination, essentially dictate the level of roof support required to maintain a stable roof during and as a result of roadway development are the structural integrity of the immediate roof (as measured by the CMRR) and the horizontal stress acting perpendicular to the drivage direction (i.e. σ_R -Dev)

The maingate belt road and primary roof support databases cannot be directly combined due to an operational factor. With respect to the primary roof support database it is known that in addition to the geotechnical/risk related issues, operational factors directly influence the level of primary support utilised within the gateroads of Australian collieries. For example many collieries elect to install a level of primary roof support off the continuous miner greater than what would be required to simply maintain satisfactory roadway conditions on development as it is operationally more convenient or effective to do so off the miner rather than installing secondary support at a later stage to maintain satisfactory roadway conditions during longwall retreat.

The maingate belt road roof support database is not subject to a similar operational issue as only that roof support deemed necessary to satisfy the geotechnical (e.g. subject to σ_R -MGB) and risk related issues is installed, i.e. a colliery would not plan to install "tertiary" support in the belt road during longwall retreat as all planned roof support is installed prior to longwall retreat.

To overcome this operational issue and in an attempt to *test* the maingate belt road database it was decided to combine the maingate belt road roof support database with those cases from the primary roof support database where the colliery proactively installed secondary tendon support within the travel road prior to longwall retreat. Of the 109 primary roof support database cases, 32 cases satisfy that criteria.

If a colliery is proactively installing secondary tendon support within their travel road prior to longwall retreat then it is reasonable to conclude that the level of primary roof support (measured by the PRSUP Rating - see Appendix A) would be sufficient to maintain satisfactory roadway conditions subsequent to development and prior to longwall retreat, however it would be deemed as insufficient by the colliery to deal with the horizontal stress increases associated with longwall retreat. Furthermore, in terms of remedial roof support, it is also reasonable to conclude that the level of roadway roof stability required by the colliery as a result of development is approximately the same as that expected in the belt road during longwall retreat.

A colliery would not typically plan to install secondary roof support to simply maintain satisfactory roadway conditions solely as a result of development in basically the same way as a colliery would not plan to install "tertiary" support in the belt road during longwall retreat. Therefore via this combined database the operational issue related to installing a level of roof support greater than that required to effectively deal with the resultant horizontal stress acting across the roof (i.e. σ_R -Dev or σ_R -MGB as the case may be) is substantially eliminated from the analyses. This combined and relatively large database of 90 cases essentially represents a level of *reinforced* roof stability in terms of:

1. a tolerable level of risk specific to Australian collieries and;
2. the two principal geotechnical drivers being the structural integrity of the immediate roof (as measured by the CMRR) and the resultant stress (σ_R).

Furthermore, if the above logic holds true then the resultant level of correlation (R^2) and the regression equation should be similar to that found in relation to the maingate belt road database on its own. Figure 7 presents the relationships for GRSUP along headings plotted against the CMRR for varying stress levels acting across the roof (i.e. σ_R). The multiple regression relationships relating GRSUP to the CMRR and σ_R are also displayed. It can be seen that the overall shape of the relationships plotted on Figure 7 are comparable with those associated with Figure 6 as well as the correlation associated with the respective regression relationships.

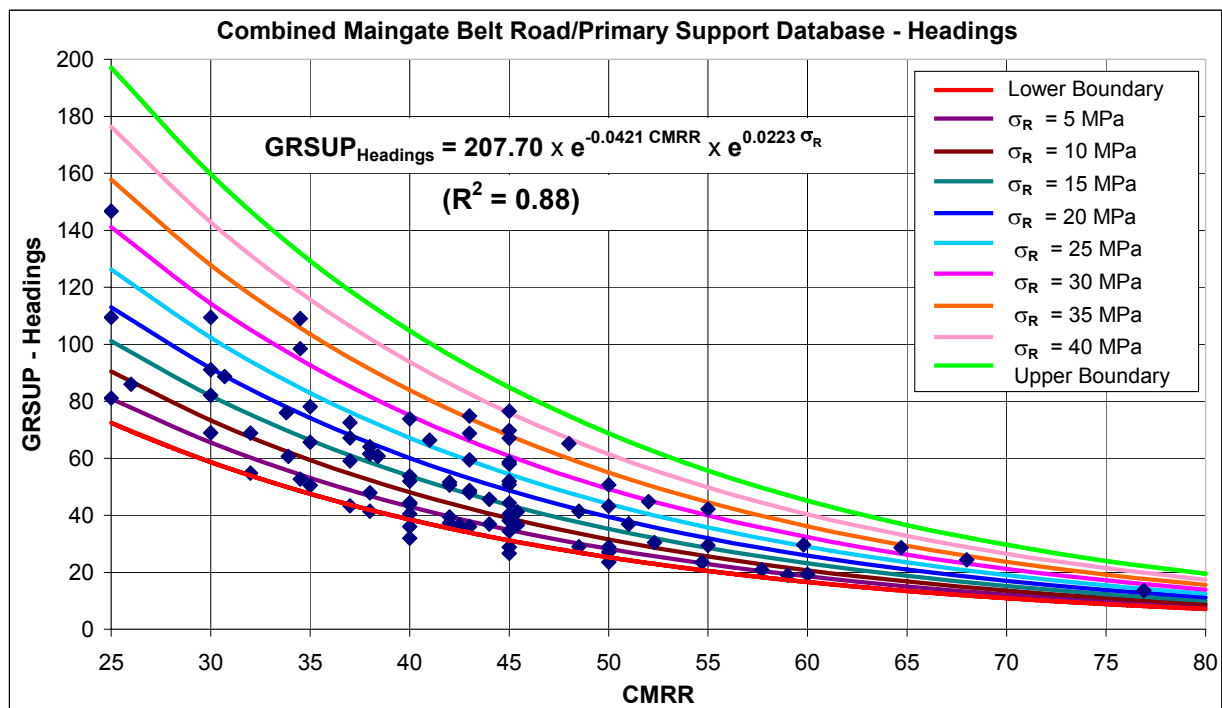


Figure 7 - GRSUP v's (CMRR & σ_R) – combined database – headings

IMPACT ON ALTS

In terms of tailgate roof support design, ALTS had been specific to the typical case where a tailgate had acted as the travel road of the previous longwall panel and therefore the roadway is subject to double (or 2nd) pass longwall extraction (e.g. refer TG 2 – Figure 1). While some level of horizontal stress increase will occur in the travel road due to the approaching longwall face it will generally be significantly less than that experienced in the belt road.

Furthermore under this travel road/tailgate scenario, once an adjacent goaf is established a substantial amount of the *in situ* horizontal stress increase (if any) is relieved and any further increase in horizontal stress acting across the roof (for example during tailgate loading refer Position d – Figure 1) is not related to the *in situ* horizontal stress and can only be as a result of Poisson's Effect associated with an increase in the vertical load acting above the riblines adjacent to the roadway.

Tailgates subject to *single or super stress notch* conditions

The maingate belt road analyses have a significant impact on ALTS in two ways. Firstly, in terms of tailgate roof support design ALTS can now clearly deal with those tailgates subject to a *single or super stress notch* conditions. Figure 1 reveals that Tailgate 1 (i.e. TG 1) would only be subject to single pass longwall extraction such that the roof about the tailgate intersection with the face is subject to a *Single Stress Notch*, similar to that experienced in a belt road. While similar, it is manifestly different in the sense that for a series of panels a notching or increase of the major horizontal stress for TG 1 (with respect to Figure 1) would mean that the maingate is technically within a zone of major horizontal stress relief, such that a notching of the minor horizontal stress would occur (or vice-versa).

Position f (refer Figure 1) relates to a specific (but not uncommon) situation which can result in a large increase in horizontal stress acting across a tailgate roof and is commonly referred to as a *super stress notch*. To occur, the longwall commences inbye of the start-line of the previous LW panel, in this case LW 2 in relation to LW 1. In this instance a larger (than typically encountered by the colliery) horizontal stress increase occurs as the faceline of LW 2 approaches and passes the start-line (or installation face) of LW 1.

While there is no database (per se) that specifically relates to either of the *Single or Super Stress Notch* tailgate scenarios, nonetheless the findings and recommendations associated with the primary roof support, maingate belt road and ALTS tailgate databases allow for a design process to be developed and high level of confidence in the roof support recommendations provided.

Under these two tailgate scenarios, the tailgate roof is subject to *in situ* horizontal stress increases as a result of longwall retreat in a similar manner as the belt road roof and will react accordingly dependent on the structural integrity of the roof (as measured by the CMRR), the level of horizontal stress acting across the roof and installed level of roof support. In this instance the stress acting across the tailgate roof as a result of longwall retreat is referred to as σ_{R-TG} (MPa). However being a tailgate (as opposed to a belt road) the design process needs to consider the possible use of or option of including secondary standing support as a part of the overall roof support strategy. The ALTS research provides the ability whereby a *trade-off* between tendon and standing support (within limits) can be assessed in terms of a serviceable tailgate.

Tailgates subject to double (or 2nd) pass longwall extraction

Previous ALTS research (Colwell, 1998 and Colwell et al, 2003) clearly revealed that chain pillars should not be designed without a detailed consideration of the level and type of ground support installed along the tailgate as well as a colliery's operational requirements. Furthermore said research established that for the same CMRR there is a *trade-off* between the total level of tailgate roof support (bolts/tendons plus standing support) and chain pillar width while maintaining the same level of tailgate serviceability.

In this instance ALTS focuses on tailgate performance (at the T-junction, refer Position d - Figure 1) as the design condition. The pillar stability factor in relation to the Tailgate (TG) loading condition is designated as the Tailgate Stability Factor (TGSF). The level of standing support is measured by the Standing Support (SSUP) Rating and therefore the total level of tailgate roof support equates to

GRSUP plus SSUP. The calculation of the TG SF and SSUP ratings remains unchanged from that previously published and the interested reader is referred to Colwell (1998) and Colwell et al (2003).

However it is recognised that this *trade-off* is within various limits. For example a base level of primary roof support is required, independent of the chain pillar size, to maintain satisfactory roadway conditions during and subsequent to development (while prior to longwall extraction). This base level of primary roof support (designated as PRSUP_{Dev}) cannot be a part of the trade-off between GRSUP + SSUP and TG SF and should to be determined (along with SSUP & TG SF) prior to calculating the recommended, upper and lower GRSUP values.

The additional roof support required to satisfy Travel Road/Tailgate serviceability is referred to as ROOFSUP_{TG}, where ROOFSUP_{TG} equals GRSUP plus SSUP minus PRSUP_{Dev}. ROOFSUP_{TG} is the measure of the level of roof support which, for a specific CMRR, can be involved in a *trade-off* with the TG SF while maintaining the same satisfactory level of tailgate serviceability.

Utilising multiple regression, it was found that when the base level of primary support was subtracted from the total installed roof support only two parameters were significant predictors of the resultant level of roof support (i.e. ROOFSUP_{TG}) being the CMRR and TG SF. It is noted that σ_R -Dev ceased to be a significant predictor of (or have an impact on) ROOFSUP_{TG} even though the logistic regression analyses found σ_R -Dev to be a significant predictor of eventual tailgate serviceability.

As discussed by Colwell (2006) it is critical when utilising empirical modelling for geotechnical design that a clear understanding of the geotechnical environment and rock mass failure/behavioural mechanisms is required. σ_R -Dev is clearly critical to primary support levels and therefore it is more than reasonable that it has a significant impact on the eventual outcome i.e. tailgate serviceability. However in terms of σ_R -Dev's impact on ROOFSUP_{TG} these analyses are totally consistent with the nature of the geotechnical environment associated with a travel road/tailgate subject to 2nd (or double) pass longwall extraction.

As previously discussed once an adjacent goaf is established any increase in horizontal stress acting across the roof is not related to the *in situ* horizontal stress and can only be as a result of Poisson's Effect associated with an increase in the vertical load acting above the riblines adjacent to the roadway. This will vary dependent on several factors including (but not limited to) the distribution of the abutment load, the nature of the coal, the rib height (i.e. the development height) & pillar width and the installed level/type of rib support. The TG SF successfully "*captures*" a large proportion of the combined effect.

COMMENTS ON EMPIRICAL MODELLING AND ALTS

The authors contend all geotechnical models utilised for design associated with underground coal mining are in fact empirical in nature as calibration may be required and engineering judgement will always need to be used when applying any design outcomes. It does not matter whether the *engine room* of the model is analytical or numerical as either will require significant calibration prior to the model being effectively or confidently utilised for design purposes, whereas the calibration process is intrinsically a part of an empirical model whose *engine room* is an industry database.

The authors assess (based on industry research/experience) that for small vertical roof displacements (up to around 50mm and possibly to 100mm), slender beam behaviour or buckling is the dominant behavioural mechanism occurring within the immediate coal mine roof measures which, if not controlled, leads to large scale roof displacement and eventually a major collapse. One of the primary reasons that numerical models (as they are being used with respect to the underground coal industry) require a high level of calibration via parameter manipulation is that the modelling process does not include the mechanistic principles of this dominant behavioural/failure mechanism.

With the advent of more powerful computers, some researchers have tended to move away from empirical and physical models to numerical modelling. While the modelling of rock behaviour using numerical methods has improved and mathematical routines have been developed in an attempt to account for both elastic and plastic behaviour (e.g. FLAC – Gadde & Peng, 2005 and Gale & Tarrant, 1997; 3STRESS – Medhurst, 1996 and MAP3D – Palmer & Morrison, 2005), the various models do not incorporate mathematical routines associated with buckling.

In addition these researchers have been considering geometries (or setting up their models) which contain structural elements that, by their very nature, cannot buckle and must fail in either direct compression (as one would observe in a laboratory based strength test) or shear. This is in complete contrast to the slender beams associated with coal mine strata, which either form the immediate roof or quickly develop within the immediate roof due to roadway formation or as a result of a horizontal stress increase. Therefore it is not surprising that the issue of buckling as a failure mechanism about mine openings/roadways has been largely ignored by researchers that rely heavily on numerical modelling in an attempt to replicate and understand roadway behaviour.

It is also realistic to suggest that there is a *point of view* held by a significant segment of the rock mechanics fraternity that numerical modelling provides a researcher/consultant with a tool to undertake *real engineering* whereas empirical-statistical techniques offer only "*simplistic formulae*" (Tarrant, 2005). It would be naïve for any researcher whose objective is to provide an underground coal mining industry with a widely accepted empirical geomechanics model, to be unaware of this *point of view*.

With the increased power of computers and possibly due to the time and considerable effort involved in collecting, verifying and analysing the large volume of information involved in formulating an industry-wide database, a number of researchers utilise numerical modelling, as Tarrant (2005) suggests, to develop a "*better understanding*" of roadway behaviour. Tarrant (2005) points out that, "*Use of such tools is limited by the simplifications required however when used in conjunction with field measurement and observation, the model findings can be tested and a level of confidence in the results defined.*"

The use of numerical modelling in the manner described by Tarrant (2005) is reasonable but unfortunately generally only provides a calibrated (via measurement) model to then be used for site specific prediction or design. Calibrating a numerical model to a limited number of sites does not provide an underground coal industry with a widely applicable and therefore accepted design tool for roadway ground support design. This is particularly the case when the numerical model being calibrated to said roadway behaviour does not incorporate mathematical code associated with buckling. Invariably one finds that in these instances the researcher does not produce a model or design technique that can be readily utilised by others in the industry, but typically it remains within the domain of the researcher or consultant for its application.

As part of his review of ACARP's geomechanics-related research, Brown (2001) considered 52 underground geomechanics-related projects and individually rated the projects in terms of their research quality and industry application. ALTS was one of only 11 projects that received the highest rating and yet it took several years for it to gain the widespread acceptance it now enjoys. It is suggested that the principal causes of this delay are a) the misguided *point of view* previously suggested and b) some ill-informed commentary concerning empirical modelling in general and specifically that with respect to ALTS.

For example, Tarrant (2004) suggests that ALPS (Analysis of Longwall Pillar Stability, refer Mark et al 1994) and ALTS provide a "*line in the sand*" in terms of chain pillar width with respect to tailgate serviceability. Neither method ever suggested there was a *line in the sand* in terms of chain pillar design and related tailgate serviceability. In fact Colwell et al (2003) detail a wide range in chain pillar width that can be employed while maintaining serviceable tailgate conditions and that the pillar width selected is contingent on the installation of recommended (i.e. engineered) levels of roof support (primary & secondary, tendon & standing). In spite of this information, Tarrant (2004) provides a diagram relating tailgate serviceability to pillar width which makes the erroneous suggestion that at a certain pillar width (derived by ALTS or ALPS) *no engineering is required* in terms of roof support.

Gale and Hebblewhite (2005) go further and state that ALPS and ALTS, "*have been developed largely on simple statistical correlations of tailgate conditions and support requirements, relative to pillar dimensions*". However the formulation of a geotechnical database, the minesite investigations, the identification & understanding of the failure mechanisms and the statistical (Data Mining) techniques employed in the development of ALTS, ALPS and ADRS is anything but a *simple* process.

Quality empirical modelling is in fact a scientific process of significant challenge and complexity. With respect to the underground coal geotechnical environment; empirical modelling allows for the development of practical and fully engineered design methodologies and techniques/tools that can provide the minesite strata control engineer with timely solutions to complex geotechnical design

issues. These techniques are also consistent with the thoughts of Professor Hustrulid (2006) where he indicates that marked progress in the field of mining rock mechanics requires, *"the careful collection, analysis and presentation of field/mine experience."*

The idea portrayed by some that ALTS, ADRS, empirical modelling per se and the resultant statistical relationships are simplistic and are limited in their application is at best misguided and at worst, misleading. Both methods (and the associated relationships) are founded on almost the entire range of geotechnical environments as well as roof and rib control practices in the Australian underground coal industry. The resulting statistical "cause and effect" relationships, which are then utilised for design, are exceptionally strong and are fully consistent with the changing nature of the geotechnical environment and failure mechanisms.

The geotechnical environment and the way in which roof and rib support interacts with the rock mass are complex issues. However it is generally recognised that without prudent simplification, the complexity of the problem will overwhelm all current geotechnical methods of modelling. While the problem should not be oversimplified (i.e. the dominant failure mechanisms or critical data input parameters should not be ignored), without question judicious simplification is at the heart of all engineering design. Therefore the findings of ALTS and ADRS should give the industry heart that the problems faced can be reasonably understood by all and ultimately designed for at a mine site level.

The principal geotechnical drivers identified for both general roof and rib stability make good engineering sense and are fully consistent with what an engineer would expect to find according to the proven principles of slender beam behaviour and buckling. For anyone to now dismiss, *pigeon hole* or use unjustified *throw away lines* in relation to ALTS or the CMRR (i.e. Tarrant, 2004 & 2005, Gale and Hebblewhite, 2005 and Calleja, 2008) would simply display a significant level of engineering ignorance as well as conveniently ignoring the scientific evidence supporting the practical benefits of this design technique and rock mass classification index.

CONCLUSIONS

The analyses reported in this paper are totally consistent with and accurately reflect the changing geotechnical environment encountered by both the belt and travel roads from development through to respectively *Maingate Stress Notching* and *Tailgate loading*. The strength of the various relationships developed for roof support design are exceptionally high and are also fully consistent with what an engineer would expect to find according to the behavioural/failure mechanisms occurring within the roof.

In relation to empirical modelling Salamon (1989) states, *"The main advantage of this approach is its firm links to actual experience. Thus, if it is judiciously applied, it can hardly result in a totally wrong answer. Also, in our legalistic world, it has the added advantage of defensibility in a court of law. After all, it is based on actual happenings and is not just a figment of imagination"*. ALTS and ADRS go even further, as the statistical relationships and the way they are utilised as a part of the design methodologies intrinsically represent a tolerable level of risk specific to Australian collieries. Therefore while the recommendations emanating from ALTS need to be applied judiciously (as for all design techniques) they can be confidently applied to all Australian longwall operations.

The ALTS 2009 Software Package assists and offers the user the ability to undertake roof and rib support design (primary & secondary, tendon & standing and also in terms of ADRS the appropriate use of rib mesh) for both the maingate belt road and travel road/tailgate and of course chain pillar design (where ALTS all started). When the original ALTS Project was completed in October 1998 the lead author had the "picture" in mind of developing an integrated approach to longwall gateroad design that could be utilised by the minesite geotechnical engineer at all Australian longwall operations. After 10 years and with the Australian coal industry's support the picture has become a reality.

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APPENDIX A. PRSUP & GRSUP RATING CALCULATIONS

Primary Roof Support (PRSUP) Rating

The Primary Roof Support (PRSUP) Rating is a measure of the bolting capacity (kN) per square metre of roof and includes all bolt/tendon support that is installed off the continuous miner or mobile bolter as part of development. The equation to calculate PRSUP is:

$$\text{PRSUP} = \frac{L_b \cdot N_b \cdot C_b}{14.5 \cdot S_b \cdot w_e} + \frac{L_b \cdot N_t \cdot C_t}{14.5 \cdot S_t \cdot w_e}$$

where L_b = Length of bolted horizon defined by the primary bolt type (m)
 N_b = Average number of bolts per row
 N_t = Average number of longer tendons per row
 C_b = Ultimate tensile strength of the primary bolt (kN)
 C_t = Ultimate tensile strength of the longer tendon (kN)
 S_b = Spacing between rows of the same bolt type (m)
 S_t = Spacing between rows of the same longer tendon type (m)
 w_e = Roadway width (m)

This rating considers all support installed at the face from the continuous miner (or mobile bolter where place changing is used) whether in the same row as the primary bolt type or not. It also considers each type of support separately and adds the values for each into a single value. The capacity for each support element is taken as the typical Ultimate Tensile Strength (UTS, kN) given in the product catalogues of the various suppliers.

Where some support elements may be longer than the primary bolt type, only the length of the primary bolt type is considered; for example where 2.1m bolts are being installed and longer tendons are also being used, a simulated value of 2.1m is assigned as the length of the longer tendons (i.e. L_b remains constant). The longer tendons were found to unfairly influence the rating if their entire length was included in the calculation.

GRSUP Calculation

The GRSUP rating incorporates all bolt and longer tendon roof support installed within the roof of a roadway into a single rating, regardless of when the roof support is installed. This includes all roof bolts, longer tendons, cables and trusses. The GRSUP is calculated in a similar manner to that of the PRSUP; in fact if no additional support is installed within the roof subsequent to that installed off the continuous miner or mobile bolter then GRSUP will equal PRSUP. The rating value for each type of roof bolt, tendon or cable is calculated and the values summed as a single number representing the total installed tendon roof support capacity.

Once again only the length of the primary roof bolt (i.e. referred to as the bolted horizon, L_b) is considered when calculating the influence of longer tendons or cables. The GRSUP is calculated as follows:

$$\text{GRSUP} = \frac{L_b}{14.5 \cdot w_e} \sum N_m \cdot C_m$$

where m = number of different support types
 N = number of support elements per metre
 C = Ultimate tensile strength of each support element (kN)
 w_e = Roadway width (m)

To clarify the PRSUP and GRSUP Ratings, consider the following example.

Example Calculation

A mine installs 6 x 2.1m X-grade bolts (UTS 340kN) at 1.2m spacing on development with 2 x 6m high strength tendons (UTS 580kN) also installed from the continuous miner between every second row of bolts (i.e. 2.4m spacing). Before the 2nd adjacent longwall begins extraction, a further 2 x 8m long single strand cable bolts (UTS 265kN) are installed every 2m. The roadway width is 5.2m.

For PRSUP, only the support installed off the miner is included, i.e. the 2.1m X-grade bolts and 6m tendons.

$$\begin{aligned} \text{PRSUP} &= \frac{L_b \cdot N_b \cdot C_b}{14.5 \cdot S_b \cdot w_e} + \frac{L_b \cdot N_t \cdot C_t}{14.5 \cdot S_t \cdot w_e} \\ \text{PRSUP} &= \frac{2.1 \times 6 \times 340}{14.5 \times 1.2 \times 5.2} + \frac{2.1 \times 2 \times 580}{14.5 \times 2.4 \times 5.2} \\ \text{PRSUP} &= 47.3 + 13.5 \\ \text{PRSUP} &= 60.8 \end{aligned}$$

To calculate GRSUP all support elements are included (i.e. 2.1m bolts, 6m tendons and 8m cables) such that:

$$\text{GRSUP} = \frac{L_b}{14.5 \cdot w_e} \sum_{m=1}^m N_m \cdot C_m$$

In this case there are three support types, so m=3 and therefore:

$$\begin{aligned} \text{GRSUP} &= \frac{L_b}{14.5 \cdot w_e} (N_1 \cdot C_1 + N_2 \cdot C_2 + N_3 \cdot C_3) \\ \text{GRSUP} &= \frac{2.1}{14.5 \times 5.2} \times \left(\frac{6}{1.2} \times 340 + \frac{2}{2.4} \times 580 + \frac{2}{2} \times 265 \right) \\ \text{GRSUP} &= 68.2 \end{aligned}$$

ESTIMATION OF COAL PILLAR STRENGTH BY FINITE DIFFERENCE MODEL

Kazem Oraee¹, Navid Hosseini², Mehran Gholinejad³

ABSTRACT: Longwall mining is now predominately used in coal mines where somewhat difficult conditions exist. As in the case of all other underground mining methods, pillars are integral part of modern mine design. The process of pillar design in longwall mining entails the selection of a safety factor, which is done by estimating the magnitude of the load applied on the pillar and the load bearing capacity of such pillars. Finite difference modelling principles have been applied to a typical coal pillar design. The pillar strength is then estimated with various width/height ratios. These results have been compared with the results obtained from the conventional pillar design methods. The effect of roof and floor quality on the strength of the typical pillar has also been evaluated in the same manner. Although the finite difference method is not always the perfect method for such estimation, nevertheless, the results clearly demonstrate that it produces more acceptable design than the conventional method, especially under undesirable conditions regarding the interface between pillars, roof and floor. An additional advantage of such method is its capability of being applied in situations where complex parameters prevail.

INTRODUCTION

In recent years, significant research has been carried out on the determination of the coal pillar strength, and various formulas are introduced. The majority of these formulas are empirical, based on pillar shape effect, dimension and laboratory testing of coal. The work of Gaddy, Holland, Obert & Duvall, Salamon & Munro, and Bieniawski formulas (Hosseini Navid, 2007) are notable examples. The results obtained from empirical formulas are reliable only with special limits. These limits are determined based on the initial condition of formula presentation. In addition, the empirical formulas are not considering the effect of surrounding roof and floor on coal pillar strength. While the friction at the interface of coal seam and footwall and hanging wall have significant effects on coal pillar bearing capacity.

Numerical techniques such as finite difference, finite element, distinct element, etc. are other methods of estimating the coal pillar strength. Based on these methods the varieties of software are presented. In this paper, finite difference method (FDM) and FLAC code (ITASCA, 2005) are used for modelling and strength analysis of coal pillar. For logical estimate of coal pillar strength the essential parameters considered are; coal properties, pillar geometry as well as the condition of surrounding roof and floor.

NUMERICAL MODELLING

FLAC software

The Fast Lagrangian Analysis of Continua (FLAC) provided by Itasca Consulting Group, Inc., and is a two-dimensional explicit FDM program. FLAC is well accepted by social mining and rock mechanics engineers and this is why it was selected for this study.

Pillar design by using FDM modelling

FDM modelling technique was used for prediction of coal pillar strength. The main advantage of this method is the integration of the surrounding roof and floor conditions on coal pillar strength.

For modelling of pillars, the two-dimensional FDM model is used. A typical coal pillar with 3.0 m (10 ft) in height was selected, and then the compressive strength of coal pillar for W/H ratios of 1 to 15 is calculated, based on different widths. Figure 1, shows two coal pillars with 3.0 m (10 ft) height and different widths.

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In FLAC modelling, the strain-softening model is used to produce the peak strength behaviour for the pillar and the pillar behaviour by FISH functions is monitored. For determining the pillar bearing capacity, the downward force is applied on the top surface of the model, until the pillar fails. During the pillar failure simulation, the average of vertical stress at mid height of the pillar was calculated at regular intervals and the peak value of this stress was considered to represent the peak pillar strength. Also, by averaging the displacement values between the top and bottom of the pillar and dividing by the original length, the pillar strain is thus obtained. Therefore, the pillar stress-strain curve could be developed.

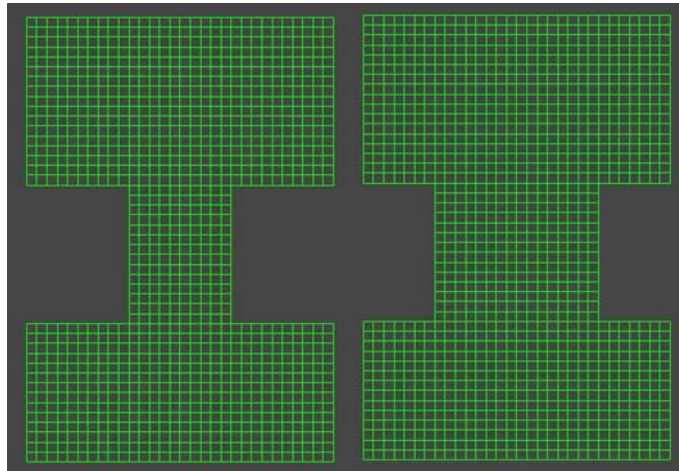


Figure 1 - Modelled coal pillar with 3.0 m (10 ft) height and different widths

To develop the study, the various conditions of roof and floor in FDM model are also applied. Typical coal pillars with both soft and strong roof and floor are considered. In this modelling, the height of the pillar is also 3 m (10 ft) and different widths are selected. Thus, the pillar strength for both soft and strong conditions of roof and floor, for various W/H ratios are calculated. In addition, to study the role of coal strength in coal pillar strength, two pillars with different coal strength properties were modelled by FDM. Following this study, the effect of parting in coal seam on coal pillar strength was analysed. For this purpose, the typical pillar with parting in coal seam was modelled and the effects of parting with different properties of rocks for various W/H ratios were determined. Typical coal pillars modelled by FDM in conditions of, with and without parting of the coal seam, is shown in Figure 2.

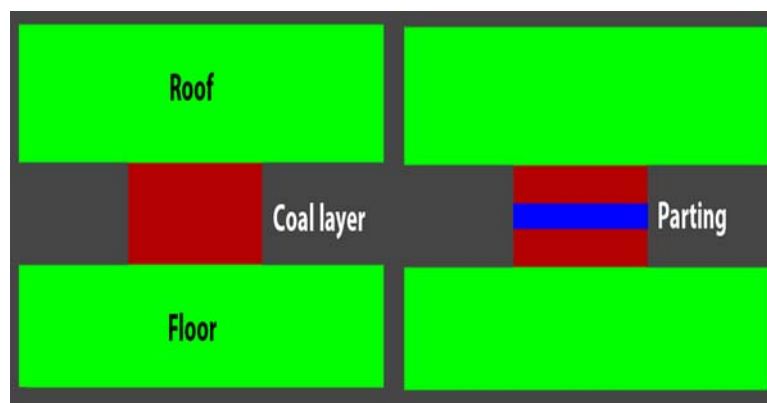


Figure 2 - Typical coal pillars modelled by FDM, with and without parting

COMPARING FDM MODELING RESULTS WITH EMPIRICAL FORMULAS

The compressive strength of typical coal pillar with 3 m (10 ft) height and different widths, based on modelling by FDM, was thus calculated. The selected coal properties were similar to US Pittsburg colliery (Maleki H, 1992) as shown in Table 1. The results and their comparison with strength values obtained by conventional formulas are shown in Figure 3.

Table 1 - Coal properties

<i>property</i>	<i>values</i>
Young's modulus	4 GPa
Poisson's ratio	0.30
Bulk modulus	3.3 GPa
Shear modulus	1.5 GPa
Uniaxial compressive strength	8 MPa
Density	1400 Kg/m ³
Tensile strength	0.9 MPa
Cohesive	0.8 MPa
Friction angle	30 degree
Dilation angle	0 degree

As demonstrated in this figure, up to W/H ratio of 4, the results of empirical formulas and FDM are similar. By increasing the W/H ratio, the resembling of these results decreases, except the result of Salamon & Munro formula, in which increases in W/H ratios, the results tallied reasonably with to FDM modelling values. According to previous studies (Daniel W. H. Su, Gregory J. Hasenfus, 1999; Oraee K., Hosseini Navid, 2007), the validity of empirical formulas results is up to W/H ratio of 4, however the FDM estimates of the coal pillar strength for higher W/H ratios are considered as valid. The comparison of FDM modelling results and several field data are illustrated in Figure 4.

In this modelling, the roof and floor condition are strong. The field data (Daniel W. H. Su, Gregory J. Hasenfus, 1999) were collected from failed pillars in US Pittsburg colliery. As, can be seen in Figure 4, at least one field data is in agreement with the FDM model, and the other is close. Based on probability of error in field measurement, perhaps the measuring accuracy is the reason of the last data field results not corresponding with the FDM model.

The effect of surrounding roof and floor on coal pillar strength

The majority of empirical estimation formulas of coal pillar strength inherently do not consider the effect of surrounding roof and floor rocks on coal pillar strength (Oraee K., Hosseini Navid, Qolinejad M., 2008a). In Figure 5, two coal pillars with same condition for various W/H ratios are modelled. The only applied difference in two models is the state of roof and floor strength. The result of FDM modelling shows that the strength of surrounding roof and floor rocks can influence coal pillar strength.

With increasing the W/H ratio, the effect of surrounding roof and floor strength increases, until the pillar coal strength changes more than 50%. Figure 5 shows the results of FDM modelling compared with several field data (Daniel W. H. Su, Gregory J. Hasenfus, 1999) with soft surrounding roof and floor. The results of FDM modelling and field data are reasonably close.

In Figure 6, the results of FDM model for strong surrounding roof and floor, is compared with Bieniawski formula and field data (Daniel W. H. Su, Gregory J. Hasenfus, 1999). The FDM model results and Bieniawski formula are similar up to W/H ratio 4. However, with increasing the W/H ratio, the estimated strength of FDM modelling is significantly higher than Bieniawski formula results. In addition, the results of FDM for strong roof and floor formation and that the field data of failed pillars under similar conditions are close. However, the field results differ significantly when compared with Bieniawski formula results. The main reason is in the empirical nature of Bieniawski formula. Bieniawski formula is based on studies of coal pillars with soft surrounding roof and floor (Oraee K., Hosseini Navid, Qolinejad M., 2008b) and therefore, only for such condition the formula is valid. Figure 7 shows the results of FDM model for soft roof and floor formation as compared with each of Bieniawski, Holland and Gaddy formulas and the field data. As can be seen in the figure, the results of FDM modelling and the field data of failure pillars are fairly close with soft surrounding roof and floor. In addition, in low W/H ratios the results of Bieniawski formula and field data is approximately the same. Therefore, in result, the Bieniawski empirical formula is valid only for low W/H ratios and soft surrounding roof and floor. However, the Holland and Gaddy formula is based on soft roof and floor formation (Oraee K., Hosseini Navid, Qolinejad M., and 2008b), underestimate coal pillar strength, and therefore the results are conservative.

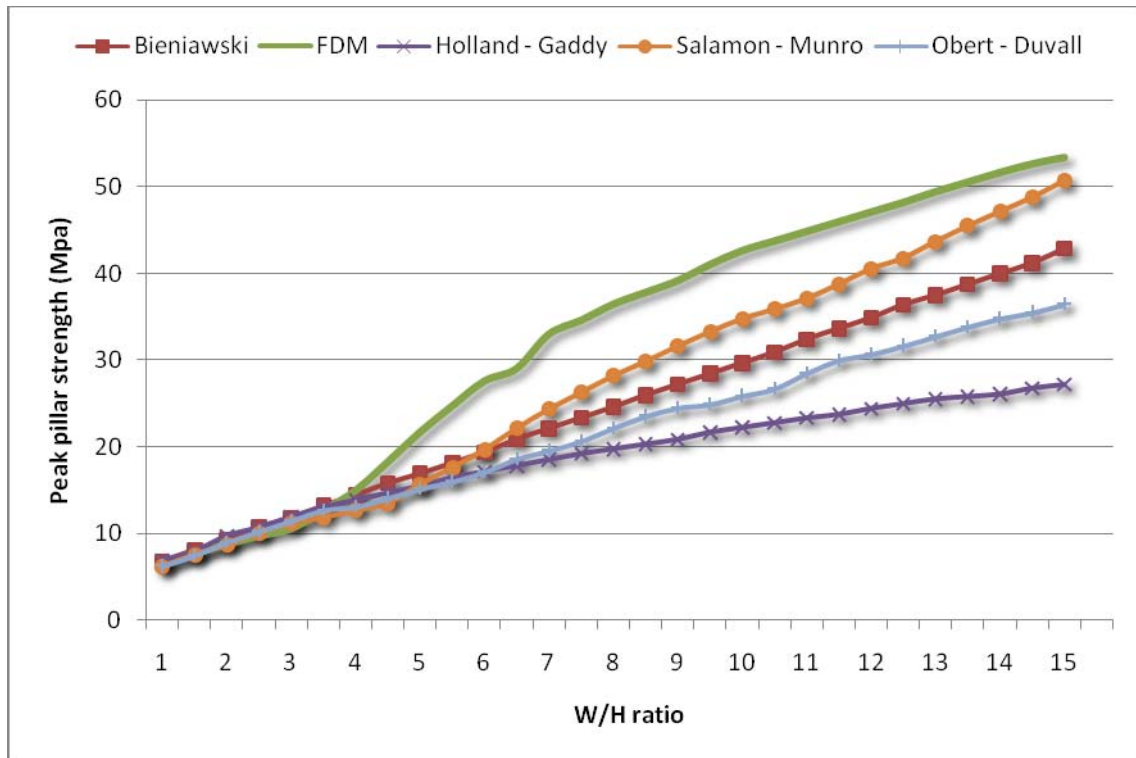


Figure 3 - The comparison of coal pillar strength by FDM modelling with empirical formulas

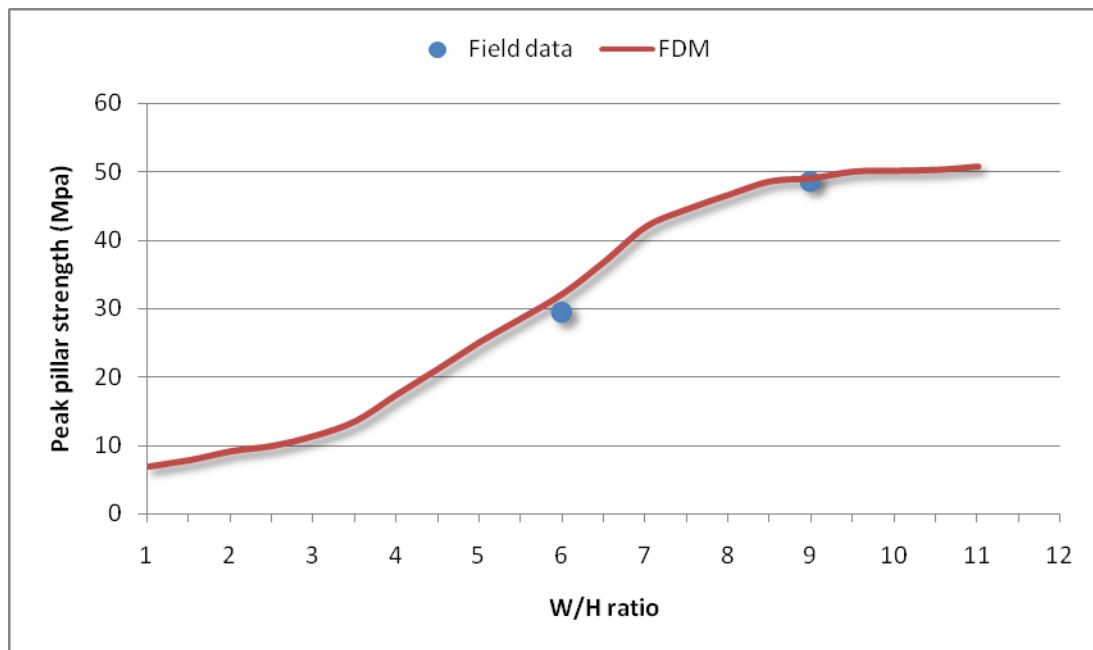


Figure 4 - The comparison results of FDM model and field data

Generally, the results of FDM modelling for soft surrounding roof and floor have more validity than the results of the empirical formulas. Because, the FDM results and the field data are very close. Similar results can be seen for strong surrounding roof and floor formation. Large differences exist between the results of the empirical formulas and the field data. Therefore, only the FDM estimation can be considered as credible.

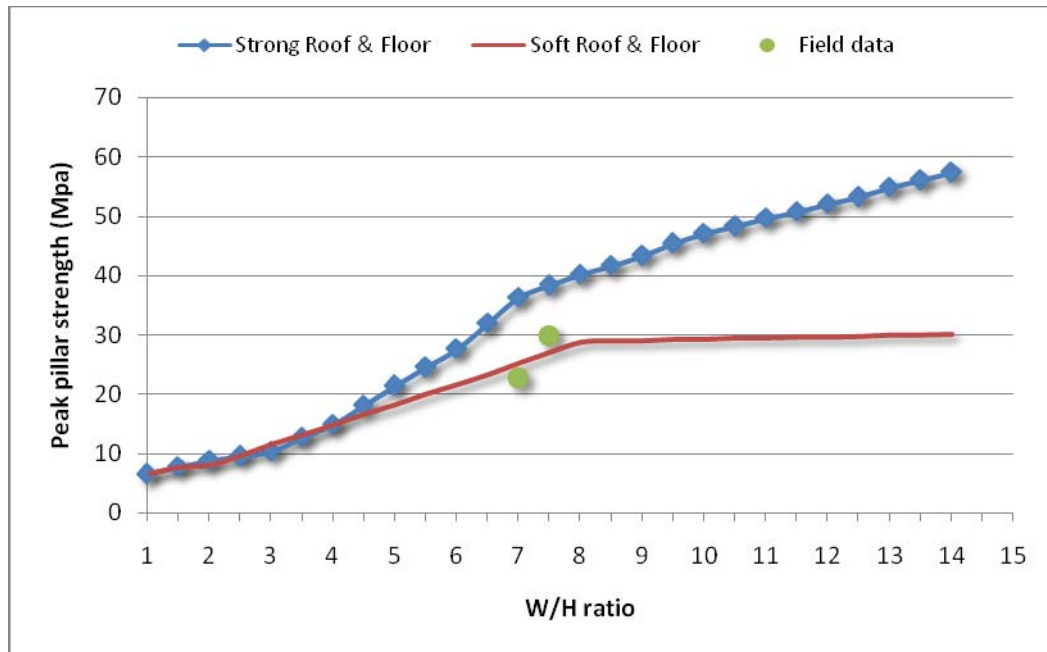


Figure 5 - The effect of surrounding roof and floor strength on coal pillar bearing capacity and comparison the results of FDM modelling with field data

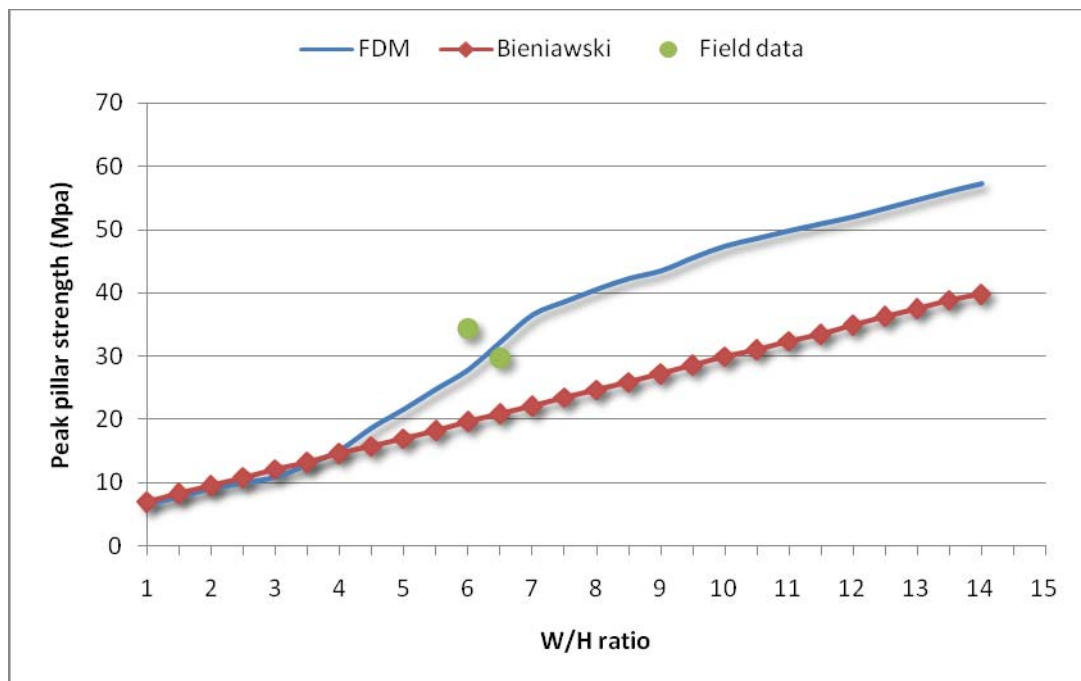


Figure 6 - Comparison the coal pillar strength of FDM modelling for strong surrounding roof and floor with Bieniawski formula and field data

The effect of coal seam strength on pillar strength

The assumption that coal strength has a significant role in coal pillar strength, is an incorrect statement. The results of two modelled coal pillars are shown in Figure 8. All conditions of both pillars assumed the same except in coal seam strength. (Hosseini Navid, 2007)

As can be seen in Figure 8, the results of FDM modelling for both pillars with 5.5 and 6.9 MPa (800 and 1000 psi) coal seam strength are very close. Therefore, the effect of coal seam strength on coal pillar strength is poor and negligible.

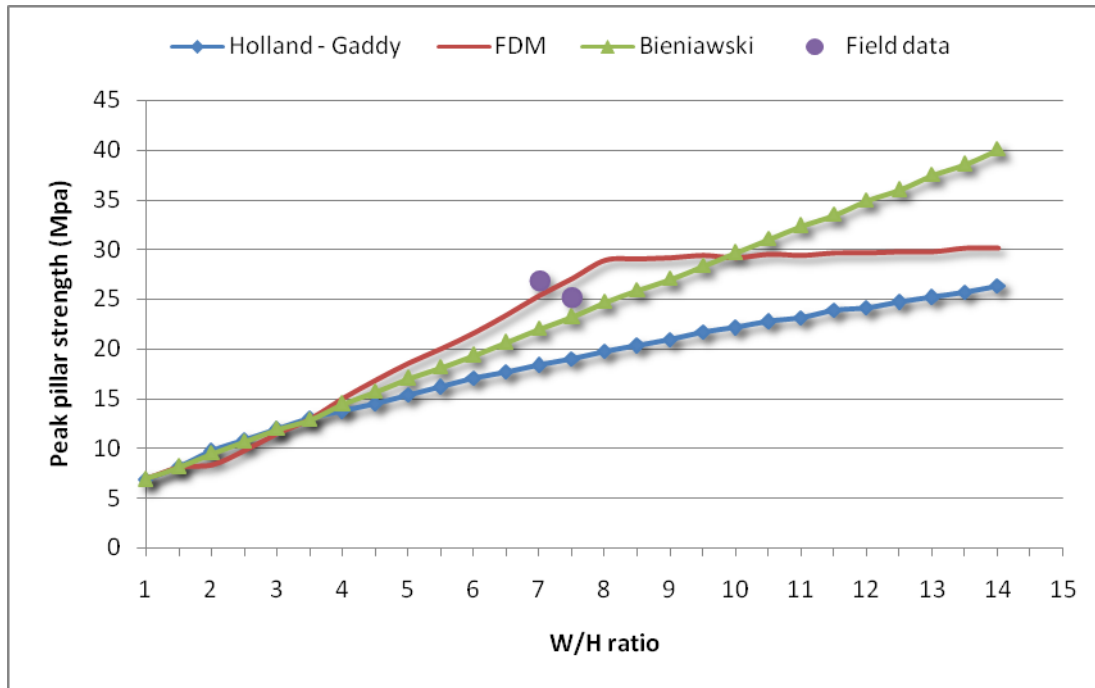


Figure 7 - Comparison the coal pillar strength of FDM modelling for soft surrounding roof and floor with Bieniawski and Holland and Gaddy formulas and field data

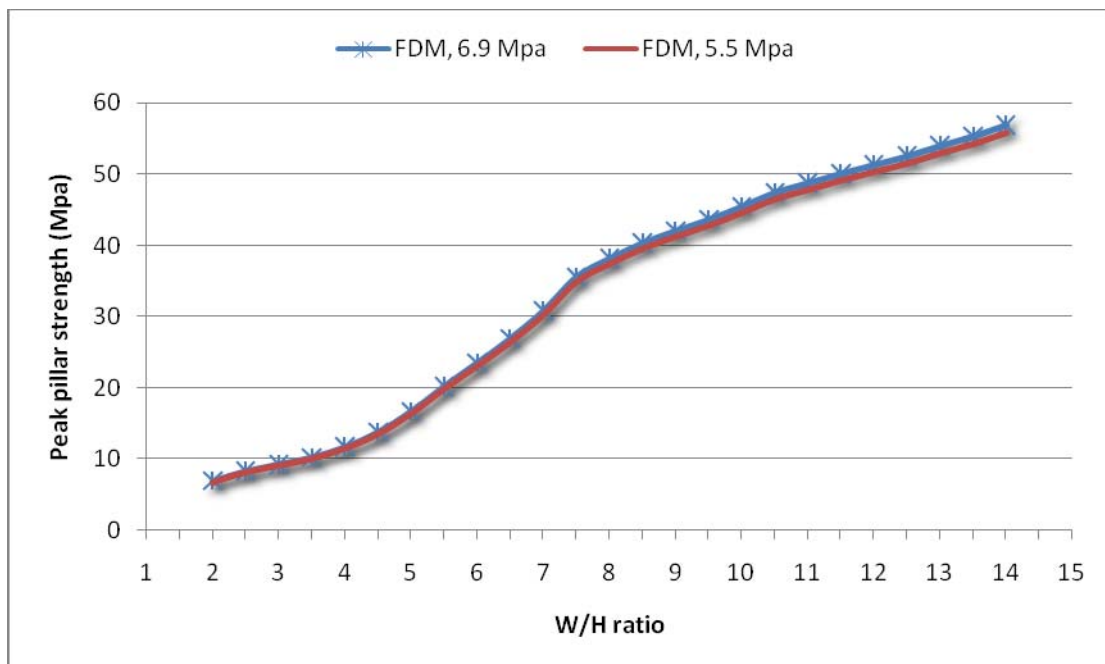


Figure 8 - The effect of strength properties of coal seam on coal pillar strength in FDM modelling

The effect of parting on coal pillar strength

Figure 9, shows three FDM modelled pillars with the same properties. The only exception is the parting. One of modelled pillars is without parting, another with the parting stronger than the coal seam, and the third with the parting but softer than the coal seam.

As shown in Figure 9, the parting is an effective parameter on coal pillar strength. The coal pillar strength is changed, based on strength properties of the parting. In theory, the distribution of field stress within pillar is disturbed by the parting. The stress concentration point within the pillar, forms and consequently, the probability of pillar failure increases. However, if the parting is stronger than the

coal seam, and resists the stresses, the coal pillar strength with the parting increases. This is because, the applied stress on pillar, concentrate within the parting and hence reduces the stress in pillar coal zone.

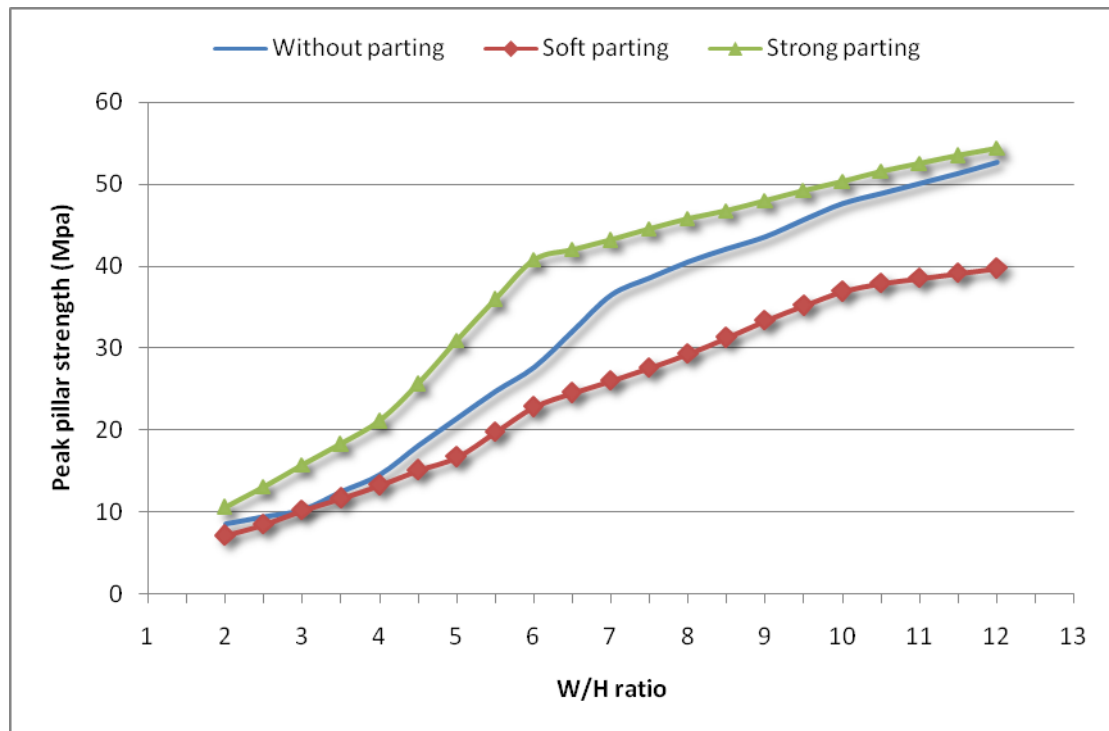


Figure 9 - The effect of the parting on coal pillar strength in FDM modelling

CONCLUSIONS

The FDM modelling can properly and accurately estimate the coal pillar strength. The strength properties of the surrounding roof and floor affect the coal pillar strength and this effect increases with increasing W/H ratio. The comparisons of FDM modelling results for the surrounding roof and floor of various conditions and field data, emphasises the effect of the state of surrounding roof and floor on the coal pillar strength, while most empirical formulas ignore the surrounding roof and floor effect.

Assuming that the coal pillar strength is directly proportional to strength properties of the coal seam, however the results of FDM modelling show that the effect of coal seam strength on coal pillar strength is negligible.

The parting within coal seam changes the coal pillar strength. If the strength of parting is more than coal seam, the coal pillar strength increases, otherwise, the coal pillar strength decreases with the parting.

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STRESSES IN THE IMMEDIATE STONE ROOF OF A COAL MINE ROADWAY

Ross Seedsman¹

ABSTRACT: An analytical approach to designing roof support needs a model that quantifies the magnitude of the stresses acting in the immediate roof. There is a large amount of measurement data on stresses above pillars but few models for the stresses within the bolting horizon. Very small roof deflections result in substantial relaxation of horizontal stresses in the immediate roof. Increased vertical and horizontal stresses at the maingate increase the height of compressive failure and hence the "softened zone" and this leads to greater loading on the bolted roof beam. The situation does not change in a material way at the tailgate unless there is major yielding of the side of the roadway in which case the horizontal stresses in the roof line may become tensile.

INTRODUCTION

ACARP project C14029 documents a series of analytical tools to assist in specifying roof and rib support. These tools are applied in a logical framework in which there are tests for compressive and tensile failure and tests for movement along discontinuities. Models for the stresses applied to the bolted horizon were required. There can be no doubting that considerations of the stress redistribution are complex, involving interactions between the in-situ stresses and the large longwall goaf, stress redistribution around the roadway itself, and any body stresses induced by the deformation and movement of the immediate rock and coal mass. This paper provides a summary of the model development for stone roof.

IN-SITU STRESSES

It is not possible to predict the state of stress in the ground from simply a knowledge of the depth of cover. The approach should be to apply regional knowledge of the general stress field to any point measurements at the mine site, and particularly to observations of how excavations behave underground. In the latter case, it is essential not to jump to the paradigm that all roof falls are due to elevated horizontal stresses; this is unlikely to be the case and certainly not once the roof is supported at the densities typical of current Australian coal operations.

Data from New South Wales and Queensland (Nemcik et al, 2006) show that the ratio of the horizontal stress to the vertical stress in stone is typically between 1.0 and 2.5 times at typical mining depths, with even higher ratio values at shallow depths. It is noted that there is a significantly different stress field in coal.

Stress magnitudes should be lower in faulted ground. Elevated deviatoric stresses generate failure and on a large scale this is evidenced as faults, more often than not with associated sub-parallel joints. Once the rock is broken, the maximum deviatoric stresses within the broken rock are lower and are controlled predominantly by the frictional resistance of the surfaces generated by the faulting. The poor roof conditions that are typically encountered in the vicinity of thrust faults are not the result of elevated horizontal stresses but are a consequence of the presence of broken rock.

REDISTRIBUTION ABOUT A LONGWALL

Our knowledge of stresses and stress changes is based on measurements in stone above the chain pillars and about 5m - 10m into the roof. These are not the stresses at the roof line.

Using standard chain pillar design methods (Colwell, 1998), the vertical stress is doubled at the maingate corner. There is a concentration of horizontal stresses above the chain pillar at the maingate corner and a reduction in the horizontal stresses behind the face adjacent to the goaf

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(Figure 1). The magnitude of the concentration of the major principal horizontal stress depends on the angle between the stress axis and gateroad direction, with the possibility that there is a doubling of the magnitude at a 45° angle.

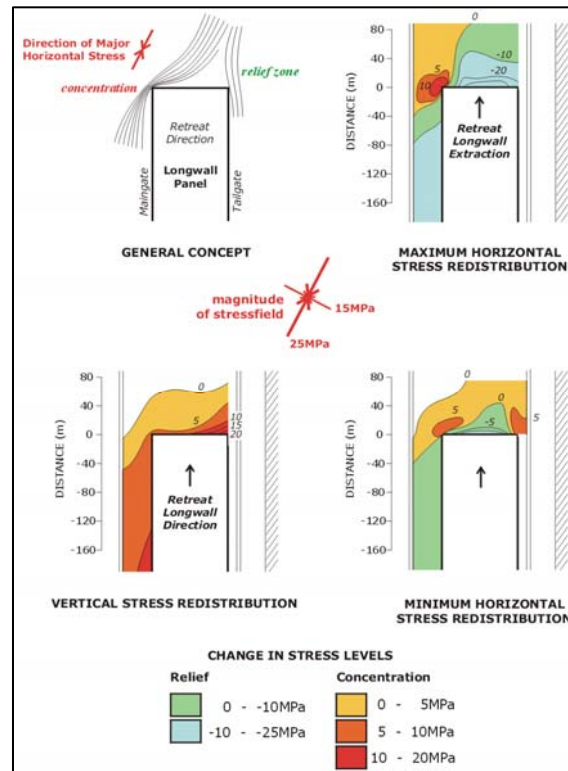


Figure 1 - General pattern of vertical and horizontal stress redistribution (Gale 2008)

There is also an increase in the vertical stress observed above the pillar behind the face and a large reduction in the horizontal stress. Tarrant (2006) provides the mechanism for this large stress relief – shear along bedding in the direction of the goaf.

Shen et al (2006) instrumented a tailgate at Ulan at a depth of about 200m and showed that overall the vertical stress increased by 3-4 MPa while at the same time the horizontal stress decreased by up to about 1 MPa. The overall pattern in the tailgate is consistent with increase in vertical stresses implicit in the pillar design models and a further reduction in horizontal stresses resulting from more lateral translation of the roof into the now two longwall voids.

The simultaneous increases in both the horizontal and vertical stress are significant in terms of the stresses induced in the immediate roof of an excavation. Depending on the concentration factor that applies to the horizontal stress, it is possible that the “vertical” stress acting in the maingate may become the largest stress component, and certainly the vertical stress is the largest stress component behind the face and in the tailgate. There is no published information on the rotation of the principal stress axes around a longwall. Making the simplifying assumption that the vertical stress is a principal stress, Figure 2 shows the way in which the K ratio (horizontal/vertical) can change.

ELASTIC STRESS REDISTRIBUTION AROUND A RECTANGULAR ROADWAY

The initial response of a roadway can be considered to be elastic and many of the failure modes that are seen develop in response to these elastic stress redistributions. It is important to realise that elastic analyses are only applicable for the initial formation of the roadway as it has been shown that even small deformations result in major relaxations in the roof and redistributions to elsewhere in the system (see later).

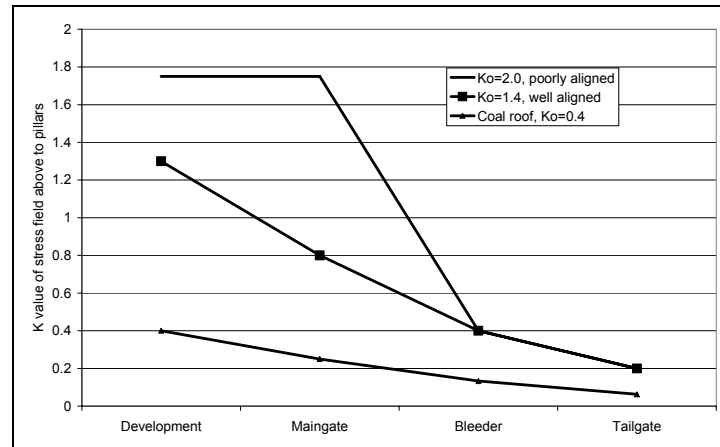


Figure 2 - Indicative changes in the K ratio

Figures 3 and 4 present the results of the analyses of a typical rectangular roadway (2.8m high and 5.2m wide) for two K ratio values - 2.0 and 0.2 with the major principal stress being 10 MPa. Two stress components are presented: deviatoric stress ($\sigma_1 - \sigma_3$) which is the driver for compressive/shear failure (Figure 3) and σ_h - the horizontal stress which, if tensile, would allow the onset of shear along vertical joints (Figure 4). A summary of the results for a range of other K values is presented in Table 1.

When the K value is greater than 1, the contours of deviatoric stress tend to form an arch over the roof line but this does not develop when the K values are less than 1 (Figure 3). The highest values are at the roof/rib corner with the magnitude ranging from about 30 MPa for a K value of 1.0 to about 21 MPa for higher or lower K values. It is this concentration at the roof corners that is one of the mechanisms for stress guttering.

Table 1- Summary of stresses for rectangular roadway ($\sigma_1 = 10$ MPa)

K	Horizontal stress at 0.2m into roof at centerline (MPa)	Horizontal stress 0.1m from rib and 0.1m into roof (MPa)	Maximum deviatoric stress 0.1m from rib and 0.1m into roof (MPa)
0.2	-5	8.3	19.5
0.5	-1	13.5	21
1.0	6	24	27
1.4	8	22	24
1.7	9	22	21
2.0	10	21	21

The centreline of the roof has tensile horizontal stress for K values less than 0.5, and the roof stress becomes increasingly compressive as the K value increases. Near the rib line the horizontal stresses are compressive for all K values (Figure 4).

NON LINEAR STRESS REDISTRIBUTIONS

Non-linear (=not elastic) stress behaviour is a key feature of underground roadway behaviour. Unfortunately, with the current state of the art, it is difficult to incorporate this behaviour into numerical analyses, and certainly not in routine design. Fortunately, it would appear not to be necessary.

It is well established that coal mine roofs deform into the excavation as the roadway is advanced. This deformation zone is routinely identified with roof extensometry and is loosely referred as the "softened zone" or "the height of softening". Whilst the term is somewhat misleading, it does have the advantage of focusing attention on what the impact may be in terms of the immediate roof stresses. Softening implies a lower modulus of deformation, which should mean that there is less of an ability to bear stresses compared to stiffer units nearby.

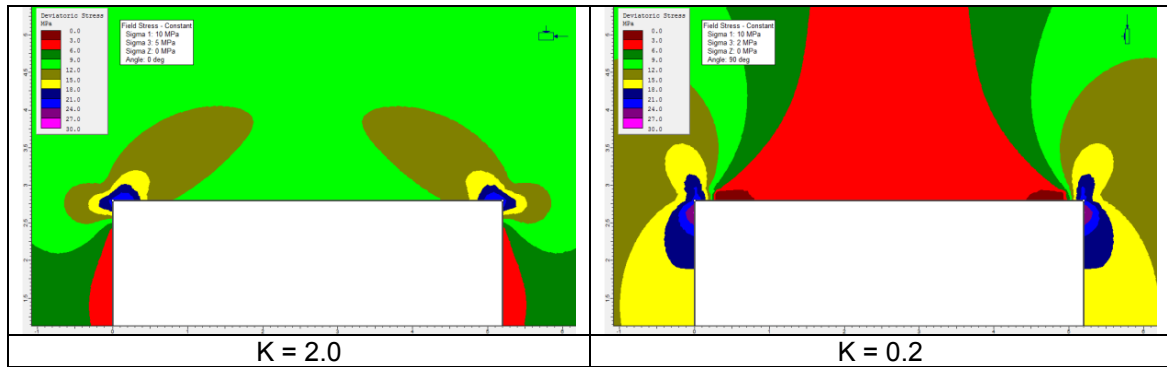


Figure 3 - Distributions of deviatoric stresses about a 1.86:1 roadway ($\sigma_1 = 10$ MPa)

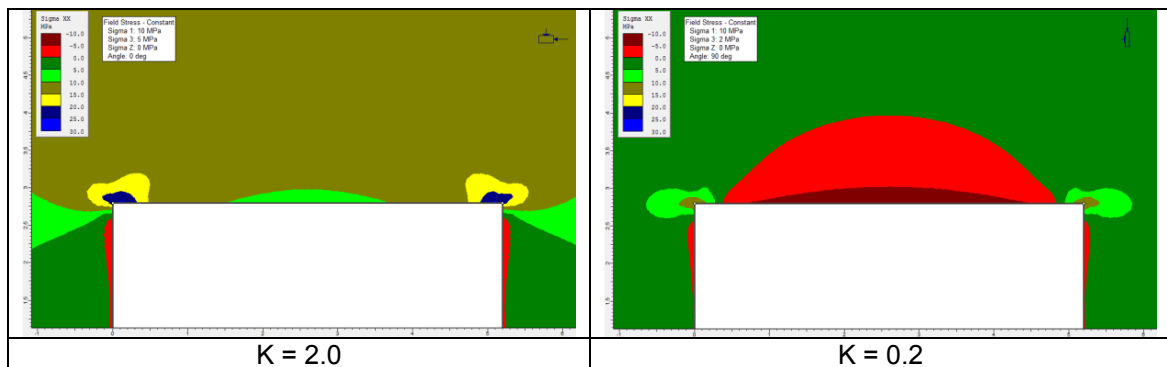


Figure 4 - Distribution of horizontal stresses about a 1.86:1 roadway ($\sigma_1 = 10$ MPa)

The scale of roof movement and the associated stress redistribution has been demonstrated recently by Mark et al (2007). These authors were able to show that even at less than 20mm of vertical roof movement; the horizontal stresses were already redistributed into an arch over the roadway (Figure 5). This redirection continued as the longwall retreated and the roadway was exposed to maingate stress concentrations.

Further evidence in support of the non-linear redistribution of stresses can be found in the data on stress relieving roadways (Figure 6, Gale and Matthews, 1992). Newton's Third law requires that the stress relief is also present within the existing roadway. The quantification of roof softening was not available at the time, but experience in the Southern Coalfield is that a bolted roof – not an overall collapse – is adequate to generate the stress relief.

Non-linear stress redistribution is problematical when considering the stresses in the roof during the formation of an intersection. The second roadway is driven in a completely different stress field to that of the first roadway such that the stresses in the intersection roof at the point of breakthrough will be less than those encountered during the straight drive.

STRESSES INDUCED WITHIN A BLOCKY ROOF

A rock or coal mass is not a continuum and it is possible that its behaviour as a discontinuous medium can significantly modify the stress around an opening. The simplification to rectangular blocks that is possible with coal measure strata allows consideration of two simple analogues.

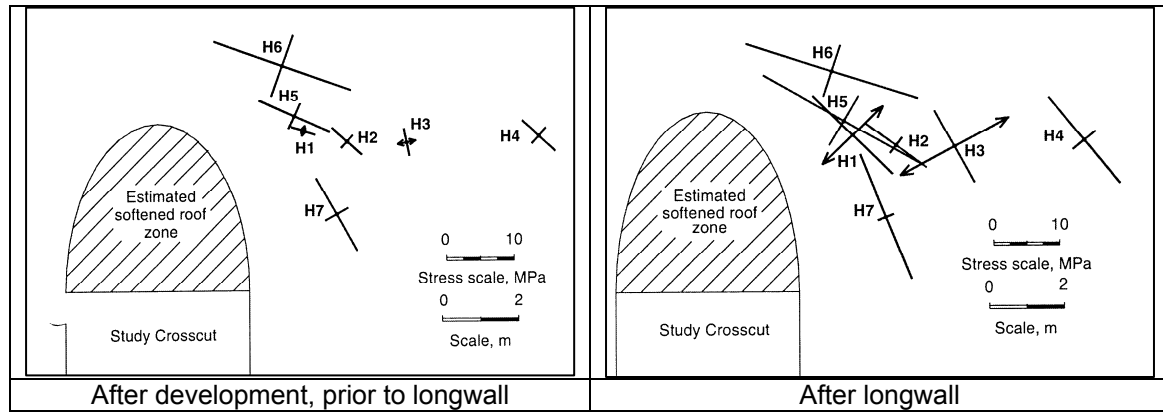


Figure 5 - Stress measurements at Emerald Mine

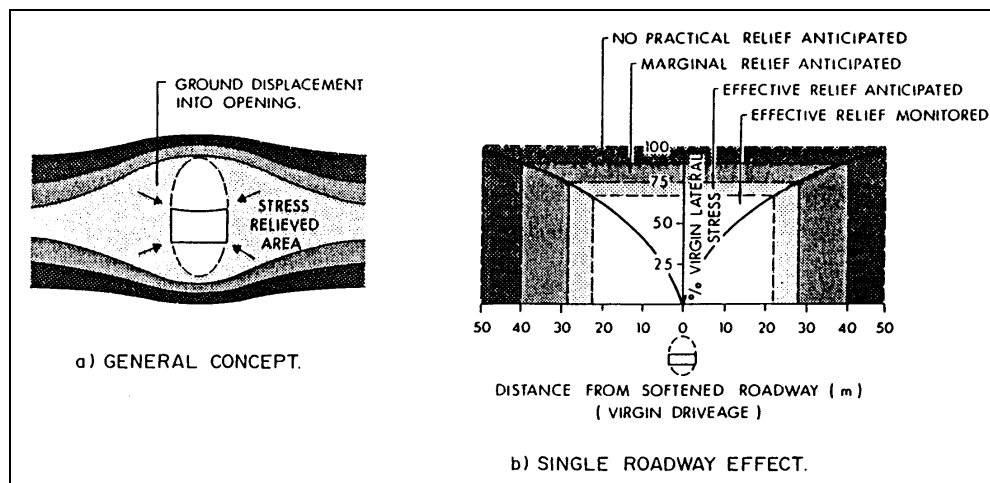


Figure 6 - Concept of a stress relieving roadway

Voussoir beams

The bedded nature of coal measures allows the ready application of the voussoir beam model (Brady and Brown, 1985). The concept is that under situations of no applied lateral force, the incipient rotation of the voussoirs induces a lateral thrust in the beam (Figure 7). The magnitude of this induced lateral thrust depends on the span, density and thickness of the beam. At the point of failure of voussoir beam, the compressive stresses at the roof/rib corner approach the magnitude of the UCS of the rock. An important point to note is that the result of voussoir action is the possible development of compressive stresses at the roof/rib corner together with tensile stresses at the roof centreline. An underground observer may observe the development of a "stress gutter".

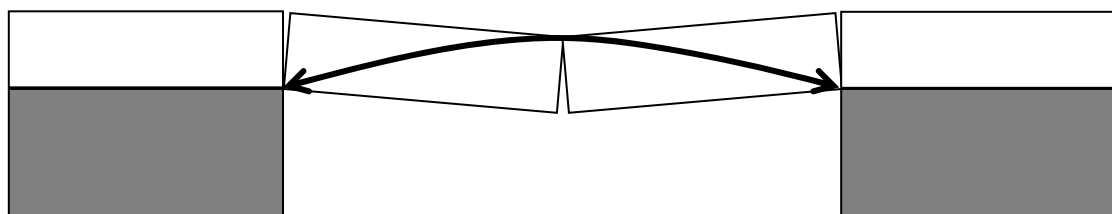


Figure 7 - Voussoir beam deformations induce compressive stresses at the roof corners and tensile stresses at the roadway centreline

Cantilevers

If the roof line is exposed to the onset of tensile stress and there is sufficient relaxation such that the joints dilate, it is possible that a cantilever will develop if joints are widely spaced. The deformation of a cantilever will generate elevated shear/compressive stresses at the roof/rib corner. Once again, there is the possibility of generating compressive failure in a situation of no imposed horizontal stress at the roof line.

OTHER STRESS REDISTRIBUTIONS

If there is differential movement between the two sides of a roadway, the result can be an increase in the bay length of the roof line (Figure 8) and a consequent reduction in the stress acting across the roof (Diederichs and Kaiser, 1999). The scale of this effect assuming the roof line is in the order of 1MPa to 2MPa for a differential compression of 100mm, with a greater reduction for roofs with higher modulus values. This may be significant when it is recalled that the stresses at the roof line after the formation of the roadway may be already low as a result of the stress relief into the goaf and non-linear effects discussed above. A potential location for even large differential movement is when the roadway is bounded by a yielding pillar or coal fender and this is considered to be the basis of the relationship between chain pillar design and tailgate roof support discussed by Colwell (1998). Another situation may be if there is yielding in the floor related to low strength claystone horizons.

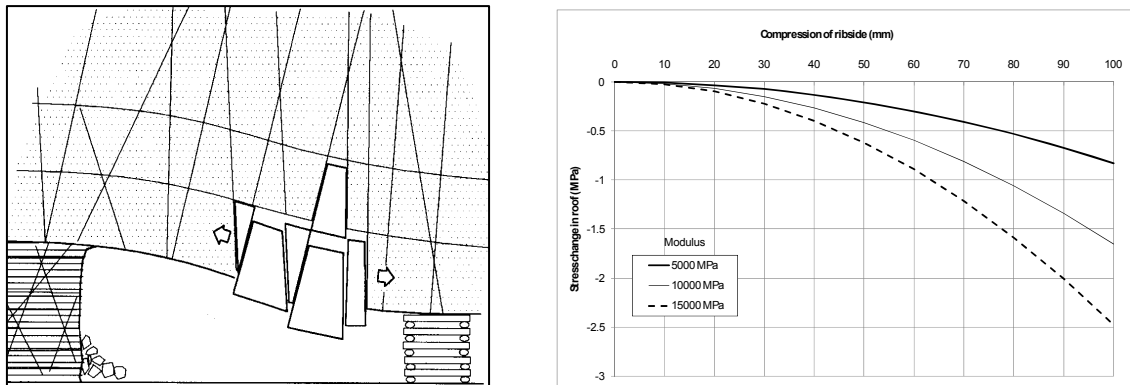


Figure 8 - Relaxation of a roof line as a result of vertical deformation in one of the sides

COMPILED MODEL FOR STRESS IN THE IMMEDIATE ROOF

Based on these previous discussions, the following is a progression through the stress history of the immediate roof of a coal mine roadway. The possible failure modes that may be induced are also included.

At the point of excavation, the reaction to the overburden load is removed and the vertical stresses at the roof line vanish. Until the roof deforms, the horizontal stresses are not yet redistributed. High deviatoric stresses and bedding-parallel shear stresses develop immediately. These stresses may induce stress guttering at the roof corners, either by compressive failure of low strength rock or by the incipient deflection of the roof beams defined by bedding partings that may be present. Simple elastic models can be used to quantify the stresses.

As the mining face advances, say to in excess of roadway width, the roof will have deformed to a 'final state', or in some situations failed if overall compressive failure develops or if the roof layers are too thin and the bolting has not been adequate. In all cases, the horizontal stress in the immediate roof will have decreased to very low values and a "stress arch" developed higher in the roof. Deflections of the bolted roof will generate body stresses associated with the formation of a voussoir beam in the immediate roof (Figure 9). The stresses within the softened zone cannot be quantified but the height of the softened zone can be estimated with simple elastic models.

At the maingate, increases in both the vertical and horizontal stresses around the retreating goaf will alter the stress arch above the roadway and this will lead to an increase in the height of softening. Elastic models can be used to estimate this height. There is no direct increase in the horizontal

stress acting in the immediate roof because that roof has already deformed and “softened”. The extra softened material under the stress arch will be an additional surcharge loading on the bolted beam that will then cause an increase in the body stresses within the voussoir arch and an indirect increase in the horizontal stresses at the roof line. This may lead to the onset of stress guttering.

Behind the faceline, the imposed horizontal stresses reduce and the vertical stress increases. The orientation of the principal stress may be skewed significantly off vertical and this could induce some small increases in the height of softening and the stress arch – at the roof line there are no material changes. Evidence from mining operations is that the changes do not induce roof collapses with the current mining geometries.

At the tailgate end of the face, the imposed stress field is now dominantly vertical with a reduction in the horizontal stress due to the presence of the goaf on one side and also behind the faceline. The horizontal stresses within the stress arch may decrease. The horizontal stresses in the immediate roof remain relatively constant and at low magnitudes, unless the chain pillar yields. In the latter case, reaction to the body stresses is lost and the horizontal stresses vanish. Roof collapse may result due to vertical shear along joints or fractures. Localised compressive stresses may develop if the roof structure allows the formation of cantilevers.

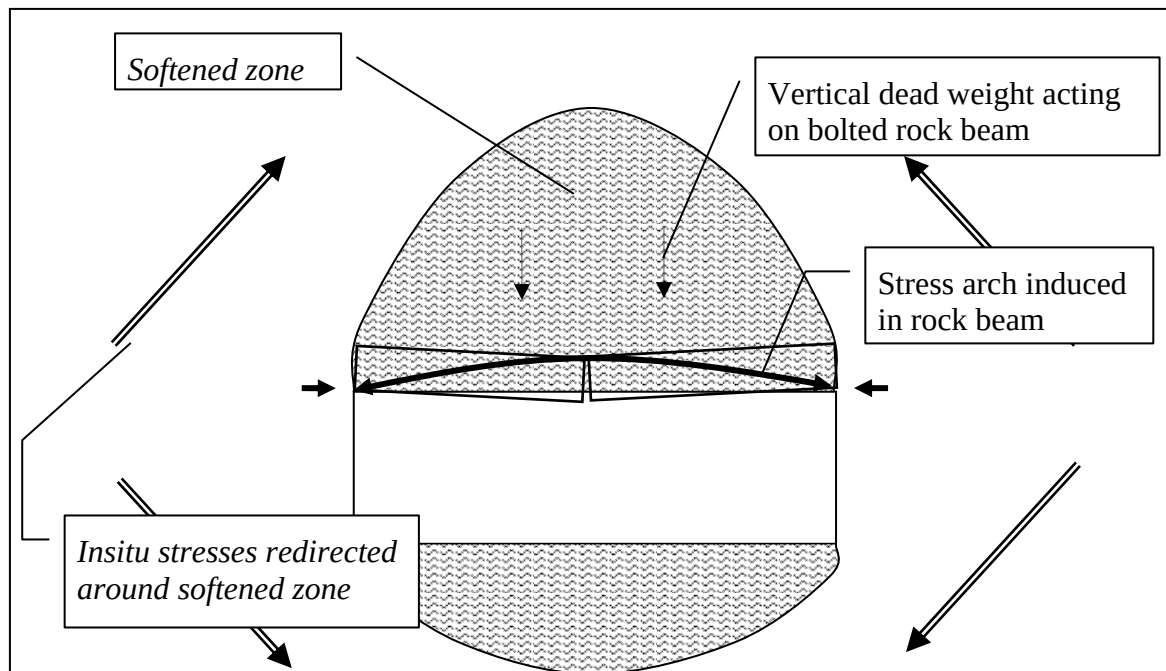


Figure 9 - Redistributed insitu and induced body stresses about a roadway with $K > 0.8$ once the roof and floor deflects.

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GEOTECHNICAL ASSESSMENT OF POLYMERIC MATERIALS AS SKIN REINFORCEMENT IN UNDERGROUND MINES

Jan Nemcik¹, Ian Porter¹, Ernest Baafi¹ and Christopher Lukey¹

ABSTRACT: Current advances in roof support automation require a fast and effective skin reinforcement of underground mine roadways. To satisfy these needs a strong and tough fibre reinforced polymeric alternative is emerging as a logical substitute to the old steel mesh support system. Differences between steel mesh and polymer skin behaviour are investigated. Computational models are utilised to compare these two skin support systems with a view to optimising the performance needed for effective roadway skin reinforcement. In particular, development of a strong and resistant shell that minimises movement along the fractured rock and coal surfaces found between the roof bolt anchors is recommended. A strong surface adhesion and the strength of a reinforced polymer skin can provide the necessary toughening mechanism required to enhance roadway surface support by forming a reinforced polymer/rock surface layer. The fractured rock mass in its undisturbed phase is relatively stiff while confinement stresses exist. However, any dilation that occurs due to displacement along the rough surfaces of the fractured rock causes strata softening, bulking and movement into the mine opening. The polymer skin can provide active resistance to any movement along the fractured rock surface as soon as any movement begins to occur. Even partial de-bonding of the polymer from the rock surface may not significantly disturb this mechanism.

INTRODUCTION

Steel mesh has been used successfully for many years to control friable roof conditions and prevent loose roof and rib material from caving into the roadway. To increase the speed of development rates of underground roadways, automation of the mining process is required. Despite its extensive use, the installation of steel mesh is difficult to automate and many other products have been trialled to take its place. Ideally, the properties of these new products should be similar or better than those of the steel mesh. Most of the Thin Spray-on Liners (TSL) trialled in the mines are weak with slow curing times, and the plastic mesh currently used to support the coal ribs is relatively weak, therefore neither material can seriously compete with steel mesh.

Currently, manually handled steel mesh is still the most widely used product to control friable strata in underground mines, however the automation process requires a suitable product that can replace the steel mesh. Now such a polymer product that cures in seconds and forms an instantaneous strata binder that surpasses the properties of steel mesh is under development at the University of Wollongong.

STRATA REINFORCEMENT SYSTEMS

It is impossible to prevent formation of mining induced fractures, however it is possible to successfully control fractured ground. The fractured rock mass in its undisturbed phase is relatively stiff while confinement stresses exist, however the loss in ground confinement results in strata softening, bulking and strata movement into the mine opening. In general, mining induced fracture surfaces are of an irregular nature and excessive shear displacements along such fractures cause significant strata dilation and therefore excessive convergence into the mine opening. It is common knowledge that reduction of strata movement is desirable for ground stability and therefore 'active' strata reinforcement is essential to minimise fracture displacements.

Historically, wooden props, sprags and arches provided passive roof and rib support that allowed large roof and rib displacements to occur before active resistance to movement was achieved. Such passive systems could not provide effective strata control and large amounts of support were required to control severe ground conditions resulting in slow mining advance and expensive labour intensive support systems that would not be suitable for today's modern high production mines.

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Effective strata reinforcement systems evolved over time with fully encapsulated high capacity steel bolts currently used as the primary reinforcing support, while high capacity cable bolting systems are used as the secondary reinforcement of severely deformed ground. It is essential that good reinforcement must be of a high capacity and stiffness to provide significant resistance to minute fracture movement whether in shear or dilation. Today's reinforcement systems ensure high strata confinement characteristics, low ground movement and superior ground stability in adverse conditions.

Although the success of steel bolts in ground reinforcement is undisputable, skin reinforcement of the mine roadways has not yet been optimised. Steel mesh has proven successful for the control of friable roof conditions but as with the wooden props its role is purely passive in nature. The steel mesh does not provide reinforcement to the strata and is exclusively used to prevent loose material from caving into the roadway.

THE ROLE OF STEEL MESH IN ROADWAY SUPPORT

Steel mesh is normally installed in mine roadways, tunnels and other underground openings where poor strata conditions prevail. Integrated together with the steel bolt reinforcement, the main role of steel mesh is to provide passive confinement to the fragmented rock surface that would normally fall out into the opening. In severe cases the steel mesh can prevent gradual degradation of loose material between the rock anchors where excessive cavities may form and affect the integrity of strata reinforcement.

To investigate properties of steel mesh, numerous testing programs have been undertaken (Tannant (1995), Thompson (2004), Villaescusa (1999) and our team) to quantify steel mesh properties and demonstrate the role of steel mesh in the civil and mining industries. The square or rectangular rock bolting pattern and the alignment of steel mesh wire have a significant influence on the load distribution in mesh. A typical steel mesh consists of 4-5 mm diameter drawn steel wire generally welded in a square or rectangular pattern. When loaded, steel mesh is stiffer in the direction parallel to the wire strands indicating that the row of bolt anchors installed parallel to the mesh wire strands would share most of the strata loads located in line between the bolts. The strata loads experienced elsewhere along the mesh may result in large displacements with some load distributed further away from the point of load application, depending on the bolt anchor pattern.

EXPERIMENTAL MEASUREMENT OF STEEL MESH BEHAVIOUR

The results of tests conducted to date indicate that steel mesh behaviour is complex and requires comprehensive tests and numerical modelling to predict its behaviour. A steel mesh deformation test, designed to allow calibration of the numerical models, is shown in Figure 1, while the ABAQUS computational modelling results for the same load are shown in Figure 2. While the general values of the experimental and model displacements are similar, the experimentally measured displacements were affected by the loading system.

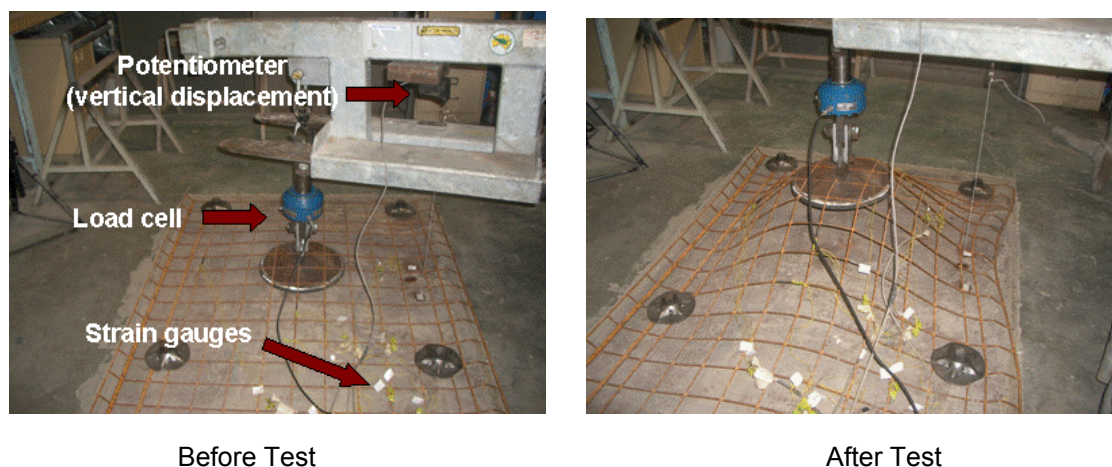
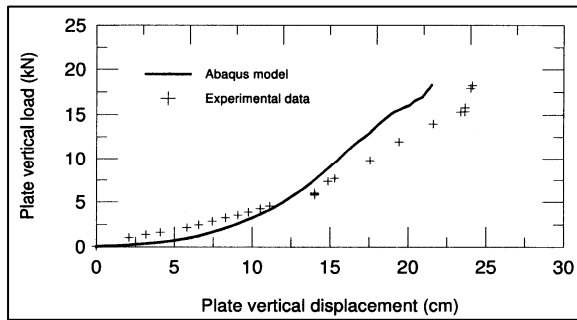
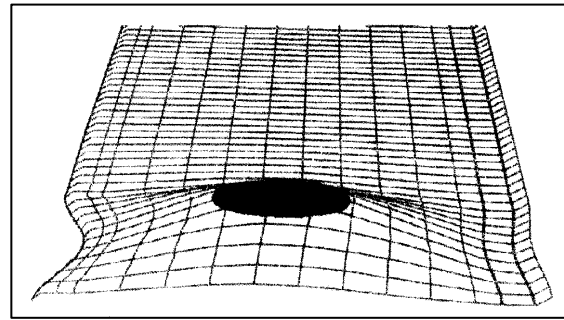


Figure 1 - Deformation Testing of Steel Mesh



Numerical Model



Experimental Data vs Numerical Model

Figure 2 - Numerical Modelling of Mesh Deformation

Both experimental results and numerical modelling involving stretching steel mesh at 45° to the steel strands (Figures 3 and 4) indicate that the mesh can deform easily, accepting strains of approximately 80% before mesh failure occurs. Apart from low stiffness, the ultimate strength of steel mesh welds loaded at 45° to the wire strands is approximately 40% of the wire strand tensile strength. The diamond pattern experienced more than 60% strain at approximately 50MPa when the welds connecting the mesh strands began to deform. The weakened steel wire, now aligned at a lower angle to the "stretch" direction, loaded quickly with welds failing at approximately 220MPa. Note that the typical tensile strength of steel wire is usually more than 500MPa. As expected, the welds joining the steel wire are weaker than the wire itself and can significantly reduce the ultimate mesh capacity if loaded in directions other than the wire strand direction.



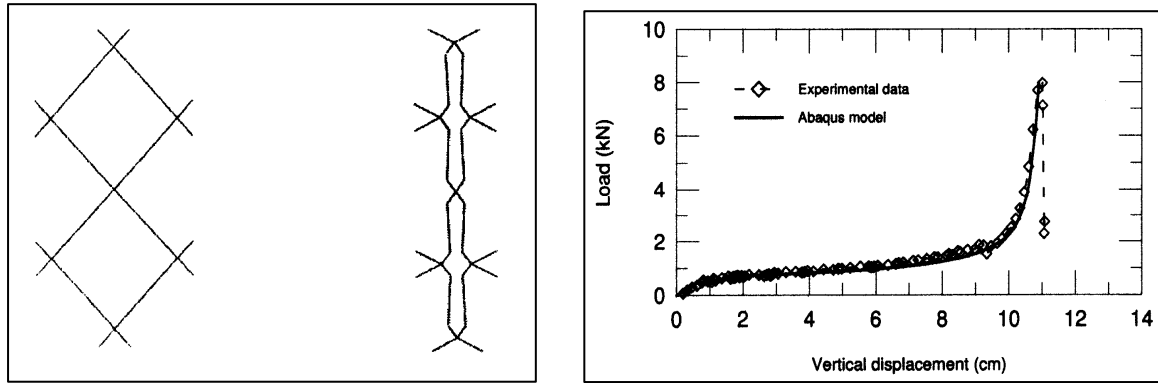
Before Test



After Test

Figure 3 - Steel Mesh "Diamond" Tensile Strength Test

Higher loads can always be expected along the wire strands that directly stretch between two adjacent bolts. These strands are often overloaded and can fail at low deflections. The subsequent loads applied elsewhere within the diagonal area would be expected to produce higher permanent mesh deformation at lower loads. If a mesh failure occurs in line between the bolts, diagonal displacements would follow and the progressive mesh failure would occur wire by wire with mesh deformations similar to those observed in the test described in Figures 3 and 4.



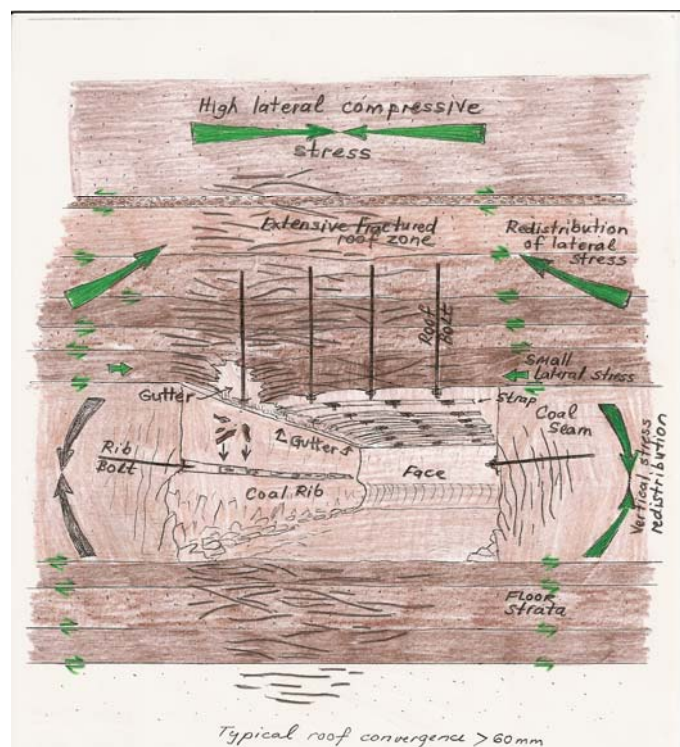
Model of 45° Tensile Test

Experimental Data vs Numerical Model

Figure 4 - Numerical Modelling of "Diamond" Tensile Test

ROCK FRACTURE MECHANISMS IN UNDERGROUND ROADWAYS

Mining induced fractures occur ahead of the roadway face where the stresses are high. These fractures gradually grow, forming a typical fractured roof as illustrated in Figure 5. The fractures develop in response to the elevated compressive stresses that intercept the stress relief towards the mine opening. In other words, while stresses may concentrate across the mine opening, reduction in stress occurs towards the mine opening. Bending of bedded strata that typically occurs in the vicinity of mine openings will result in failure along the weak bedding planes. It is impossible to prevent the development of mining induced fractures, however it is possible to minimise their displacement.

**Figure 5 - Typical roof conditions in a mine roadway requiring steel mesh application**

In strain softened strata, gradual displacements reduce the virgin compressive stress until equilibrium is reached where the remnant compressive stresses that remain within the strata provide enough confinement to arrest any further movement along the fractures. If the strata are severely broken and the confining stresses are totally dissipated, strata will lose self supporting capability and disintegrate. In particular, large displacements tend to occur when the fractured rock mass is stress free. For this

reason even a small confining stress may be enough to arrest significant rock displacements and falls of "loose" rock material. Just prior to a fall, accelerated fracture displacements and fragment rotations may occur that will "unlock" the rock structure and eventually cause the yielded rock zone to fall. Inadequately supported roadway skin can slowly deteriorate and affect the ground stability between the bolts as illustrated in Figure 6. Ideally, it is the function of the reinforcing members, such as the rock bolt together with the appropriate skin reinforcement, to prevent the last phase of rock de-fragmentation and thus improve integrity of the fractured rock zones.

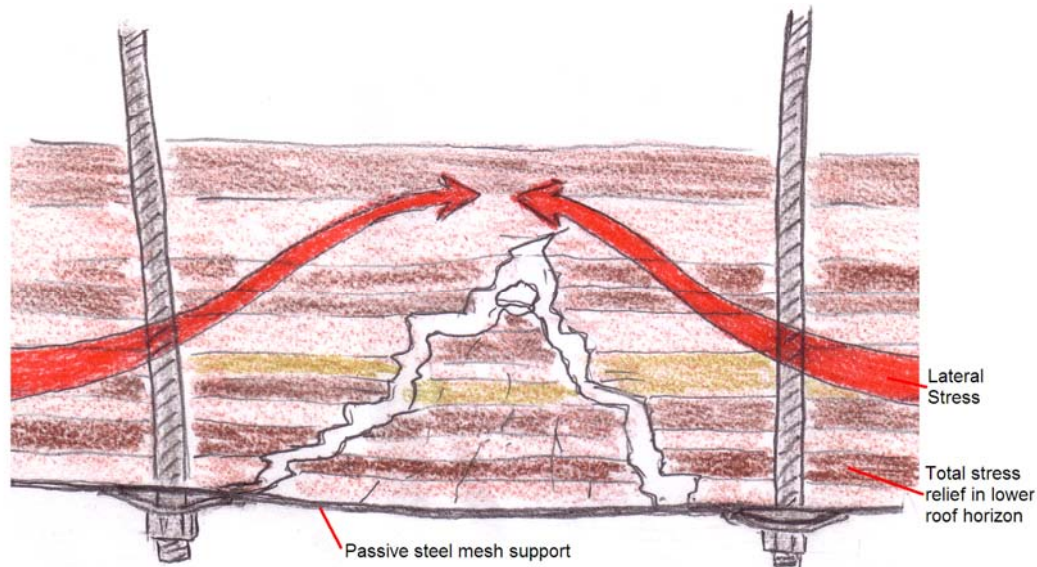


Figure 6 - Partial Roof Failure Between Bolts Affecting Roof Stability

COMPARISON OF THE POLYMER SKIN REINFORCEMENT AND STEEL MESH SUPPORT

The fundamental difference between the polymer skin and the steel mesh support is similar to the difference between a point anchor and a fully encapsulated bolt system. A fully encapsulated bolt provides immediate resistance to any fracture movement via its continuous anchorage to the surrounding strata while the point anchor bolt needs to stretch significantly over its entire length to provide comparable strata confinement. In a similar manner the reinforced polymer skin bonded to the strata surface provides immediate resistance to any crack movement that occurs at the strata surface while the steel mesh will support the strata only after significant roof deformation occurs. The reinforced polymer skin adhesion to the strata provides an additional active reinforcing mechanism to compliment the fully encapsulated bolt reinforcement system and contribute to the overall stability of strata adjacent to the mine roadway.

To investigate the polymer skin reinforcing capabilities, several numerical models were constructed to test the roof response to displacement of fractures in the failed roof. The models were designed to simulate fracture behaviour with dilation and displacement parallel and perpendicular to the fractures. The numerical simulations were done with and without the polymer skin reinforcement.

During the fracture movement two distinct displacement mechanisms occur:

- (i) fracture displacement along its length (shearing); and
- (ii) fracture dilation perpendicular to the fracture plane due to fracture irregularities.

The mechanism of fracture movement can be seen in Figure 7 with combined fracture displacements parallel to the fracture plane and the influence of the fracture asperities on fracture dilation. Other fracture opening mechanisms are also common during strata bending or tension.

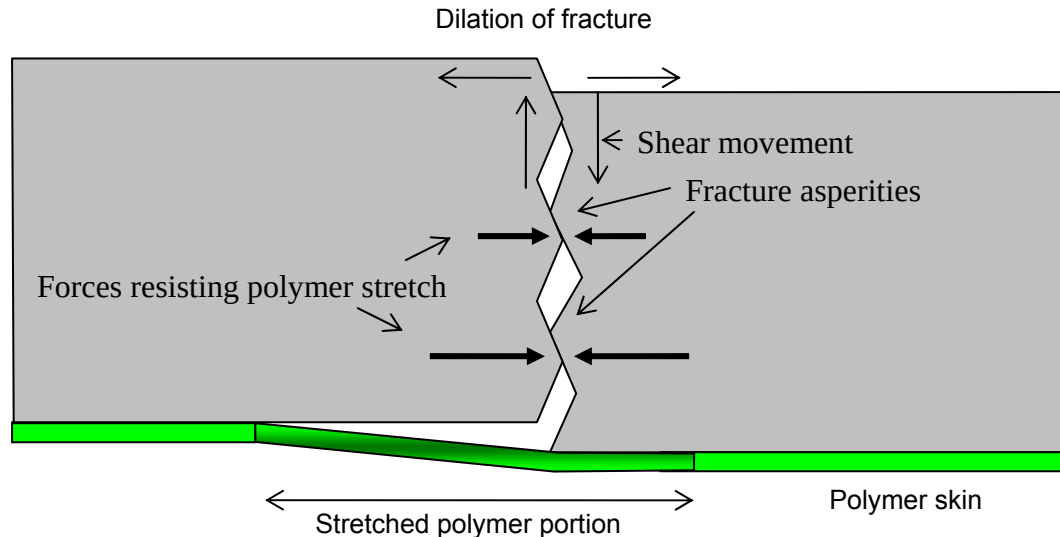


Figure 7 - Shearing and dilation mechanisms of fracture surface

To quantify the influence of both fracture shearing and dilation mechanisms and study the response of polymer skin to these movements, fracture dilation was studied separately to fracture shearing. In the first model shown in Figure 8 the fracture at the rock skin surface was parted perpendicular to the surface with the polymer bond to imitate the effect of fracture dilation (without fracture shearing) that usually occurs during shearing along the fracture plane. Typical fracture dilation is in the order of a few millimetres depending upon the roughness of the fracture surface and the fracture displacement. The numerical model was set up to simulate dilation of a single fracture oriented perpendicular to the polymer layer and the stress response within the strata was studied. The polymer skin was bonded to the rock surface and fracture dilation simulated. In response to the fracture dilation the polymer skin de-bonded in the immediate vicinity of the crack but the strong adhesion to the rock kept the polymer skin anchored a short distance from the crack opening. Opposing the fracture opening, the stretched polymer induced a compressive stress within the rock. The test can be observed in Figure 9.

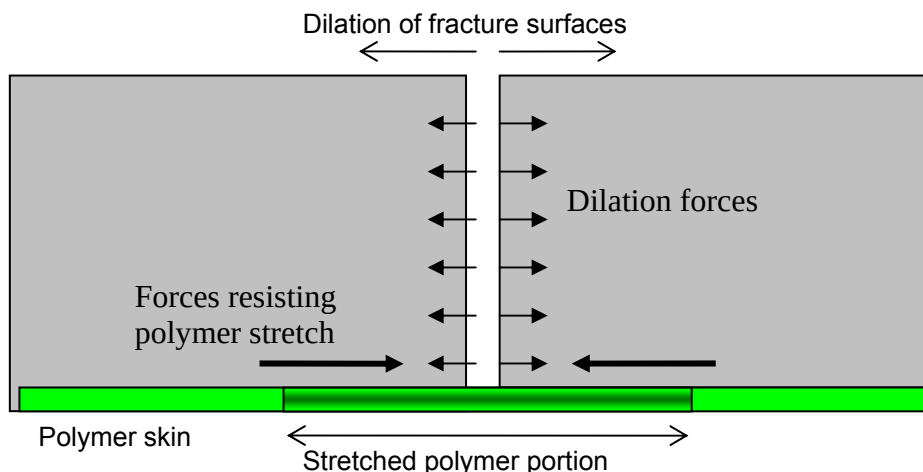


Figure 8 - Dilation mechanism parting the fracture surface

The stress contours shown in Figure 9 clearly show a significant compressive stress normal to the fracture that occurred in response to 2mm of fracture dilation.

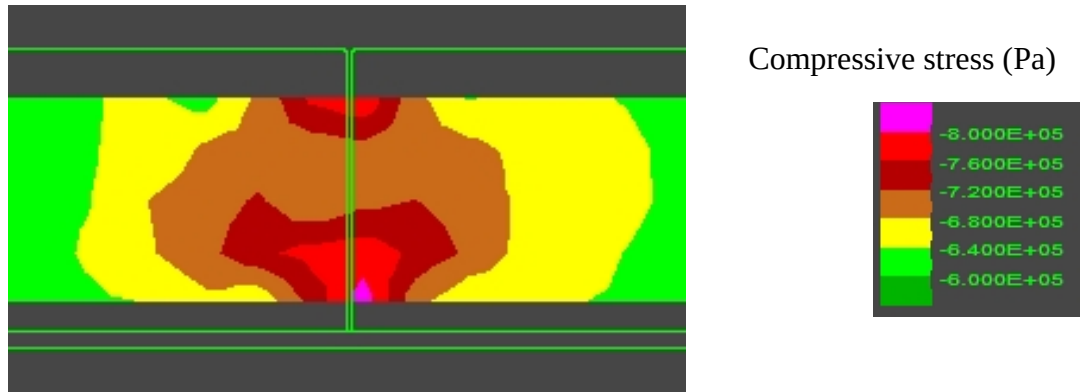


Figure 9 - Effect of fracture dilation on lateral stress across the fractured surface

The magnitudes of stress that develop within the lower roof depend on:

- (i) fracture displacement,
- (ii) polymer bonding properties,
- (iii) polymer stiffness, and
- (iv) polymer thickness.

Further research is underway to quantify the de-bonding length for various values of polymer adhesion and thus determine the total normal fracture loads for a given fracture displacement.

In the second model (Figure 10) a completely smooth vertical fracture was gradually displaced and the polymer skin response observed. During the vertical shearing, strata movement parted the polymer skin away from the rock surface, de-bonding a short section of the polymer/rock interface. The polymer was thus forced to stretch across the fracture as the distance from the bonded anchorage on each side increased. The reaction force at the skin anchorage induced a small compressive stress within the rock across the fracture area, as shown by contours in Figure 11. It is the fracture dilation that in this case produced larger stress across the fractured surface. It must be pointed out that the fracture shearing and dilation are not separate events but occur together and therefore the polymer induced compressive stresses across the fracture shown in Figures 9 and 11 would combine together.

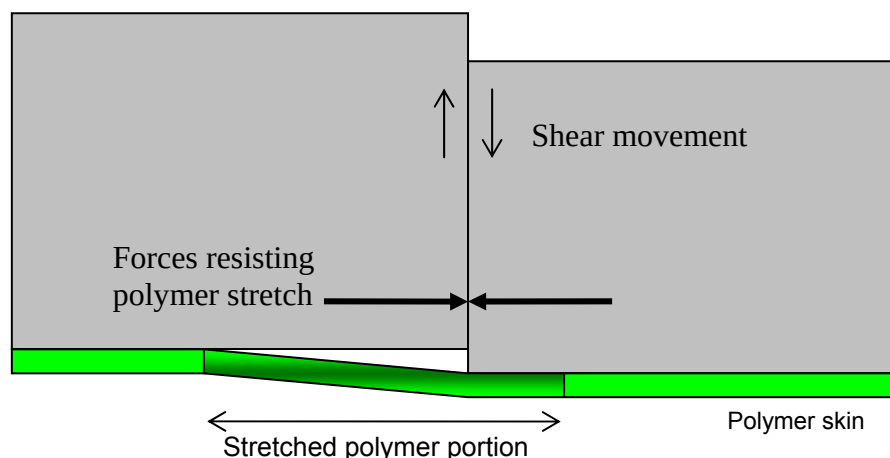


Figure 10 - Shearing mechanism along the fracture surface

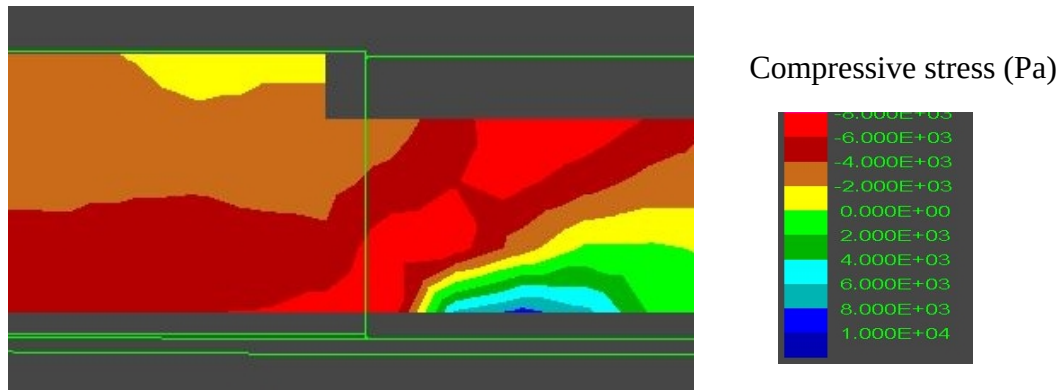


Figure 11 - Effect of fracture shearing on lateral stress across the fractured surface

CONCLUSION

The above investigations indicate that the reinforced polymer may provide a strata skin reinforcement system superior to the currently used steel mesh. Benefits of the polymer skin can come from the ability to adhere well to rock/coal surfaces and provide resistance to strata displacements and fracture opening. The adhesion is not negligible and would have a positive influence on the overall roof support. The reinforced polymer skin is fundamentally a different type of support to the passive steel mesh, providing active resistance to any movement as soon as movement begins to occur. The partial de-bonding of the polymer from the rock surface during fracture movement is unavoidable and may not significantly disturb the polymer reinforcing capabilities. Severe roof movement that may occur in heavily loaded roadways may eventually de-bond the polymer from the fractured rock mass. However, the tough nature of the polymer mesh will further resist the severe strata displacements in a manner similar to that of steel mesh, while the polymer fibre yielding mechanism will give an audible warning reminiscent of the sound made during yield of the old wooden prop system.

Further benefits of the polymer skin include automated application where continuous or intermittent applications of polymer skin of various thickness and patterns are possible. The polymer skin can be applied on the roof and rib strata close to the working face or as required. The fully automated fast setting polymer application can be incorporated together with the automated bolting system with the aim to speed up roadway development and remove mine personnel from the working face area.

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A NEW APPROACH FOR DETERMINATION OF TUNNEL SUPPORTING SYSTEM USING ANALYTICAL HIERARCHY PROCESS (AHP)

Kazem Oraee¹, Navid Hosseini², Mehran Gholinejad³

ABSTRACT: In underground mining, the selection of support system for mine tunnel development plays a significant role in safety and economics of operations. Traditionally, such selection is on the basis of the experience of the design engineer. Nevertheless, the validity of such selection is questionable. A new approach for selecting the optimum tunnel support system based on Analytical Hierarchy Process (AHP) is proposed. In this new approach, the selection of the tunnel support system is considered as a multi criteria decision-making problem. Firstly, by using the numerical Finite Difference Method (FDM), based on technical and stability parameters of the tunnel, different support systems are specified. Then, by considering the economics and performance indices of each support system, a decision tree based on AHP model is generated and the optimum support system is selected. As a field study, the method is applied to Tabas collieries in Iran. It is concluded that the proposed support system determination is advantageous compared to other alternatives. Therefore, the proposed approach can assist the engineer in selection of optimum tunnel support system in different underground mining situations.

INTRODUCTION

In many cases, the support system for tunnel construction is selected based on the experience of the design engineer, hence personal judgment is often the main basis in stead of intellectual and scientific criteria (Oraee K., 2001). There are various support systems for any particular situation but the selection of the optimum system depends on many technical and economical parameters.

The main aim of this study is to select proper support system by using the AHP model. As a field study, this technique was applied to tunnel C1, which is one of the main entries in various Tabas coalfield mines, which is located in the central part of Iran. Due to being a large reserve and of regular geometry of the coal seams, coal is mined by mechanized longwall mining. This method requires excavation of several tunnels; some must be stable for long time and even during entire life of the mine. Therefore, the selection of appropriate tunnel support system is an important aspect of tunnel construction.

GEOMECHANICAL PROPERTIES OF TABAS COAL FIELD

The validity of numerical modelling and the results of model analysis are dependent on the determination of the geomechanical parameters of surrounding rock mass. Hence, by field studies and existing technical reports (Hosseini Navid, 2008) the geomechanical properties of tunnel rock mass is collected. Based on laboratory and field data, the uniaxial compressive strength of surrounding rock mass of the tunnel is 10.7 MPa, and the obtained tensile strength, based on Brazilian test, is 1.3 MPa. The Young's modulus and Poisson's ratio are 4385 MPa and 0.25, respectively. Based on the triaxial compressive test results, the resultant friction angle is 35 degree and cohesion of rock mass is 5 MPa. The geomechanical parameters of rock mass are shown in Table 1.

Table 1: Geomechanical parameters of rock mass (Hosseini Navid, 2008)

σ_c	σ_t	E	ν	ϕ	c
10.7 MPa	1.3 MPa	4385 MPa	0.25	35 Deg.	5 MPa

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MODELING THE BEHAVIOR PARAMETERS OF SUPPORT SYSTEM

In this study, the states of various support systems (Hosseini Navid, 2008; Oraee K., 2005) were analysed by using numerical modelling. The defaulted mechanical properties of each support system such as Young's modulus, Poisson's ratio, bulk modulus and rigidity modulus for steel sets and rock bolts were defined according to relevant standards. In addition, the Young's modulus of shotcrete can be calculated by equation 1.

$$E = A \cdot \sigma_b^n \quad (1)$$

Where E is the Young's modulus of shotcrete (MPa), σ_b is the concrete uniaxial compressive strength (MPa), and A and n are statistical constants, which depend on the cementation degree and compressive strength characteristics. Based on engineering judgment and long-term of the tunnel life (Oraee K., 2005) A and n are selected as 6500 and 0.5 respectively.

THE STATE OF IN-SITU STRESSES

In-situ stresses are one of important effective parameters on the state of tunnel support system. For the analysis of the support system state in numerical model the magnitude and direction of in-situ stresses in modelling must be defined. The in-situ stresses are calculated by using equation 2 and 3.

$$\sigma_v = \gamma \cdot h \quad (2)$$

$$\sigma_H = \frac{\nu}{1 - \nu} \sigma_v \quad (3)$$

Where σ_v and σ_H are the vertical and horizontal in-situ stress, respectively, γ is the average density of overlying strata, h is the depth below ground surface and ν is the Poisson's ratio. Based on existing data (Hosseini Navid, 2008), the vertical and horizontal in-situ stresses are 17.4 MPa and 5.1 MPa, respectively.

NUMERICAL MODELING AND SUPPORT SYSTEM SELECTION

In this study, FLAC^{3D} software (Itasca, 2002) was used for modelling. FLAC^{3D} is a numerical code, based on three-dimensional finite difference method, which has comprehensive usage in rock mechanics engineering study.

For modelling, the tunnel geometry was defined in FLAC^{3D} software as the first step. Then, the geomechanical properties of tunnel surrounding rock mass like, Young's modulus, Poisson's ratio, bulk modulus, rigidity modulus and also compressive strength, tensile strength, internal friction angle, density and cohesion were input in the model. Consequently, the behaviour of tunnel surrounding rock mass by FLAC^{3D} was analysed and the potential of failure and displacement was calculated. In the next stage, the various support systems were applied and the mechanical state of the tunnel after applying each support system was determined. Based on this modelling, the proper support system was selected from technical viewpoint. The grid model of FLAC^{3D} is shown in Figure 1.

To analyse the support system capability, four critical points were selected according to Figure 2. As can be seen in the figure, point 1 is at tunnel roof; point 2 at tunnel floor and points 3 and 4 are locating at the wall and floor intersections of, horizontal and vertical directions, respectively.

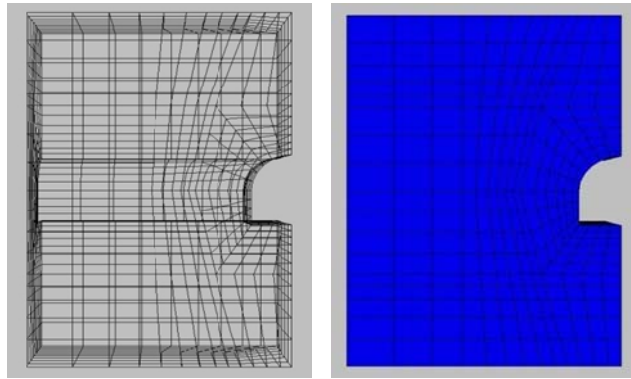


Figure 1 - Grid 3D model of tunnel

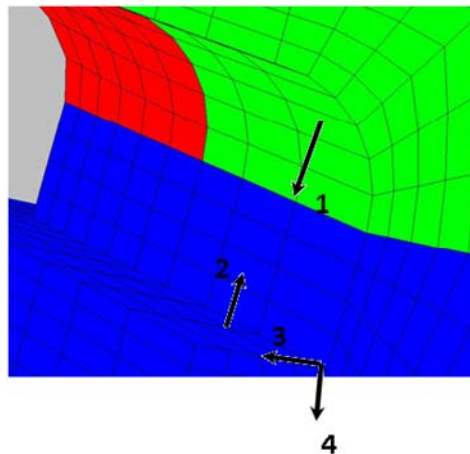


Figure 2 - Selected critical points at tunnel

In total 10 different support systems (Oraee K., 2005) were applied in model and the stability state for each support system was analysed as presented in Table 2.

Table 2 - Studied support systems with index

No.	Support system explanation	Index
1	Supporting by B40 shotcrete 5 cm in thickness	A
2	Supporting by B40 shotcrete 8 cm in thickness	B
3	Supporting by B40 shotcrete 8 cm in thickness together with rockbolt	C
4	Application of roof piping together cement injection	D
5	Application of rockbolt to the gallery roof and sides	E
6	Application of steel arches with 1m spacing	F
7	Application of steel arches with 0.5 m spacing	G
8	Supporting by B50 shotcrete 5 cm in thickness	H
9	Supporting by B50 shotcrete 8 cm in thickness	I
10	Application of steel arches with 1 m spacing together with rockbolt	J

After the applying the support system, the displacement state of tunnel's surrounding rock mass, was determined at the four points as depicted in Figure 2. The maximum stress at the tunnel surrounding was also estimated. Based on the above results and the maximum pressure of support system, the safety factor for each support system was calculated. Displacement at these four points and safety factor are shown in Table 3.

Table 3 - Results of numerical model

Model index	Displacement at point (cm)				The maximum stress on tunnel circumference (MPa)	Safety factor
	1	2	3	4		
A	11.51	26.82	12.25	4.03	36.44	1.04
B	8.92	24.00	11.03	2.08	29.93	1.47
C	1.89	3.72	1.33	0.50	24.75	2.32
D	2.10	3.92	1.02	0.43	23.73	2.44
E	10.30	23.36	8.19	5.11	29.47	1.15
F	4.14	6.35	4.12	3.19	22.82	1.79
G	2.81	3.63	1.30	0.61	25.70	2.13
H	10.62	25.11	11.83	3.29	35.61	1.25
I	8.13	23.91	10.09	2.01	30.04	1.59
J	3.50	4.01	2.61	0.82	25.11	2.28

As, the minimum acceptable factor of safety is two (2), for this result, the four support systems of C, D, G and J are the only accepted from technical viewpoint. Therefore, the final optimum support system will be selected from one of them.

Consequently, based on experience and viewpoints of expert engineers the decision criterions should be determined for the selection of the proper support system (Yavuz M., Iphar M., Once G., 2007). The considered decision criterions for the selection of the proper support system are given in Table 4.

Table 4 - Considered decision criterions for proper support system selection

No.	Criteria explanation	Index
1	The vertical displacement at point 1	C1
2	The vertical displacement at point 2	C2
3	The vertical displacement at point 3	C3
4	The horizontal displacement at point 3	C4
5	The support system costs	C5
6	The support system performance	C6
7	Safety factor	C7

THE HIERARCHY DESIGN

After determination of goal, options and criterions the hierarchy tree of AHP model (Saaty T.L., 1980) were designed. placed in the first level of hierarchy, is the goal, which is support system selection. In second level, is criterions and in the third level, options are arranged. The hierarchy designed for this study is presented in Figure 3.

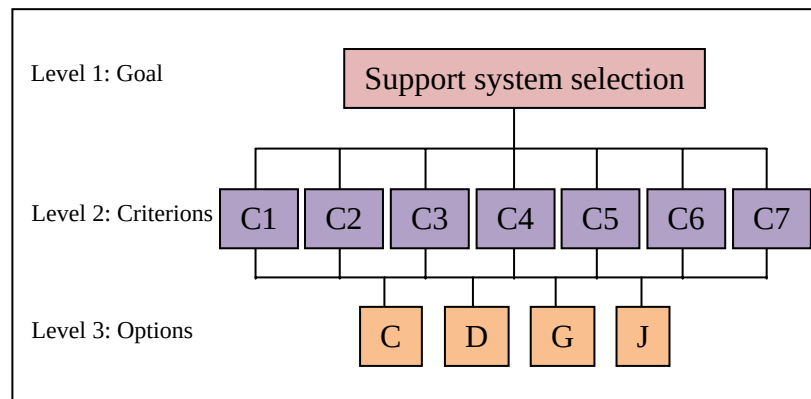


Figure 3 - Hierarchy designed for proper support system selection

Among the ten support systems studied, four were technically acceptable and were arranged in level 3, as options.

Generally, in AHP model the elements of each level with its respective element in above level are compared as pair-wise, and therefore the local priority are calculated. Then, with assimilating the local priorities, overall priority was calculated. As expressed, in AHP model all comparisons were pair-wise and based on oral judgments that expressed by Preference values as mentioned in Table 5.

In this stage, the relative weight of each support system must be determined by the mathematical mean method (Amirafshari M., Qolinejad Mehran., Hosseini Navid, 2005). The pair-wise comparison of acceptable support systems (C, D, G and J), the summation weights matrix, and the average weights matrix of each rows, all based on criterion C1 are shown in Tables 6, 7 and 8, respectively.

Table 5 - Preference values for pair-wise comparison (Saaty T.L., 1980)

Oral judgments	Numeral value
Extremely preferred	9
Very strongly preferred	7
Strongly preferred	5
Moderately preferred	3
Equally preferred	1
Intermediate values	2, 4, 6 and 8

Table 6 - The pair-wise comparison of support system based on criterion C1

	C	D	G	J
C	1.00	3.00	5.00	7.00
D	0.33	1.00	4.00	6.00
G	0.20	0.25	1.00	5.00
J	0.14	0.17	0.20	1.00

Table 7 - Summation weights matrix based on criterion C1

	C	D	G	J
C	1.00	3.00	5.00	7.00
D	0.33	1.00	4.00	6.00
G	0.20	0.25	1.00	5.00
J	0.14	0.17	0.20	1.00
Summation of each columns	1.68	4.42	10.20	19.00

Table 8 - Average weights matrix of each rows based on criterion C1

	C	D	G	J	The average of each rows
C	0.60	0.68	0.49	0.37	0.53
D	0.20	0.23	0.39	0.32	0.28
G	0.12	0.06	0.10	0.26	0.13
J	0.09	0.04	0.02	0.05	0.05
Summation of each columns	1.00	1.00	1.00	1.00	1.00

Similarly, for criteria C2 to C7 the pair-wise comparison of support system were constructed, then the summation weights matrix and also the average weights matrix of each rows based on criterion C2 to C7 were calculated as shown in Tables 9 to 26, respectively.

Table 9 - The pair-wise comparison of support system based on criterion C2

	C	D	G	J
C	1.00	0.50	0.20	0.14
D	2.00	1.00	0.33	0.50
G	5.00	3.00	1.00	0.13
J	7.00	2.00	8.00	1.00

Table 10 - Summation weights matrix based on criterion C2

	C	D	G	J
C	1.00	0.50	0.20	0.14
D	2.00	1.00	0.33	0.50
G	5.00	3.00	1.00	0.13
J	7.00	2.00	8.00	1.00
Summation of each columns	15.00	6.50	9.53	1.77

Table 11 - Average weights matrix of each rows based on criterion C2

	C	D	G	J	The average of each rows
C	0.07	0.08	0.02	0.08	0.06
D	0.13	0.15	0.03	0.28	0.15
G	0.33	0.46	0.10	0.07	0.24
J	0.47	0.31	0.84	0.57	0.54
Summation of each columns	1.00	1.00	1.00	1.00	1.00

Table 12 - The pair-wise comparison of support system based on criterion C3

	C	D	G	J
C	1.00	3.00	2.00	0.50
D	0.33	1.00	0.50	0.17
G	0.50	2.00	1.00	0.25
J	2.00	6.00	4.00	1.00

Table 13 - Summation weights matrix based on criterion C3

	C	D	G	J
C	1.00	3.00	2.00	0.50
D	0.33	1.00	0.50	0.17
G	0.50	2.00	1.00	0.25
J	2.00	6.00	4.00	1.00
Summation of each columns	3.83	12.00	7.50	1.92

Table 14 - Average weights matrix of each rows based on criterion C3

	C	D	G	J	The average of each rows
C	0.26	0.25	0.27	0.26	0.26
D	0.09	0.08	0.07	0.09	0.08
G	0.13	0.17	0.13	0.13	0.14
J	0.52	0.50	0.53	0.52	0.52
Summation of each columns	1.00	1.00	1.00	1.00	1.00

Table 15 - The pair-wise comparison of support system based on criterion C4

	C	D	G	J
C	1.00	3.00	0.50	0.25
D	0.33	1.00	0.20	0.13
G	2.00	5.00	1.00	0.33
J	4.00	8.00	3.00	1.00

Table 16 - Summation weights matrix based on criterion C4

	C	D	G	J
C	1.00	3.00	0.50	0.25
D	0.33	1.00	0.20	0.13
G	2.00	5.00	1.00	0.33
J	4.00	8.00	3.00	1.00
Summation of each columns	7.33	17.00	4.70	1.71

Table 17 - Average weights matrix of each rows based on criterion C4

	C	D	G	J	The average of each rows
C	0.14	0.18	0.11	0.15	0.14
D	0.05	0.06	0.04	0.07	0.06
G	0.27	0.29	0.21	0.20	0.24
J	0.55	0.47	0.64	0.59	0.56
Summation of each columns	1.00	1.00	1.00	1.00	1.00

Table 18 - The pair-wise comparison of support system based on criterion C5

	C	D	G	J
C	1.00	0.17	0.25	3.00
D	6.00	1.00	2.00	9.00
G	4.00	0.50	1.00	5.00
J	0.33	0.11	0.20	1.00

Table 19 - Summation weights matrix based on criterion C5

	C	D	G	J
C	1.00	0.17	0.25	3.00
D	6.00	1.00	2.00	9.00
G	4.00	0.50	1.00	5.00
J	0.33	0.11	0.20	1.00
Summation of each columns	11.33	1.78	3.45	18.00

Table 20 - Average weights matrix of each rows based on criterion C5

	C	D	G	J	The average of each rows
C	0.09	0.09	0.07	0.17	0.11
D	0.53	0.56	0.58	0.50	0.54
G	0.35	0.28	0.29	0.28	0.30
J	0.03	0.06	0.06	0.06	0.05
Summation of each columns	1.00	1.00	1.00	1.00	1.00

Table 21 - The pair-wise comparison of support system based on criterion C6

	C	D	G	J
C	1.00	0.50	5.00	3.00
D	2.00	1.00	8.00	6.00
G	0.20	0.13	1.00	0.33
J	0.33	0.17	3.00	1.00

Table 22 - Summation weights matrix based on criterion C6

	C	D	G	J
C	1.00	0.50	5.00	3.00
D	2.00	1.00	8.00	6.00
G	0.20	0.13	1.00	0.33
J	0.33	0.17	3.00	1.00
Summation of each columns	3.53	1.79	17.00	10.33

Table 23 - Average weights matrix of each rows based on criterion C6

	C	D	G	J	The average of each rows
C	0.28	0.28	0.29	0.29	0.29
D	0.57	0.56	0.47	0.58	0.54
G	0.06	0.07	0.06	0.03	0.05
J	0.09	0.09	0.18	0.10	0.12
Summation of each columns	1.00	1.00	1.00	1.00	1.00

Table 24 - The pair-wise comparison of support system based on criterion C7

	C	D	G	J
C	1.00	2.00	0.25	0.33
D	0.50	1.00	0.17	0.25
G	4.00	6.00	1.00	3.00
J	3.00	4.00	0.33	1.00

Table 25 - Summation weights matrix based on criterion C7

	C	D	G	J
C	1.00	2.00	0.25	0.33
D	0.50	1.00	0.17	0.25
G	4.00	6.00	1.00	3.00
J	3.00	4.00	0.33	1.00
Summation of each columns	8.50	13.00	1.75	4.58

Table 26 - Average weights matrix of each rows based on criterion C7

	C	D	G	J	The average of each rows
C	0.12	0.15	0.14	0.07	0.12
D	0.06	0.08	0.10	0.05	0.07
G	0.47	0.46	0.57	0.65	0.54
J	0.35	0.31	0.19	0.22	0.27
Summation of each columns	1.00	1.00	1.00	1.00	1.00

In fact, the last columns in average weights matrix tables (Tables 8, 11, 14, 17, 20, 23 and 26) shows the relative directions vector for support system options based on criterions. Therefore, the weights of support system options relative to criterions are shown in Table 27.

Table 27 - Weights of support system options relation to criterion

	C1	C2	C3	C4	C5	C6	C7
C	0.53	0.06	0.26	0.14	0.11	0.29	0.12
D	0.28	0.15	0.08	0.06	0.54	0.54	0.07
G	0.13	0.24	0.14	0.24	0.30	0.05	0.54
J	0.05	0.54	0.52	0.56	0.05	0.12	0.27

After determination of support system options weight relative to criterions, consideration was given next to the pair-wise comparison of criterions. In other word, the role and share of each criterion in the selection of the proper support system should be understood. For this purpose, the criterions must be compared as pair-wise. The results are given in Table 28.

Table 28 - The pair-wise comparison of criterions relative to itself

	C1	C2	C3	C4	C5	C6	C7
C1	1.00	0.50	0.33	0.20	2.00	2.00	4.00
C2	2.00	1.00	0.50	0.25	5.00	3.00	3.00
C3	3.00	2.00	1.00	0.33	7.00	3.00	5.00
C4	5.00	4.00	3.00	1.00	8.00	5.00	9.00
C5	0.50	0.20	0.14	0.13	1.00	0.50	2.00
C6	0.50	0.33	0.33	0.20	2.00	1.00	3.00
C7	0.25	0.33	0.20	0.11	0.50	0.33	1.00

Similarly, the summation weights matrix and the average weights matrix were calculated and the results are shown in Tables 29 and 30, respectively.

Table 29 - Summation weights matrix of criterions

	C1	C2	C3	C4	C5	C6	C7
C1	1.00	0.50	0.33	0.20	2.00	2.00	4.00
C2	2.00	1.00	0.50	0.25	5.00	3.00	3.00
C3	3.00	2.00	1.00	0.33	7.00	3.00	5.00
C4	5.00	4.00	3.00	1.00	8.00	5.00	9.00
C5	0.50	0.20	0.14	0.13	1.00	0.50	2.00
C6	0.50	0.33	0.33	0.20	2.00	1.00	3.00
C7	0.25	0.33	0.20	0.11	0.50	0.33	1.00
Summation of each columns	12.25	8.37	5.51	2.22	25.50	14.83	27.00

Table 30 - Average weights matrix of each row

	C1	C2	C3	C4	C5	C6	C7	The average of each rows
C1	0.08	0.06	0.06	0.09	0.08	0.13	0.15	0.09
C2	0.16	0.12	0.09	0.11	0.20	0.20	0.11	0.14
C3	0.24	0.24	0.18	0.15	0.27	0.20	0.19	0.21
C4	0.41	0.48	0.54	0.45	0.31	0.34	0.33	0.41
C5	0.04	0.02	0.03	0.06	0.04	0.03	0.07	0.04
C6	0.04	0.04	0.06	0.09	0.08	0.07	0.11	0.07
C7	0.02	0.04	0.04	0.05	0.02	0.02	0.04	0.03
Summation of each columns	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00

In fact, the last column of Table 30 is the total utilization vector, as follow:

$$(0.09, 0.14, 0.21, 0.41, 0.04, 0.07, 0.03) \quad (4)$$

By multiplying this vector in matrix of Table 27, the final weight of each support system options were obtain. Therefore:

The weight of C:

$$(0.53 \times 0.09) + (0.06 \times 0.14) + (0.26 \times 0.21) + (0.14 \times 0.41) + (0.11 \times 0.04) + (0.29 \times 0.07) + (0.12 \times 0.03) = 0.1964$$

The weight of D:

$$(0.28 \times 0.09) + (0.15 \times 0.14) + (0.08 \times 0.21) + (0.06 \times 0.41) + (0.54 \times 0.04) + (0.54 \times 0.07) + (0.07 \times 0.03) = 0.1491$$

The weight of G:

$$(0.13 \times 0.09) + (0.24 \times 0.14) + (0.14 \times 0.21) + (0.24 \times 0.41) + (0.30 \times 0.04) + (0.05 \times 0.07) + (0.54 \times 0.03) = 0.2048$$

The weight of J:

$$(0.05 \times 0.09) + (0.54 \times 0.14) + (0.52 \times 0.21) + (0.56 \times 0.41) + (0.05 \times 0.04) + (0.12 \times 0.07) + (0.27 \times 0.03) = 0.4374$$

By arranging the final weight of support system options, the preferences of support systems options are demonstrated in Table 31.

Table 31 - Preference of support system options

preference	Support system	Final weight
1	J	0.4374
2	G	0.2048
3	C	0.1964
4	D	0.1491

Based on the result of AHP model, option J, i.e., application of steel arches with 1 m spacing together with rockbolt is the most preference option. In sequence, option G (application of steel arches with 0.5 m spacing), option C (supporting by B40 shotcrete 8 cm in thickness together with rockbolt), and option D (application of roof piping together cement injection), are the next preference, respectively.

CONCLUSIONS

This study shows that the AHP model is an adequate technique for selection of tunnel support system. Usually the system selection based on experience with consideration of the many decision criterions not only is confusing task, but also the share of each criterion in final selection is not well understood. However, the organization problem based on AHP model can result to valuable decision criterion. In this study, among the technical viewpoint acceptable options, the option J, i.e., application of steel arches with 1 m spacing together with rockbolt was selected. Combined of steel arches with rockbolt provides the stability and reduce the costs of support system as well. Of course, before the usage of the AHP model, for safety factor determination, the support system should be analysed based on empirical, analytical or numerical conventional methods. However, based on this case study, it is concluded that the combination of the numerical model for determination of safety factor and the AHP model for preferential, is a suitable approach in selection of main tunnel support system.

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GATE ROAD DEVELOPMENT IN HIGH GAS CONTENT COAL SEAMS AT KARAGANDA BASIN COAL MINES, KAZAKHSTAN

Dzhakan Baimukhametov¹, Aleksandr Polchin², Tanat Dauov² and Sergey Ogay²

ABSTRACT: Most coal seams, currently mined in the Karaganda Coal Basin, are prone to outburst. The main thick D6 coal seam is considered as most prone to outburst risk. Trials of advance degassing from the surface have not given positive results because of low permeability. 100mm diameter in-seam holes are subsequently drilled in almost all longwall blocks to facilitate preliminary degassing of the coal seams. Gas extraction quantities are however low, even the holes are placed on suction. To facilitate the gas release during longwall block development, of the main seam, a method of development below the seam was used. This gave rise to increase in permeability of overlying thick seam with high gas content, achieving a local degasification of the overlying seam by up to 90%.

The initial development roadway was driven under the seam, in rock, at a distance of between 8-12 m from seam floor in the same contour of the future development roadway in the coal seam. A relief area was created as a result of stress redistribution above the roadway. Degassing holes were drilled from the rock development, into the seam area of the future coal seam development heading.

During traditional in seam development, outburst preventive measures were taken, which increased the labour intensiveness of development working, sharply decreasing the development rate. Application of these new techniques allowed increased development rates in seams to be realised, from 25-40 m per month to 120-150 m in the outburst prone areas.

INTRODUCTION

There are eight underground coal mines in JSC "ArcelorMittal Temirtau" Coal Division. Coal production is 12 Mt per annum. Most of the mines are operated at a depth of more than 500 m, the thickness of each mineable coal seam is more than 3 m and gas content from 18 to 24 m³/tonne. All coal seams are deemed to fall into the dangerous category with regard to coal and gas outburst, and D6 coal seam, which is 4-6.5 m thick, is considered most prone from outburst risk. The D6 seam is mined in two lifts, the top section being mined first and then the gas free bottom section mined 18 months later.

PRE DRAINAGE METHODS

In order to reduce gas emission from the face as it is being mined; pre drainage was used in almost all blocks by means of 100 mm diameter in-seam holes drilled to the depth 150-170 m, parallel to production face. Hole spacing varied from 4 to 8 m. The decrease in coal seam gas content reached up to 20-25% after a year of degassing. Trials of advance degassing from the surface did not give positive results due to low permeability of coal seams ($3-5 \times 10^{-2}$ md).

Lower seam extraction

The method of extracting the underlying seam is used to increase gas permeability of the overlying thick high gas content target seam. This achieved the degassing of the overlying seam by up to 90%. Unfortunately for the D6 seam non sequential extraction was not possible.

Goaf gas extraction

Goaf degassing to the depth of 500 m from the surface was carried out by drilling of vertical wells from the surface, through at least two goaf areas. However, the difficulties with drilled holes due to depth

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increase greater than 500 m, were such, that mine management opted for a system of road headings of special gas-drainage "sewers" driven above the seam at a distance of 20-30 m.

Ventilation system

Z-type ventilation system is mostly used for face ventilation. Z system prevents dangerous methane concentration near the production face. Typical scheme of goaf ventilation and degassing is shown in Figure 1.

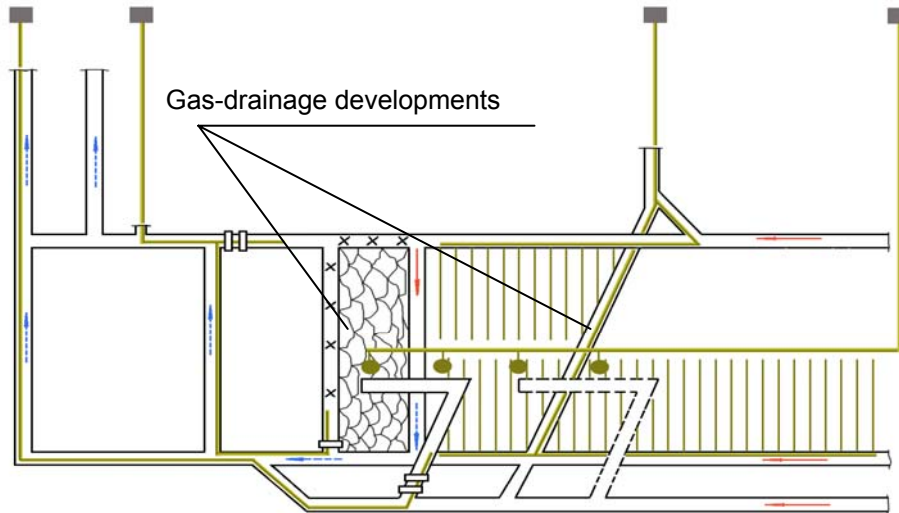


Figure 1 - Z-type ventilation scheme at Shakhtinskaya mine

The Z type ventilation proved to be costly in terms of heading support maintenance and longwall face ventilation management. Accordingly the system was changed to U-type ventilation system which enabled better longwall face operation, effective gas drainage management, and improvement of headings support systems, by using rock bolting, in preference to the traditional steel arches (Figure 2).

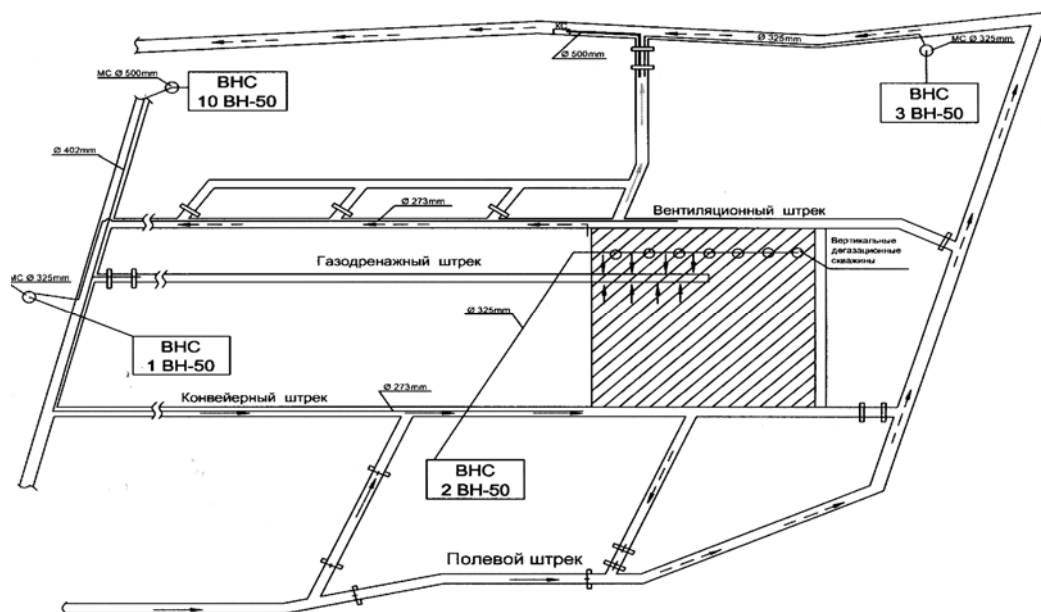


Figure 2 - U-type ventilation scheme at Abayskaya mine

OUTBURST CONDITIONS

One of the restrictive factors of timely development of the longwall production areas was the low drivage rate of mine workings (25-40 m/month) in D6 seam, which is most prone to outburst risk. Low rate of development was due to the numerous outburst preventive measures in place.

Thick coal seam D6 is considered as most prone to outburst risk, particularly in Lenina and Kazakhstanskaya mines. Sudden coal and gas outbursts which took place at Lenina mine in 1995 and 1998 were considered to be the most powerful in coal mining throughout the world. The amount of pulverised coal and the volume of methane gas ejected in 1995 were estimated at about 640 t and 550,000 m³ respectively. However, in 1998 outburst, 3250 t of coal and 1.3 Mm³ of methane were ejected. Since then, a set of outburst preventing measures have been introduced during the development and subsequent operation at D6 coal seam. These measures are based on (dry drilling) drilling relief holes ahead of the face.

The introduction of preventive measures caused labour practices which affected the rate of heading development in the upper level of D6 coal seam.

The thick D6 seam has a Lower layer of very friable coal. The relief holes were quickly filled with drill cuttings and often the volume of coal fines released from the holes, can reach up to several cubic meters. However, on a number of occasions these typical pockets of soft coal and accumulated gas resulted in sudden coal and gas outbursts.

Despite of the high gas content, the soft coal zone has very low gas permeability. This low permeability was proven in the process of hole drilling, when the intensity of gas emission was similarly low in the adjacent holes drilled 1.0-1.5m away.

DEVELOPMENT OF OUTBURSTS PROCEDURES

The development of the D6 seam is made more difficult, due to the lack of overlying and underlying protective seams and a high gas content of coal. The D6 seam consists of an upper vitrinite rich section and a lower section which contained a soft coal layer, which is best described as a continuous shear zone varying in thickness of between 10's cm to around 100cm.

Different pre-drainage methods of gas-drainage holes were tested and procedures of outburst preventive measures were designed, for the management of gas-dynamic phenomena, particularly during roadways drivage in the upper layer of D6 seam. This resulted in 40-50 m of exploratory wells to be drilled for every one meter of development face advance.

Current outburst risk prediction is performed by evidence of gas and crushed coal yield from boreholes. Additional boreholes are drilled on the occasion of gas-dynamic phenomena signs, such as drilling assembly jamming, increased gas emission, gas and coal fines outburst.

Outburst risk prediction and outburst preventive measures are required to be implemented in every 4m of development face advance.

Availability of weak coal zone in the lower layer influences relief drilling length and speed. The holes are dry drilled and the 80 mm diameter holes are 30-40 m long. For larger 200-250 mm diameter holes, the borehole length can be as much as 20 m (average 12-15 m). Trials of longer hole drilling leads to drilling assembly jamming.

Normal development rates using the above procedures, in the upper layer of D6 coal seam were in the order of 25-30m per month.

UNDERLYING STONE DEVELOPMENT DRIVAGE

To overcome the low development efficiencies, JSC "ArcelorMittal Temirtau" Coal Division specialists designed a new method of development for the outburst risk prone D6 seam. The main points are summarised as follows:

The gate development roadways are driven under the coal seam at a distance 8-12 m from seam floor and beneath the headings, which will serve the future working of the upper layer. The development of the lower roadway driven in rock below the seam (rock gate) causes the redistribution of the stresses in the strata above. Degassing holes are drilled into the area of future roadway in coal seam from this roadway 60-80 meters behind the face. 3-5 holes are drilled to the seam roof in the form of a fan covering the future in seam development heading area.

The distance between fan holes cluster is 4 m. The degassing hole angle is defined by covering 4 m of rib area of the in seam gate road. The degassing holes are connected to a vacuum gas pipe range. The favourable conditions for safe roadway development in the upper layer of seam are created as a result of stress relief and the degassing operation from the underlying stone development. This also improves the upper roadway development rates. Figure 3 shows the current development method for the D6 seam as used in the Kazakhstanskaya and Lenina mines.

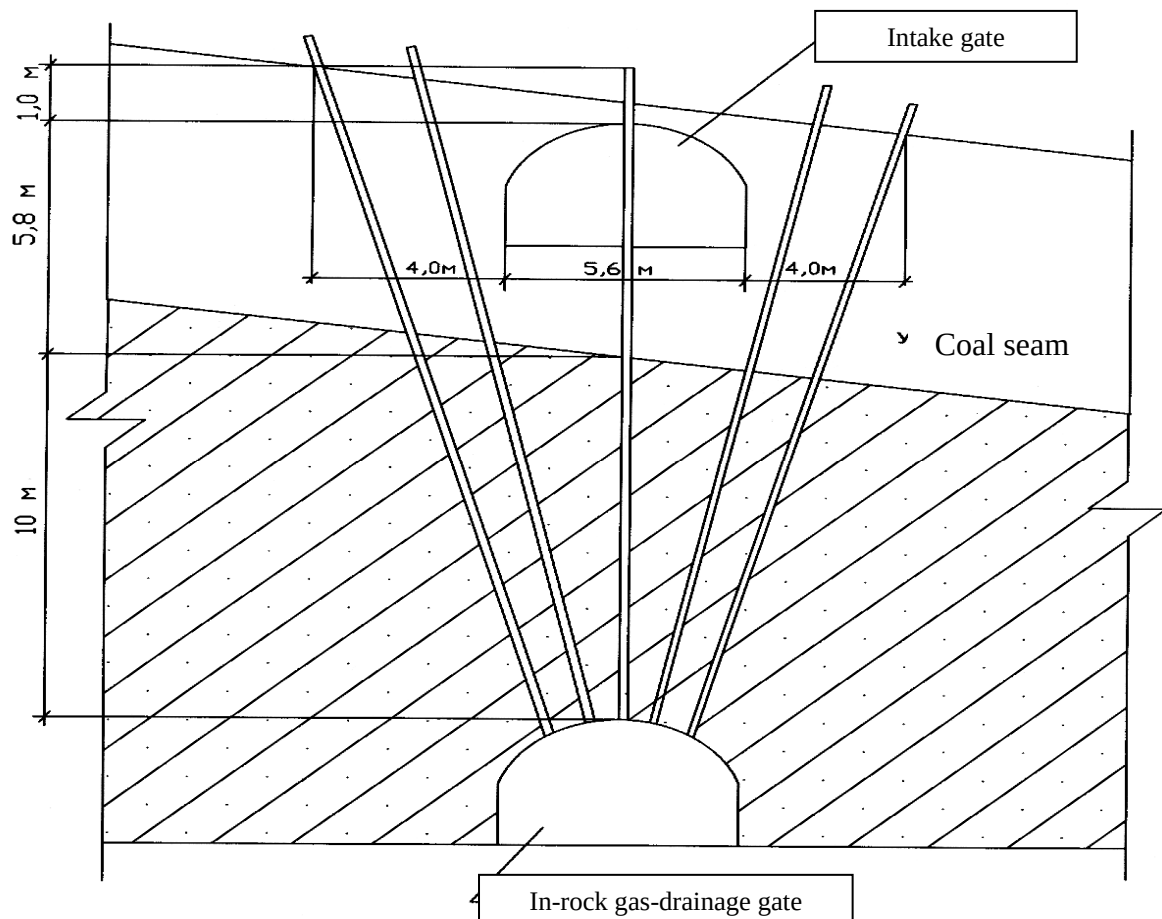


Figure 3 - Development method of most prone from outburst risk seam

OPERATIONAL EXPERIENCES

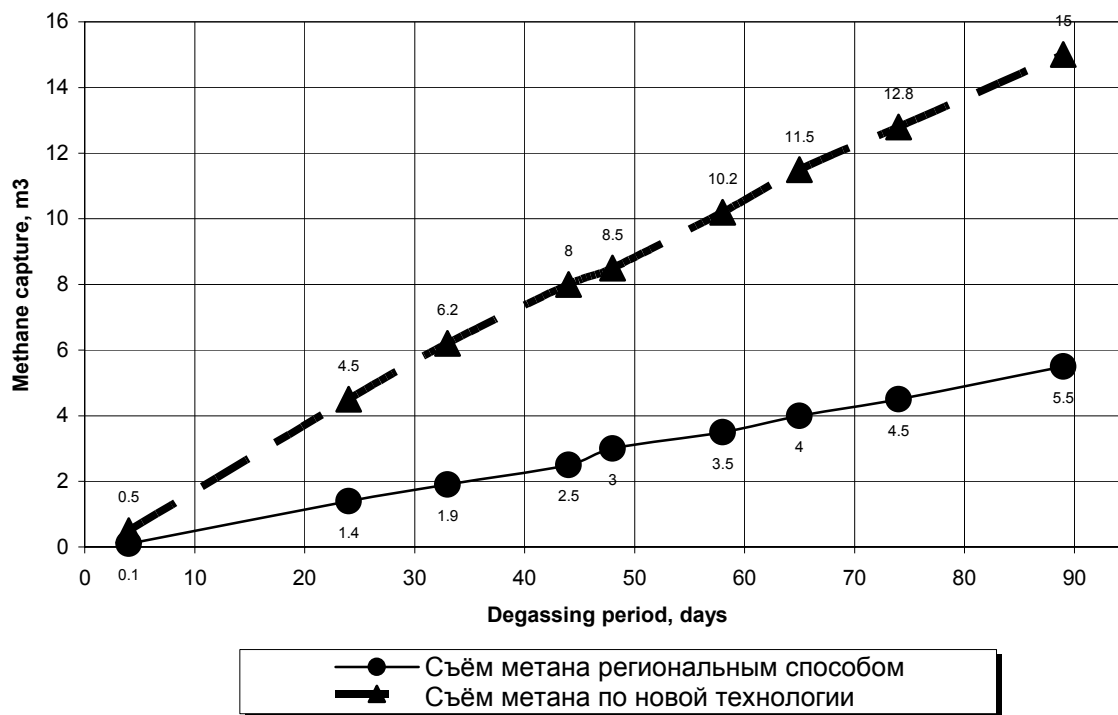
Methane extraction per tonne of coal in the vicinity of adjacent roadways at Kazakhstanskaya mine was 14-17 m³. This creates the conditions for safe in-seam roadways development in the upper layer. The average efficiency of holes was approximately 2 m³/min, and maximum up to 3.4 m³/min. The gas drainage allowed driving of in-seam roadways in the upper layer of D6 seam without outburst preventive measures and at a high rate of roadway development. The average development rates of 232 D₆-13 intake gate was 117m. The maximum development rate is now 150 m per month.

For comparison, the average development rates of 232 D₆-13 intake gateroad, 799m long, was 266 m per month. This was achieved with the implementation of all outburst preventive measures. Degassing implementation of the headings and surrounding areas is cost-effective. This accounts to 6.25 mln. tenge (US\$ 52,000) per m of development or 7800 tenge (US\$64) per tonne of working in the upper layer of D6 seam, even with longer in-rock gate and intake gate.

Degasification reduces the gas accumulation zones, thus contributing to minimising the possible coal and gas outburst occurrences. Such measures provided safe working conditions in the upper layer of D6 seam. A significant advantage of the new method is that the total production unit development period was reduced by six months even with the development increased development requirement.

The success of the operation at Kazakhstanskaya mine, allowed the introduction of the method to Lenina mine. As can be seen from Figure 4, the decrease in coal seam gas content adjacent to the newly developed of a drift was 15 m³ per tonne.

306 D6-east slope brake block #1 at Lenina mine



1 – methane capture by regional method

2 – methane capture by new technology

Figure 4 - Methane capture by degassing means

CONCLUSION

The experiences of the new development gas drainage method as applied of the longwall development of the outburst prone D6 seam, has the following advantages:

1. By carrying out the initial development in a gas free (stone) environment, underneath the seam outburst risks are minimal.
2. The stone drivages allows stress relief of the overlying coal seam and increasing of seam gas recovery in this area
3. Holes drilled form the stone drives enabled better effective gas drainage to a safe levels before developing the in seam longwall gate roads, particularly in the upper layer of D6 seam..
4. Development rates in relieved and degassed area in the upper layer of D6 seam has increased by three to four times. This contributed to improvement in mining and development rates.

A new development method at D6 seam is now being recommended for the further use at mines of JSC "ArcelorMittal Temirtau" Coal Division.

SYSTEM MANAGEMENT APPROACH TO IMPROVEMENTS IN LONGWALL DEVELOPMENT

Scott Barker¹ and Mehmet Kizil¹

ABSTRACT: The requirement for continuous improvement in the coal industry to achieve the combination of lower costs, greater capital efficiency and higher production. Longwall panels are now being mined at a faster rate requiring a commensurate increase in development rates. Development operations that cannot deliver continuity of longwall mining result in substantial flow on costs for the entire mine – from higher unit costs, lower production and loss of reliable supplier status in the coal market. A development unit which can remain well ahead of the longwall as a result of an efficient system – machines; human resources; supply logistics and efficient planning and scheduling is a vital factor in delivering a productive longwall mine. While capital solutions are often used to increase production, they may not lead to improved profitability. Often, a more efficient approach is to generate improvements through better management of existing equipment. Through an analysis of the development delays at an Australian longwall mine, a number of potential areas for improvement have been identified. Increasing operating time is one aspect of increasing roadway development rates and the second is increasing the cutting rate. The time and motion study which was carried out at the same mine identified the bolting and “shuttle car away” components of the cycle as holding the most potential for improvement.

INTRODUCTION

Coal mining is vital to Australia, both in terms of its contribution to Gross Domestic Product (GDP) as well as providing a significant proportion of the country's energy needs. Underground mining accounts for approximately 25 percent of the total tonnage produced in Australia and longwall mining accounts for 80 per cent of the underground output (ABARE, 2008).

Longwall mining requires a substantial amount of roadway development prior to the actual installation of the longwall equipment and mining of the longwall block. With the recent increase in the coal price and the overall push for higher production, the focus from most mines has been on increasing the productivity of the longwall machines as this equipment is the principal driver of the production output of the mine.

Longwall Roadway Development

Roadway development utilises three key pieces of equipment which are: the continuous miner; the shuttle car; and the conveyor system. The continuous miners used in Australia are generally in place systems which utilise a combined bolter miner set up to avoid excessive flitting between headings. The typical development panel layout is set out in Figure 1.

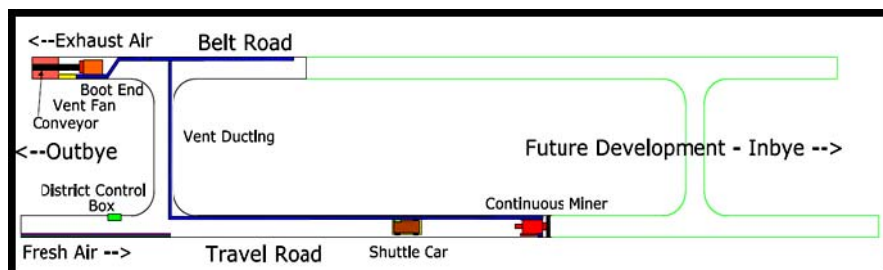


Figure 1 - Development Panel Layout

The mining cycle includes cutting, bolting, meshing, extending ventilation tubing, panel extensions and shuttle car loading and travel times. A typical advance rate of a continuous miner is 2 m/hr or 10 km

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over a calendar year. Cut-through roadways are driven between the belt road and travel road every 100 m, to leave regular pillars.

The cut-throughs help complete the ventilation circuit for development and allow travel between headings in the longwall. Each time the development section finishes a new cut-through section of the gate road, a panel extension takes place. A typical cycle from one panel extension to the next means that the miner has completed 230 m of development (100 + 100 + 30) which typically takes 9 days (18 x 12 hour shifts). Figure 2 shows an active development face including: bolting rigs, cutting drum and personnel.



Figure 2 - Active Development Face

The development section of the mine must remain ahead of the longwall so that the longwall equipment can be installed into the new block without waiting for the development to finish. A problem that the industry is currently facing is that longwall production rates have increased faster than development rates and many mines are struggling to maintain a development float. Wilkinson (2008) states that *"Longwalls are recognised money spinners, but the "pulse" of a longwall mine is determined by development rates. Generally, throughout Australian longwall mines development float is marginal or non-existent, and various efforts and techniques are employed to reduce the possible negative and restrictive effects it could have on longwall production levels."*

Development Delay Time Analysis

In order to increase the roadway development rate, the continuous miner must operate for longer or at a quicker rate. A development delay time analysis was carried out at an Australian mine to determine the important factors affecting the utilisation the development process. It can be seen in Figure 3 that operational delays make up 29 per cent of the calendar time with panel extensions accounting for 17 per cent of these delays. Planned maintenance accounts for 14 per cent of total calendar time which is the second biggest single delay component after operational delays.

The operational delays include over 68 categories. Of these categories, the top two comprise over a quarter of the operational delay time and the top 11 form over 63 percent of the operational delay time (Figure 4).

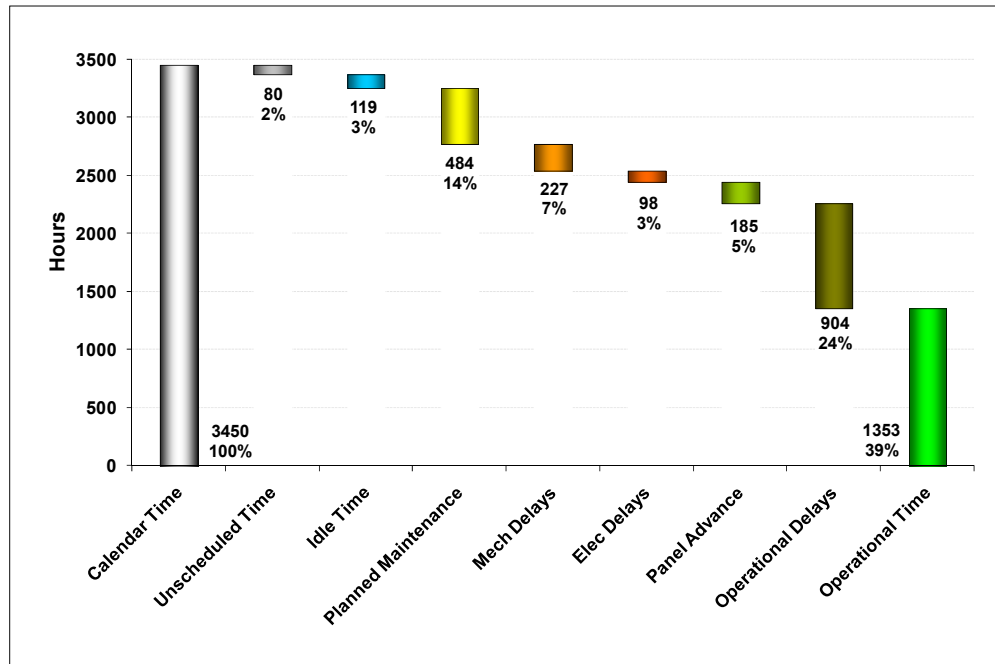


Figure 3 - Break Down of Calendar Time

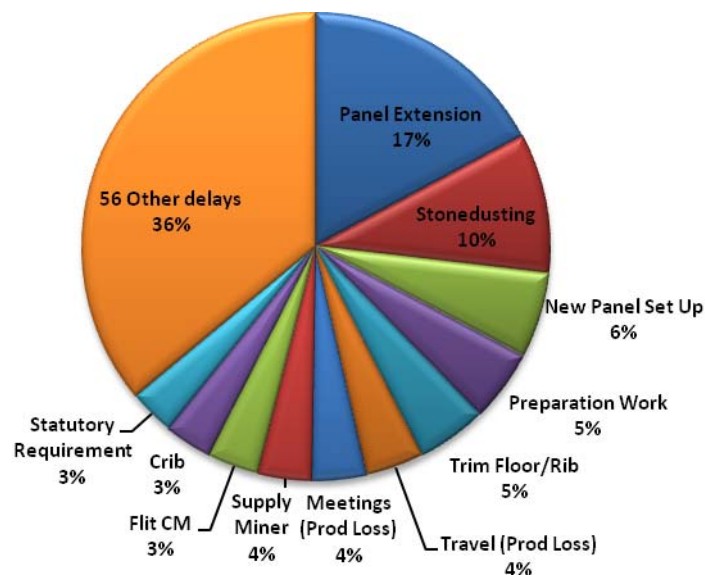


Figure 4 - Top 11 Operational Delays

The operational delay component was recognised as having the most potential for improvement. This is because most of these delays can either be avoided by improved process management or reduced by planning for activities. An example of such management initiative is to increase manning levels on panel advance shifts in order to complete the panel advance more quickly. Further to this, delays such as travel, crib and meetings rarely occur in the longwall sections of the mine due to "hot seat change over" policies, tighter monitoring and accountability. These standards should also be applied to the development sections.

The significance of increasing the operating time increases with higher cutting rates. It is therefore important to study both the operating rate and the operating time for the development section. The time and motion study conducted at the same longwall mine was aimed at identifying process inefficiencies within the cutting cycle to increase the operating rate (metres per operating hour).

Time and Motion Study

The cycle times recorded in the study ranged from 10 minutes and 25 seconds to 24 minutes and 45 seconds, equating to between 2.4 – 5.76 metres per operating hour. It was found that 55 per cent of the cycles lie within the 10 – 16 minute bracket however there tends to be a small number of cycle times well above the median time which greatly reduce the overall metres cut per shift.

The timing study provided a sample of the current production activities which provide an example of the current roadway development practice. To illustrate the raw timing data, Figure 5 shows the high, low and average times for each event in the cutting cycle. The important concept to be noted from Figure 5 is that the events with a large time variance are the events that potentially require review. Consistency is the key to reducing the overall cycle time and in turn optimising development. It can be seen that the shuttle car away and bolting events had the most variance with the maximum times recorded being 3 to 5 times the minimum. For an efficient development cycle, each event must fit in with the other events so that whenever possible tasks are being completed in parallel.

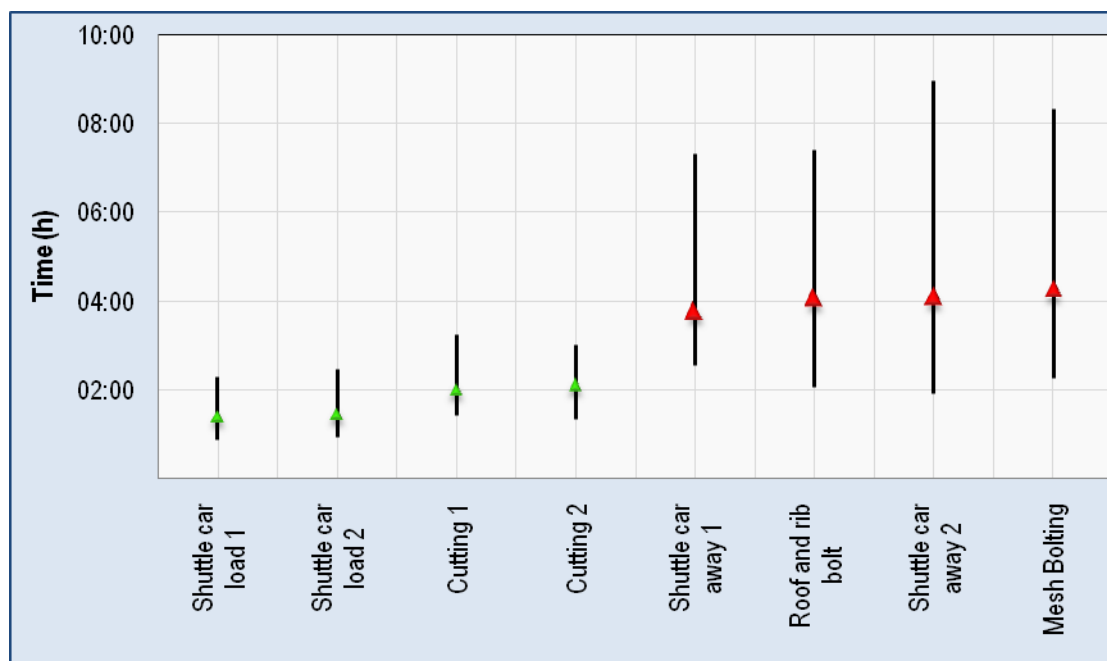


Figure 5 - Delay Cycle Components

The major technological improvements are currently aimed at improving the bolting and shuttle car away times (SCAT) with the use of one step bolts and flexible conveyors. Reducing these two components of the cycle by a certain per cent has the greatest impact on reducing the overall cycle time.

The potential for reducing SCAT depends mainly on the wheeling speed. The wheeling speed is determined by a combination of the following parameters:

1. Roadway floor condition (consistent horizon control);
2. Spillage of coal on floor (correctly loaded shuttle cars);
3. Angle of cut-through roadway (mine planning);
4. Condition of shuttle car (maintenance history);
5. Operator experience and training;
6. Panel procedures – roadway clear of pedestrians (SOP's); and
7. Conditions at boot end (clear spillage regularly).

If the SCAT is reduced, the critical path of the total cycle time will be constrained by bolting for a greater proportion of the pillar. It will therefore become more important to reduce the bolting times.

A greater level of automation with the use of one step bolting has significant potential for increasing the consistency and reducing the length of bolting times. Gibson (2005) identified technology improvements in coal clearance and self-drilling bolts as having the most potential for improved roadway development. Other parameters which impact on the bolting times are:

- Condition of equipment;
- Quality of bolts, drill bits and resins;
- Mining height and roadway width; and
- Ergonomics of equipment.

For improvements to be made within the cutting cycle, a system for monitoring change and providing feed back to operators is essential. An electronic monitoring system and production improvement study, in most development operations would help in create a reliable development float for the operation resulting in improved productivity for the mine.

Scheduling and Mine Planning

The key issue for roadway development is to avoid standing the longwall panel. Apart from improving development rates, as previously discussed, system scheduling can help deliver continuity in longwall production.

As a result of the shortage of qualified planning engineers and an overall shortage of skills throughout the industry, longwall and development supervisors are rarely engaged with the logic behind the final mine schedule. This then forces supervisors into a messenger type role in regards to shiftly and weekly production targets. In order to shift short term planning information to the supervisor level, the longwall/development animation spreadsheet has been designed. By enabling supervisors to run basic scheduling simulations, development and longwall sections will most likely be better managed as the front line management will have a better understanding of the interaction between development and longwall production. In the long run, a greater level of accountability will be expected from supervisors in terms of the development float.

With scheduling software packages that are designed for highly trained technical personnel becoming more complex, a potential market for software which enables front line supervisors to complete simple scheduling tasks which will contribute to improved system performance has been identified.

The spreadsheet software which was designed fits this purpose and has the potential to become a simple and useful scheduling software package in the future. The spreadsheet can be used to:

- Quickly determine development float/longwall stand time given the appropriate input parameters;
- Display the number of metres or tonnes currently mined on a shift by shift basis;
- Assess the impact of changing mining rates or operating time;
- Determine the effects of changing the mining parameters such as longwall block length, cut-through length and width and longwall face width;
- Estimate the financial outputs given the input parameters; and
- Clearly show/teach the impacts of various mining parameters on the mine schedule to crews, students and non-mining personnel.

The animation output page (Figure 6) is where the input parameters are used to simulate the longwall and development mining operations. The longwall panel at the top of the figure shows the current development section and the panel at the bottom represents the active longwall. As each section progresses shift by shift, the program indicates the mined area by changing colour (red for development and blue for longwall mining) and the tables at the bottom of the page display information relating to the performance of each section. The cell at the bottom of the sheet displays the development float/longwall stand information depending on which section is ahead of the other.

Once the animation has finished, the panels are shown as fully extracted and the tables at the bottom of the sheet provide the final output details. The cell at the bottom of the page also displays the difference in the time taken to mine the two production sections which indicates either a longwall stand or development float (Figure 6).

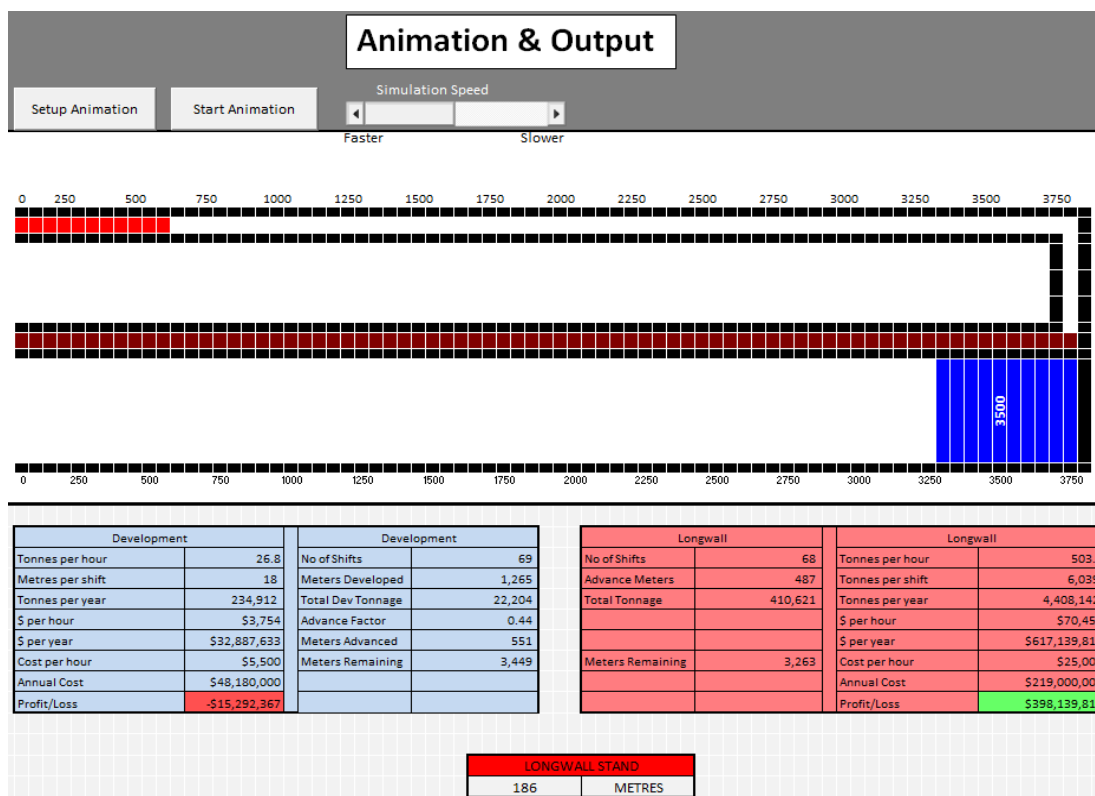


Figure 6 - Longwall mine scheduling tool

The animation itself was programmed using Visual Basic in Microsoft Excel (VBA), which provided a platform to create a concept level simulation program. Future versions of this animation spreadsheet will require more detailed programming software to avoid the restrictions of working within cells in Excel.

CONCLUSIONS

It is widely accepted that the roadway development potential for current equipment has yet to be fully realised due to the number of restraints in the system. The current improvement strategy focused on the two variables which determine production rate, these are operating time (hours) and operating rate (metres per operating hours).

After completing the time and motion study, it was found that the bolting and SCAT time components of the cycle had the greatest impact on the cutting time and were the largest inhibitors to the overall operating rate due to the large variance in these component times. It was found that operators had a larger input to the bolting times and SCAT than the other cycle components which provides some explanation of the large variance in recorded times.

In response to an industry which is struggling to maintain development float, a simple scheduling tool was designed to help supervisory level employees conceptualise the interaction between longwall and development production rates.

An increase in roadway development rates can be achieved not through the introduction of more equipment but through a good framework of management and operating systems which assist in maintaining production and process continuity. Through the use of electronic monitoring, future development sections will have increased accountability for the work carried out by personnel and equipment as well as provide a platform for improvement initiatives.

ACKNOWLEDGEMENTS

The authors would like to thank Xstrata Coal Queensland for allowing access to one of its mine and the support collecting the relevant data for this publication.

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A SIMULATION MODEL FOR ROADWAY DEVELOPMENT TO SUPPORT LONGWALL MINING

Geoff Gray¹, Ernest Baafi², Ian Porter² and Osvaldo Rojas¹

ABSTRACT: The practice of longwall mining is potentially an efficient and cost effective means of extracting coal from underground seams. However, for production targets to be achieved not only must the longwall perform as expected but roadway development must be kept ahead of the longwall advance. The evaluation of development options is a challenging exercise due to the interactions between mining, tramming and clearance operations. In particular, the high levels of variability and uncertainty in operations make it difficult to assess how a particular configuration may perform. A simulation model *RoadSIM* using the ARENA modelling system has been produced to assist in the analysis of the roadway development process. The *RoadSIM* simulator provides means for assessing the operational limitations of roadway development practices at a particular coalmine. The simulation model provides a what if tool to allow a range of equipment, configuration and operating practices to be assessed in terms of achievable advance rates and equipment utilisation. Output from the simulator is in the form of a dynamic visualisation as well as summary reports and detailed logs of operations over time.

INTRODUCTION

The practice of longwall mining is potentially an efficient and cost effective means of extracting coal from underground seams. For longwall production targets to be achieved roadway development must be kept ahead of the longwall advance. In practice there are various inherent characteristics of roadway development practices which fundamentally influence their overall performance rates. One of the recommendations made in a recent report on current practices of Australian Roadway Development by Gibson (2005) was to use a process simulation model to map, manage and monitor the performance of roadway development. In response to this recommendation a project was setup under the Australian Coal Industry Research Program (ACARP) to map and model the roadway development process. As a result of the ACARP project a simulation model of roadway development operations, *RoadSIM*, has been developed.

DYNAMIC SIMULATION

Dynamic system simulation provides a proven technique to study interaction between elements of a complex system. The basic methodology has been available for over 40 years. However advances in computing technology and software development tools have seen increased use of the technique over the last 5-10 years. This type of simulation technology is referred to by a number of different names, these include:

- dynamic simulation,
- discrete event simulation,
- numerical simulation and
- Monte Carlo simulation.

The technique allows for the modelling of a system over time and the interactions between system components to be considered. For example interactions between the miner and shuttle cars, or traffic issues related to shuttle car movements. An important aspect of this technique is its ability to explicitly allow for the randomness/variability in a system. Without allowing for unplanned events and the impact operating cycles any modelling approach is almost certain to over estimate production capacity.

A simulation model allows a model user to identify bottlenecks and establish if local improvements are likely to have a system wide impact. This approach provides the opportunity to optimize an operation

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and allocate resources efficiently. The *RoadSIM* system utilises the technique of dynamic simulation. The model has been developed using the ARENA (Kelton, 2007) modelling system.

MODELLING THE ROADWAY DEVELOPMENT PROCESS

The current modelling focuses on the unit operations of two-heading developments from the face to the boot end. The modelling allows, for but does not explicitly model the movement of services etc associated with a sequence move. The aspects of the roadway development process explicitly considered in the modelling include:

- cutting and loading at the development face,
- support,
- shuttle car tramming and
- shuttle car discharging.

To provide useful results any modelling system will need to consider:

- Operations over a complete pillar.
- Allowance for planned and unplanned events that effect production.

In modelling/analysing the underground roadway development process it is essential to consider operations across the entire pillar as the dynamics changes as the miner moves away from the boot end. As the miner advances the tramming distance increases. The increased tramming distance increases the shuttle cycle time and reduces its haulage rate. As a result, the combination of miner, support and car capacities that are most productive at the start of the pillar may not be the best combination towards the end of the pillar. Analysis needs to consider operations across the entire pillar.

RoadSIM modelling system

RoadSIM simulator has inbuilt ability to respond to "what if" variations and provides means for assessing the operational limitations of roadway development practices at a coalmine. Figure 1 shows the basic components of the system. The key attributes of the *RoadSIM* simulation system are:

- The ability to respond to "what if" questions. For example, impact on roadway development rate if the development resources, technologies or practices are altered.
- Reproduction of the randomness of roadway development processes.
- The use of animation to view roadway development operations and bottlenecks.

The *RoadSIM* system has been developed using Microsoft's Excel spreadsheet software and the Arena Simulation system. Excel is used to provide a flexible and familiar data entry and reporting facility while ARENA is used to provide the simulation engine and animation facilities.

Using the system involves defining the configuration to be modelled then testing likely performance by running the model and reviewing the animation and output report. Each model run of the simulator is replicated many times with the same model setup but different unplanned events. Results generated provide likely range for time to develop a pillar length, exposure rates (m/hr) as well as utilisation of time. The model once configured can be used to explore the potential impact on development rates and equipment utilisation of alternate practices and/or equipment.

Typically *RoadSIM* would be used to consider the impact on development rates of aspects of operations such as:

- pillar and cut through dimensions,
- number of shuttle cars in use,
- cycle times for cutting and loading at the development face,
- cycle times for bolting,
- shuttle car tramming speeds,
- cycle time for discharge of a shuttle car,

- delays effecting f out bye services, and
- delays effecting face operations.

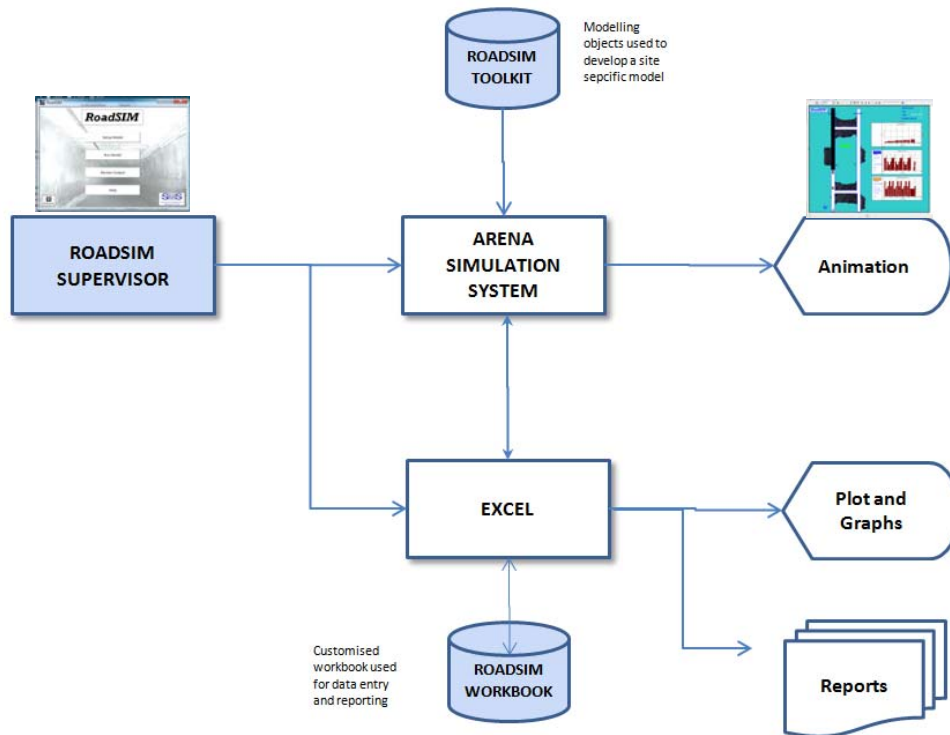


Figure 1 - Basic Elements of RoadSIM System

RoadSIM setup

An example of the Excel workbook used to set up the simulation model is shown in Figure 2. Data is entered using customised Excel spreadsheets enhanced with a menu and navigation system. Data required to setup the model includes:

- roadway dimensions,
- the performances of continuous miner, bolter and the shuttle car, and
- planned delays and unplanned (randomly imposed) delays.

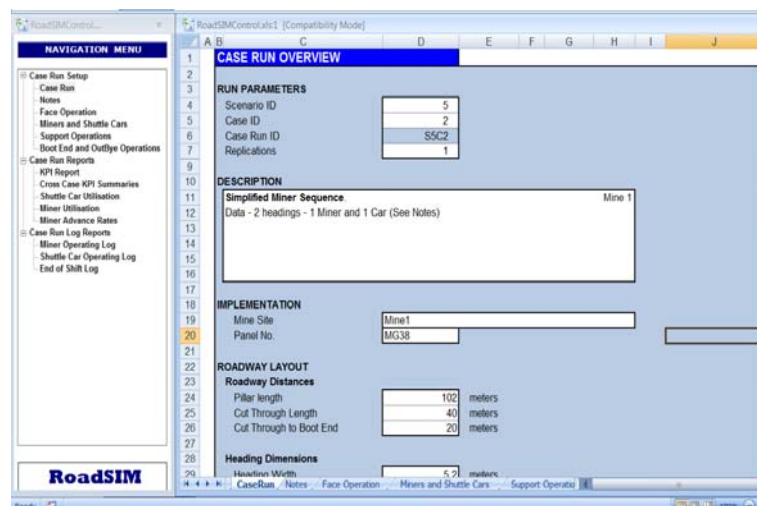


Figure 2 - RoadSIM run overview

Once the model is configured the model is run so that the animation (Figure 3) can be reviewed to both ensure the operations are as intended and to provide an understanding of the dynamics of the system. To complete the analysis each model run is replicated many times with the same model setup, but different unplanned events. When analysis is complete the model results are loaded back into Excel for review.

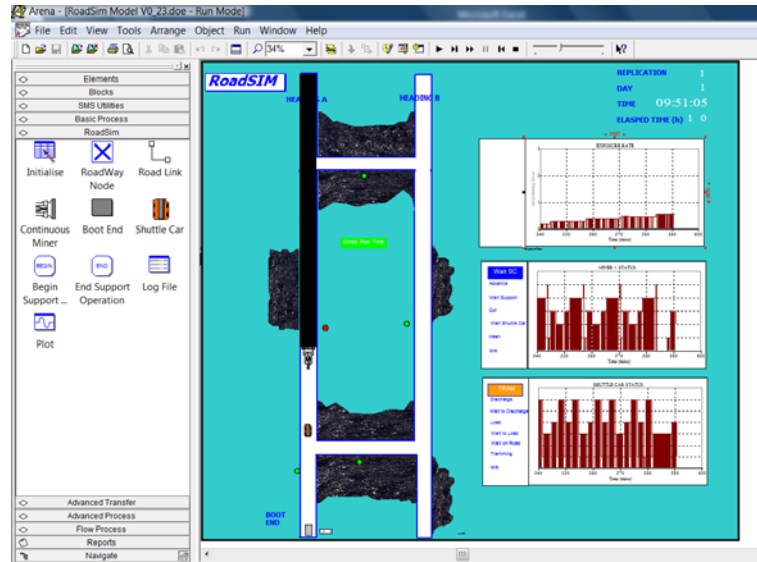


Figure 3 - RoadSIM animated display

RoadSIM output

Model output provides summary information on performance indicators such as:

- First coal to first coal time between pillars.
- Face exposure rates in meters per operating hours.
- Miner utilisation.
- Shuttle car utilisation.

As well as high level summaries (Figure 4) more detailed information is available on likely advance rates across the length of the pillar (Figure 5).

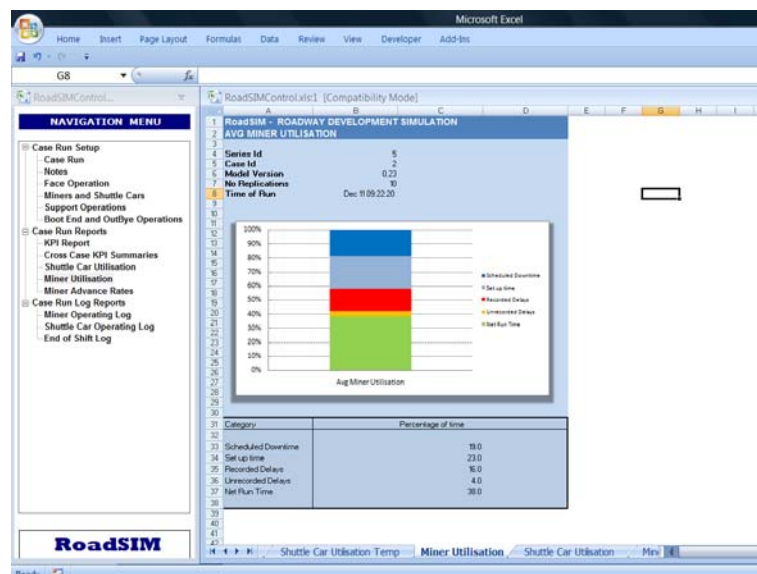


Figure 4 - RoadSIM Summary Statistics

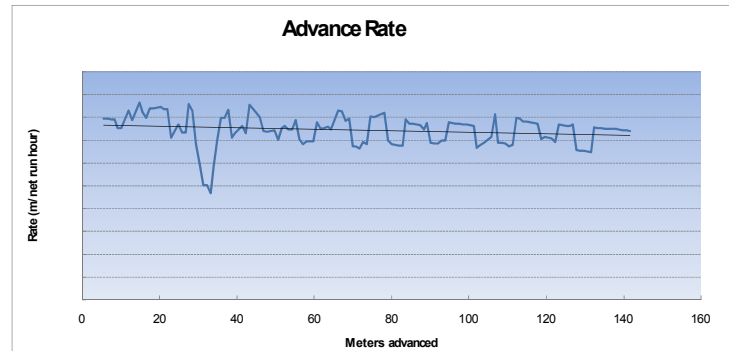


Figure 5 - RoadSIM Sample Output

By comparing the performance statistics generated the model can be used to determine if options being considered provide a improvement in development rates, or simply move a bottleneck to another point in the process.

CONCLUDING REMARKS

A discrete simulation model has been developed using ARENA for the Australian coal mining industry to model and manage the performance of roadway development as required to support longwall mining operations. The modelling system allows various options to be explored to determine how a required development rate can be best achieved to support longwall advance rates. The current version of the *RoadSIM* template provides a model for up to two continuous miner with single or multiple shuttle car operations operating in two headings. Future development work will include alternative headings and face equipment configurations as well as continuous haulage systems.

ACKNOWLEDGEMENT

The authors will like to thank Gary Gibson of Gary Gibson & Associates, Richard Porteus of Xstrata Coal and members of Roadway Development Task Group of ACARP for their valuable input during the model development and also making mine operational data available for *RoadSIM* model configuration and validation. The development of *RoadSIM* was funded under ACARP Project C17019.

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A WEB-BASED DATABASE FOR ASSESSING ROADWAY DEVELOPMENT PERFORMANCE

Luke Viglione¹, Ernest Baafi¹, Amir Hesami¹ and Gary Gibson²

ABSTRACT: There are various inherent characteristics of roadway development practices which fundamentally influence their overall performance rates. As part of a strategy to improve roadway development performance in Australian longwall mines, Australian Coal Association Research Project (ACARP) has initiated a benchmarking study by surveying each longwall mines. A web-based relational database system which has been developed purposely to monitor the performance of Australia longwall roadway developments is described. The benchmark study is aimed at assessing both the physical and operational factors influence roadway development practices and their performance indicators.

INTRODUCTION

In responding to why some longwall roadway development rates are different among mines, Roadway Development Task Group (RDTG) of ACARP has initiated a benchmarking study to determine the correlation between roadway development practices and roadway development performances. Since 2006 mine operational survey data has been compiled using EXCEL spreadsheet and via an online web-based database management system. Statistical analysis of an earlier survey suggested that continuous miner units that achieved more than 1,750 m were units mining two-heading development. The same survey results showed that having six bolting rigs improves the probability of higher development metres, while having less than six bolting rigs heavily limits performance capability. Also, installing 1500 mm rib bolts also appears to limit development rates, with those continuous miners achieving higher production levels all installing 1200 mm rib bolts.

Unfortunately there has been a limited success in providing data due to lack of participation by many of longwall mines due to the amount of time required in completing the survey information. This paper describes a user friendly web-based online relational database system, *BenchDat*, with various functionalities aimed at reducing the amount of time required by the user to complete the mine operational and performance data. The model structure of *BenchDat* is shown in Figure 1. *BenchDat* has two main databases with the first database made up eight tables. The second database was created using Visual Studio (VS) and Active Server Pages (ASP) Net configuration with the usual fields and tables.

USING BENCHDAT SYSTEM

Upon logging onto *BenchDat* website, the image shown in Figure 2 is displayed for the user to update any changes to the mine's basic information. This includes the number of operating units of gateroads and mains. Once the mine's respondent details have been updated the main body of the database can be completed or updated by selecting each of the circled sections of Figure 3.

The basic information required from the mines is classified under:

- Mine Parameters
- Development Parameters
- Gas and Ventilation
- Shifts and Personnel
- Management of Development Performance
- Development Performance

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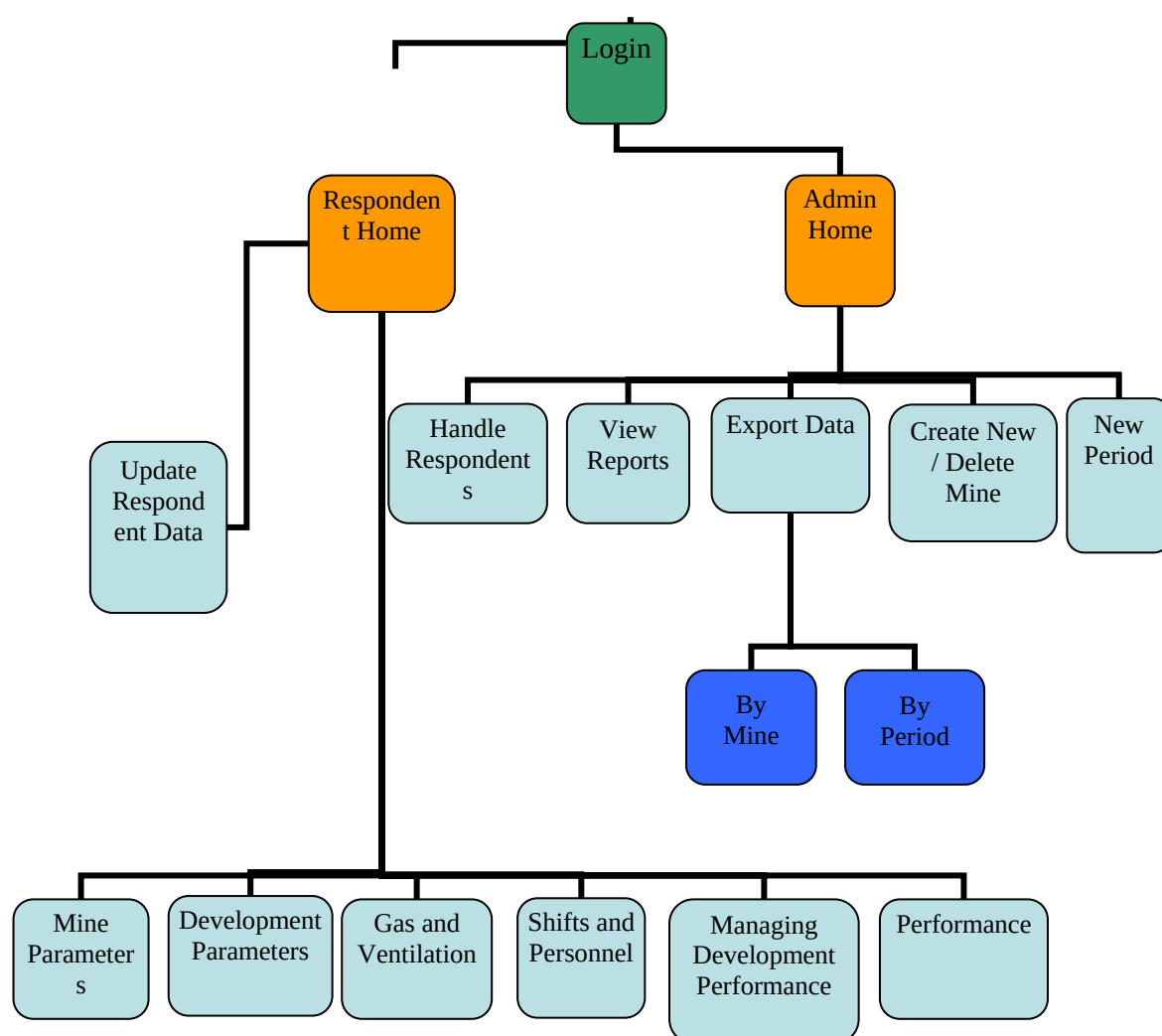
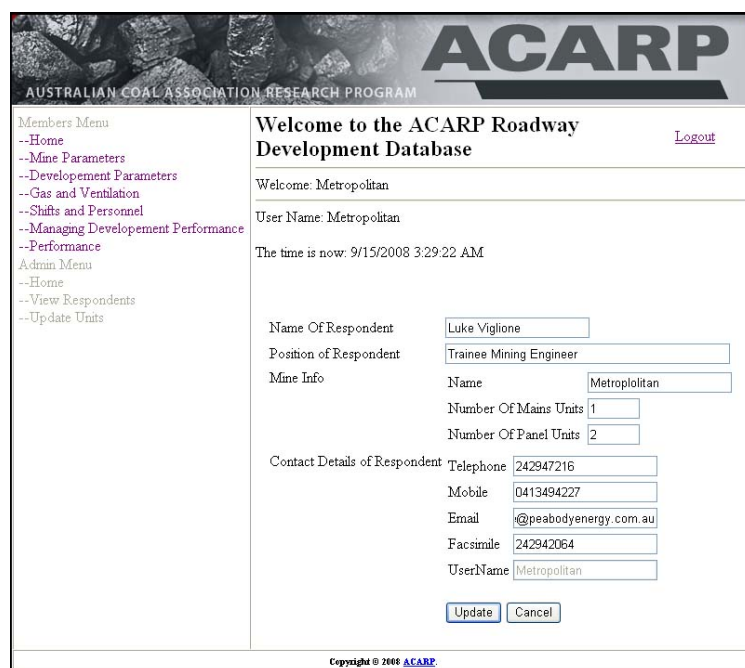


Figure 1 – Structure of BenchDat

The information required from mines under Development Performance option are:

Additional roof/rib support
 Floor conditions
 Flitting equipment within panel
 Stone dusting
 Services installation and/or extension
 Belt extensions/panel moves
 Travel/transport delays
 Insufficient labour/manning levels
 Meetings and/or briefings

Poor roof conditions
 Difficulty of mining conditions
 Supplies
 Pumping
 Panel preparation
 Panel relocation
 Training
 Absenteeism



ACARP
AUSTRALIAN COAL ASSOCIATION RESEARCH PROGRAM

Welcome to the ACARP Roadway Development Database [Logout](#)

Welcome: Metropolitan

User Name: Metropolitan

The time is now: 9/15/2008 3:29:22 AM

Name Of Respondent:

Position of Respondent:

Mine Info: Name

Number Of Mains Units:

Number Of Panel Units:

Contact Details of Respondent: Telephone

Mobile:

Email:

Facsimile:

UserName:

Copyright © 2008 ACARP

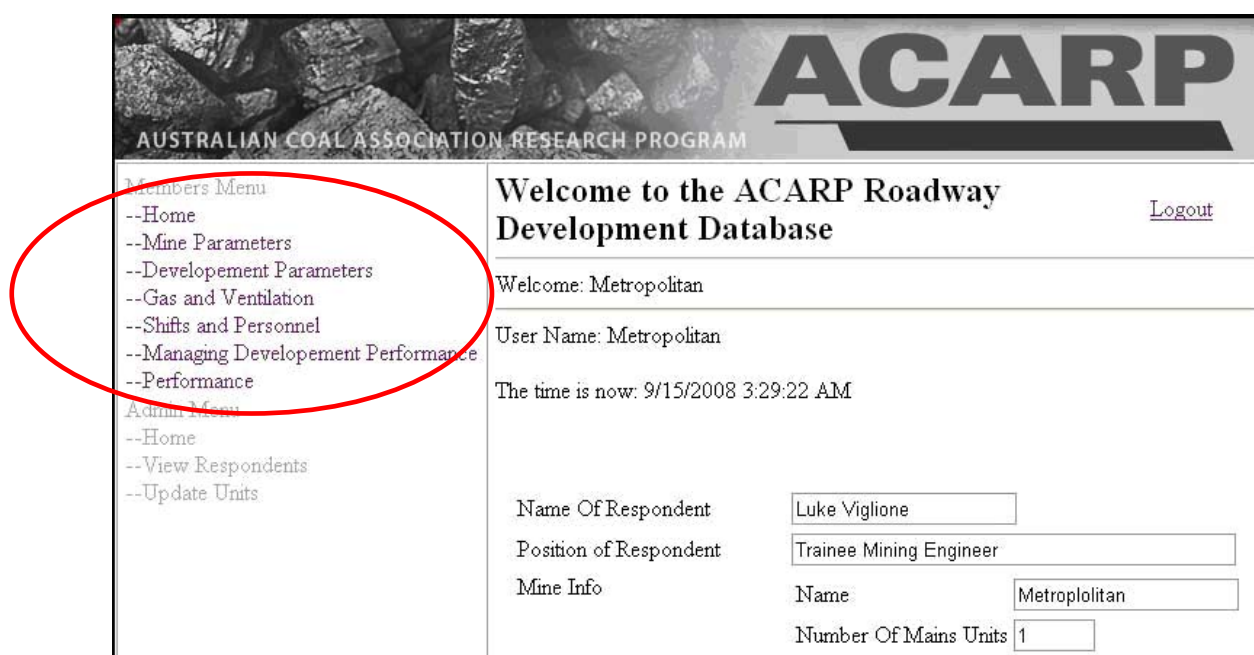
Members Menu

- Home
- Mine Parameters
- Development Parameters
- Gas and Ventilation
- Shifts and Personnel
- Managing Development Performance
- Performance

Admin Menu

- Home
- View Respondents
- Update Units

Figure 2 – Home page of BenchDat



ACARP
AUSTRALIAN COAL ASSOCIATION RESEARCH PROGRAM

Welcome to the ACARP Roadway Development Database [Logout](#)

Welcome: Metropolitan

User Name: Metropolitan

The time is now: 9/15/2008 3:29:22 AM

Name Of Respondent:

Position of Respondent:

Mine Info: Name

Number Of Mains Units:

Members Menu

- Home
- Mine Parameters
- Development Parameters
- Gas and Ventilation
- Shifts and Personnel
- Managing Development Performance
- Performance

Admin Menu

- Home
- View Respondents
- Update Units

Figure 3 – Mine primary data input options

THE AUTO FILL FUNCTION

A mine with a number of units of gateroads or mains presents a tedious task of entering identical data for each unit. The **Auto Fill** tool of *BenchDat* system allows data from similar units with the same response to be entered once. To use this function the fields for the first unit information is filled and then select the **Auto Fill** box is clicked. The remaining fields are filled once the **Save** button is selected. An example of this is shown in Figure 5.

Some fields in the database only accept only numerical data. In the case that a text is entered into these fields an error message is displayed and the field in question highlighted in red for a correction by the respondent. After completing all the necessary mine data the user logs out from the system.

SHIFTS AND PERSONNEL PARAMETERS for Ulan		Mains Unit 1	Panel Unit 1	Panel Unit 2	Panel Unit 3	Panel Unit 4
1	Indicate the mines personnel and materials access		☒ Auto Fill			
	Shaft haulage/winder	☒	☐	☐	☐	☐
	Drift haulage/winder	☒	☐	☐	☐	☐
	Inclined drift/rubber tyred	☐	☒	☐	☐	☐
	In seam entry off highwall or outcrop	☐	☐	☐	☐	☐
2	Typical personnel travel distances and times between the surface muster area and the working faces (kms/minutes)		☒ Auto Fill			
	Distance (km)	4	9.8			
	Time (minutes)	20	40			
3	Typical material transport distances and times between the surface materials supply and loading area and the working faces		☒ Auto Fill			
	Distance (km)	4	9.8			
	Time (minutes)	20	90			
4	Material supply system utilised in face area (✓)		☒ Auto Fill			
	LHD	☒	☒	☐	☐	☐
	Trailers	☒	☒	☐	☐	☐
	Supply cassettes and/or pods	☒	☒	☐	☐	☐
	QDS loading attachments	☒	☒	☐	☐	☐
	Manually transferred and loaded	☐	☐	☐	☐	☐

Before Saving

SHIFTS AND PERSONNEL PARAMETERS for Ulan		Mains Unit 1	Panel Unit 1	Panel Unit 2	Panel Unit 3	Panel Unit 4
1	Indicate the mines personnel and materials access		☐ Auto Fill			
	Shaft haulage/winder	☒	☐	☐	☐	☐
	Drift haulage/winder	☒	☐	☐	☐	☐
	Inclined drift/rubber tyred	☐	☒	☒	☒	☒
	In seam entry off highwall or outcrop	☐	☐	☐	☐	☐
2	Typical personnel travel distances and times between the surface muster area and the working faces (kms/minutes)		☐ Auto Fill			
	Distance (km)	4	9.8	9.8	9.6	9.2
	Time (minutes)	20	40	40	40	40
3	Typical material transport distances and times between the surface materials supply and loading area and the working faces		☐ Auto Fill			
	Distance (km)	4	9.8	9.8	9.6	9.2
	Time (minutes)	20	90	90	90	90
4	Material supply system utilised in face area (✓)		☐ Auto Fill			
	LHD	☒	☒	☒	☒	☒
	Trailers	☒	☒	☒	☒	☒
	Supply cassettes and/or pods	☒	☒	☒	☒	☒
	QDS loading attachments	☒	☒	☒	☒	☒
	Manually transferred and loaded	☐	☐	☐	☐	☐

After Saving

Figure 5 – Auto Fill function

SYSTEM ADMINISTRATOR

The *BenchDat* database system has been designed to compile the annual development performance of Australia longwall mines. One of the roles of the system administrator is to reset the periods. This function involves first selecting the *Stop Period* button and then selecting the *Start New Period* button (refer to Figure 6). Other options made available to the system administrator include managing the contact details of the mine respondents, viewing responses and exporting data for statistical analysis. At any stage, the system administrator can easily determine which mines have responded and to what extent.

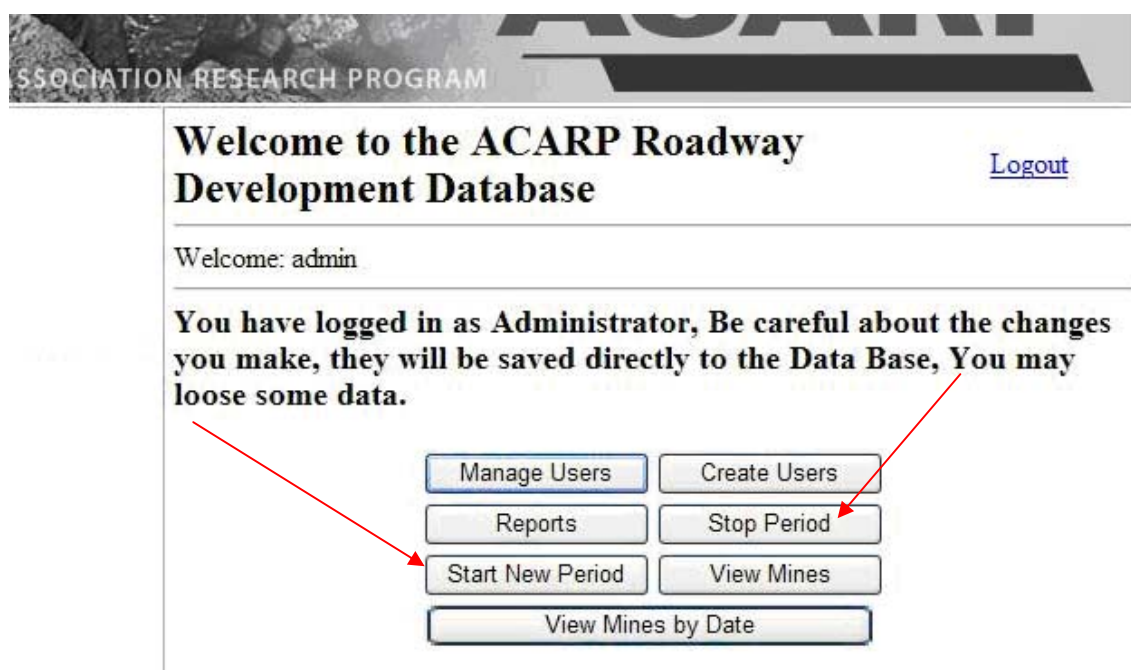


Figure 6 – System administrator menu

CONCLUDING REMARK

Future developments of *BenchDat* database include a detailed reporting structure for exporting compiled data to Microsoft Excel Comma-Separated-Value (csv) format for a comprehensive statistical analysis. The site will eventually reside at the industry's website *undergroundcoal.com.au* currently under construction. The development of *BenchDat* database was managed by Luke Viglione in partial fulfilment of the subject *ENGG371 Scholars Project 3* at University of Wollongong

OUTCOMES OF THE INDEPENDENT INQUIRY INTO IMPACTS OF UNDERGROUND COAL MINING ON NATURAL FEATURES IN THE SOUTHERN COALFIELD – AN OVERVIEW

Bruce Hebblewhite¹

ABSTRACT: An independent panel of experts was appointed in December 2006, jointly by the NSW Minister for Planning and the Minister for Primary Industries & Energy to review the current status and future implications for underground coal mining in the Southern Coalfield of NSW, with respect to its impact on natural features, particularly emphasizing risks to water flows, water quality and ecosystems; and to provide advice on best practice in regard to assessment of subsidence impacts; avoiding or minimizing adverse impacts; and management, monitoring and remediation of subsidence and subsidence-related impacts. The Panel received extensive submissions and presentations in September, 2007 before developing its findings which were reported to the Ministers and released in June 2008. A summary of the process followed by the Panel, and a discussion of the outcomes and recommendations from the Independent Inquiry, and their relevance and application across the underground coal industry in the Southern Coalfield are presented.

INTRODUCTION

On 6 December 2006, the NSW Government established an independent Inquiry into underground coal mining in the Southern Coalfield and appointed an Independent Expert Panel to conduct the Inquiry. The Inquiry was established by the Minister for Planning, the Hon Frank Sartor MP, and the Minister for Primary Industries, the Hon Ian Macdonald MLC.

The Inquiry was established because of concerns held by the government over both past and potential future impacts of mining-induced ground movements on significant natural features in the Southern Coalfield. These concerns first surfaced in the community in 1994 when the bed of the Cataract River suffered cracking and other impacts caused by mine-related subsidence from the underlying Tower Colliery.² Sections of the local and broader community have continued to express concerns at further subsidence-related impacts associated with this and other coal mines in the Southern Coalfield.

From 2010 all proposed extensions to underground coal mining operations will require approval under Part 3A of the *Environmental Planning and Assessment Act 1979*. Given the community concerns and the changes in the planning system, the Government announced the inquiry to provide a sound technical foundation for assessment under Part 3A (and other regulatory and approval processes) and long term management of underground mining in the Southern Coalfield by both the Department of Planning (DoP) and the Department of Primary Industries (DPI) and other key agencies (such as the Department of Environment and Climate Change (DECC), the Sydney Catchment Authority (SCA) and the Department of Water and Energy (DWE)).

Terms of Reference

The Terms of Reference for the Inquiry were to:

1. Undertake a strategic review of the impacts of underground mining in the Southern Coalfield on significant natural features (ie rivers and significant streams, swamps and cliff lines), with particular emphasis on risks to water flows, water quality and aquatic ecosystems; and
2. Provide advice on best practice in regard to:
 - a) assessment of subsidence impacts;
 - b) avoiding and/or minimising adverse impacts on significant natural features; and

¹ Professor & Head of School, School of Mining Engineering, UNSW

² Tower Colliery is now known as 'Appin West Coal Mine'. Appin West also includes the Douglas mining area.

- c) management, monitoring and remediation of subsidence and subsidence-related impacts; and
3. Report on the social and economic significance to the region and the State of the coal resources in the Southern Coalfield.

The terms of reference required the Panel to focus its examination on the subsidence-related impacts of underground mining on 'significant natural features'. These features were defined as 'rivers and significant streams, swamps and cliff lines.' Other natural features, for example plains, plateaus and general landforms, and any impacts of subsidence on infrastructure, buildings or other structures were not within the Panel's terms of reference. Similarly, impacts associated with constructing and operating surface facilities were considered beyond the scope of the inquiry. However, it was considered that certain values contributed to the significance of some natural features. These include values in respect of Aboriginal heritage, non-Aboriginal heritage, conservation, scenery, recreation and similar values.

In considering impacts on rivers, significant streams and swamps, the Panel was asked to place particular emphasis on 'risks to water flows, water quality and aquatic ecosystems'.

The reference to water flows and water quality was considered to relate not only to ecosystem functioning but also to reflect the water catchment values of large sections of the Southern Coalfield, which contains a number of water supply catchments, dams and other water supply assets. The reference within the terms of reference to 'aquatic ecosystems' was considered by the Panel to also include groundwater dependent ecosystems.

The Panel did not consider that its terms of reference extended to advising on the 'acceptability' of particular subsidence impacts. The Panel was not assigned this role. The role of determining the acceptability of environmental impacts rests with the Government and its agencies, as informed and influenced by the mining industry and other key stakeholders and the general community. The acceptability of predicted impacts is assessed and considered through various Government approval processes, in particular approval processes under the *Environmental Planning & Assessment Act 1979* and the *Mining Act 1992*. Similarly, the terms of reference did not ask the Panel to scale or measure the value or significance of individual examples of the listed significant natural features.

The Panel focused its inquiry on those parts of the Southern Coalfield which are subject to historic, current and prospective underground coal mining. This is principally the Illawarra Region extending westward to the townships of Tahmoor and Bargo.

The Panel comprised the following members:

- Professor Bruce Hebblewhite (Chair, subsidence expert);
- Emeritus Professor Jim Galvin (subsidence expert);
- Mr Colin Mackie (groundwater expert);
- Associate Professor Ron West (aquatic ecologist); and
- Mr Drew Collins (economist).

PROCESS

Following its appointment, the Panel sought a number of briefing sessions from relevant Government agencies (including DPI, DECC, SCA and DWE), industry groups (including the NSW Minerals Council and mining companies active in the Southern Coalfield) as well as community organisations actively expressing concern at subsidence-related impacts in the area.

The Panel, through the Department of Planning, then advertised its terms of reference and asked for written submissions from the wider community as well as offering the opportunity for presentations to be made before the Panel at public hearings. The advertisements sought submissions from the community, the industry and agencies and other interested parties by 30 July 2007. The Panel received 53 submissions by this date. A further 3 submissions were received after that date which, for their relevance to the Inquiry, were accepted by the Panel.

Of the submissions received, 6 were from Government agencies and statutory bodies, 26 were from interest groups (including community and other interest groups and local Government authorities), 7

were from industry bodies (including mining companies) and 17 were from individual community members.

The Panel held public hearings in Camden from 18 – 21 September 2007. At the hearings 28 persons made oral presentations. Of these presentations, two were made on behalf of Government agencies (DECC & SCA), 14 were made on behalf of community groups, interest groups and local Government authorities, four were made on behalf of industry bodies and eight were made by individual community members.

During the period of the public hearings (and subsequently), the Panel made a number of very informative field inspections covering areas of Waratah Rivulet, various smaller creeks and swamp locations, plus sections of the Cataract, Nepean, Georges and Bargo Rivers. The Panel was accompanied by representatives of both the mining industry and local community groups on each of these inspections.

Following the public hearings, all submissions were placed on the Department of Planning's website to give all submitters the opportunity to make a supplementary submission based on their review of other parties' submissions together with the information provided by way of presentation at the hearings. The Panel received 13 supplementary submissions through this process.

KEY FINDINGS

This paper does not attempt to address all the elements considered by the Panel. Rather, it is focused on a selection of the key findings directly related to subsidence behaviour and related prediction capabilities and future needs. The full report of the Panel is available from the Department of Planning website at www.planning.gov.nsw.au. The Appendix to this paper contains a direct extract from the Panel Report, containing the 22 Recommendations of the Panel Report.

Terminology

Within the Panel Inquiry submissions and presentations, there were a number of often contradictory, inconsistent or ambiguous uses of subsidence terminology encountered. The Panel felt the need to clarify this terminology and has attempted to provide a future standard for adoption by all parties. Particular terms of interest were: subsidence *effects*, *impacts* and *consequences*; together with the concepts of *conventional* and *non-conventional* subsidence behaviour.

The Panel has used the term *subsidence effects* to describe subsidence itself – ie deformation of the ground mass caused by mining, including all mining-induced ground movements such as vertical and horizontal displacements and curvature as measured by tilts and strains.

The term *subsidence impacts* are then used to describe the physical changes to the ground and its surface caused by these subsidence effects. These impacts are principally tensile and shear cracking of the rock mass and localised buckling of strata caused by valley closure and upsidence but also include subsidence depressions or troughs.

The environmental *consequences* of these impacts include loss of surface flows to the subsurface, loss of standing pools, adverse water quality impacts, development of iron bacterial mats, cliff falls and rock falls, damage to Aboriginal heritage sites, impacts on aquatic ecology, ponding, etc.

The *conventional* or general model of surface subsidence, which finds worldwide acceptance, is based on assuming the following generalised site conditions:

- the surface topography is relatively flat;
- the seam is level;
- the surrounding rock mass is relatively uniform and free of major geological disturbances or dissimilarities;
- the surrounding rock mass does not contain any extremely strong or extremely weak strata; and
- the mine workings are laid out on a regular pattern.

Where these conditions are not met, surface subsidence effects vary from those that would be predicted using the conventional model. Such subsidence behaviour and resultant effects are generally known as '*non-conventional*'. The following are the more common site specific variations to the conventional model of surface subsidence behaviour:

- massive overburden strata units
- pillar foundation (floor) settlement
- steep topography
- valleys and gorges (incl. steep slopes)
- regional far-field horizontal movement
- large scale geological features (dykes, faults etc)

Conventional surface subsidence effects and their impacts are well understood and are readily and reasonably predictable by a variety of established methods. The understanding of *non-conventional* surface subsidence effects (far-field horizontal movements, valley closure, upsidence and other topographical effects) is not as advanced. Valley closure and particularly upsidence are difficult to predict. However, there is a rapidly developing database of non-conventional surface subsidence impacts in the Southern Coalfield which is being used to develop improved prediction.

Non-Conventional Subsidence

Valley Closure

The mechanism(s) involved when the surface topography contains valleys, gorges or significant slope changes is one which is not fully understood, although there are a number of clearly significant contributing factors. The first of these relates to the role of a dominant pre-mining horizontal stress field. Mining causes further disruptions to this natural regional horizontal stress system because:

- it creates a void which then redirects horizontal stress into the roof and floor of the void. The effective height of the void is increased if fracturing and/or caving of the undermined strata occur. If a constrained zone exists above the mine workings, some of the horizontal stress will be redistributed through this zone. This increases the horizontal stress acting across the valley floor; and
- it removes or reduces the resistance to horizontal movement in the zone comprised of caved and fractured material, thereby permitting the surrounding rock mass to relax and to move towards the excavation.

Two responses arising from these mining-related stress behaviours are:

- *valley closure*, whereby the two sides of a valley move horizontally towards the valley centreline; and
- *uplift* of the valley floor, as a result of valley bulging and buckling and shearing of the valley floor and near surface strata. (This second behaviour is a direct and logical consequence of the first, in that if valley sides close, then at the base of the valley, the valley floor must be compressed, leading to buckling; which then gives rise to uplift).

The ground movements that occur around excavations in steeply incised terrain in a high horizontal stress environment are complex and it is difficult to identify the individual contribution of the various components to these movements, which include:

- conventional subsidence movements;
- elastic ground movements associated with redistribution of horizontal stress on a regional basis;
- movements associated with localised buckling and shear failure; and
- gravity-induced downhill slippage.

Some of these components may operate simultaneously in opposite senses. For example, an area may be subject to downwards vertical displacement at the same time that it is being subjected to upwards valley bulging.

Valley closure is not significantly influenced by the orientation of the valley relative to the mining layout or to the goaf. In the steep-sided Cataract and Nepean River Gorges, it has been found that the

closures in the sides of the gorges were almost mass movements with little differential shear displacement between different horizons in the strata. Closure at the base of the Cataract Gorge was some 86% of that at the top of the gorge.

Valley closure in excess of 460 mm has been measured at one site in the Cataract Gorge, with the maximum rate of closure occurring in that time between when the traveling longwall face passed beneath the gorge until the face was 200m past it. Closure was effectively completed by the time that the face was 500m past the gorge (Hebblewhite et al, 2000). Significant relative uplift also occurred at the site. Higher closures have been measured in steep valleys in the Western Coalfield but associated uplift levels have been considerably less. Valley shape has an influence on both closure and uplift. Whilst closure is also known to occur in wide flat valleys, such as exist in the Newcastle Coalfield, the magnitude of the closures is smaller, by comparison. Geology can also impact on closure, with higher closures typically occurring in shale strata. In some instances, valley closure can be followed by a degree of valley opening as the valley is then impacted upon by conventional subsidence displacements.

The direction of horizontal displacements associated with valley closures are understood to be associated with a combination of valley orientation (i.e. movement towards the valley centre); horizontal stress orientation; and also movement towards the direction of the triggering mining goaf region. These three directional factors can all play a part in both the mechanism and the direction of horizontal displacement and valley closure.

Upsidence

As indicated above, valley floor buckling then leads to uplift in the floor region. This introduces the concept of upsidence. The term *upsidence* has been used by subsidence engineers for some time to describe different types of upward vertical movement or uplift. In some instances it describes the absolute upward vertical movement of the surface at the edges of a region of subsidence influence, associated with massive strata cantilevering. However, the more common, and widely accepted current use of the term, is associated with the types of valley effect described above, where there is a component of relative upward movement, or uplift, created by the horizontal compression and buckling behaviour of the rock strata in the vicinity of the valley floor. It is in this sense that the term is used in relation to the Southern Coalfield Panel Report.

Upsidence is therefore defined as the relative upward displacement that occurs due to mining subsidence effects specifically associated with irregular surface topography such as valleys and gorges. Whereas mining subsidence of flat-lying surface topography normally produces nett downward vertical displacements over a region above and adjacent to mining; in the case of valleys, valley closure and valley floor buckling lead to a certain amount of uplift of the valley floor, superimposed on the conventional downward subsidence displacement. Upsidence is a measure of this relative uplift, compared to the conventional downwards displacement that would have been expected, had the terrain been relatively flat. Depending on the relative magnitudes of upsidence and conventional downward subsidence displacements, the absolute amount of vertical displacement in valley floor regions is normally still downward, but at a reduced level due to upsidence. However in some circumstances, the value of upsidence can exceed the conventional downward movement, leading to an absolute uplift of the valley floor. It is a phenomenon that is not confined to underground mining situations. Similar behaviour has been observed in civil and mining surface excavations when slots are excavated in material subjected to elevated horizontal stress.

Buckling and shear in the near-surface strata, which leads to upsidence, can also generate an extensive network of fractures and voids in the valley floor. Ground movements due to conventional subsidence can also contribute to the formation of this network if the upsidence occurs within the angle of draw of the mine workings. The formation of an upsidence fracture network has been monitored in detail at Waratah Rivulet (overlying longwall panels at Metropolitan Colliery) for a number of years using an array of surface and subsurface instrumentation (Mills, 2003; Mills and Huuskes, 2004; Mills, 2008). This has revealed that the network becomes deeper with the passage of each longwall in its vicinity. The main fracture network extends to a depth of about 12 m and bed separation extends to a depth of some 20 m (see Figure 1). In general, the extent and intensity of the fracture network increases with upsidence which, in turn, increases with subsidence. Figure 2 typifies the buckling, bed separation, extensive vertical fracturing and 'popping up' of slabs of rock observed by the Panel in Waratah Rivulet.

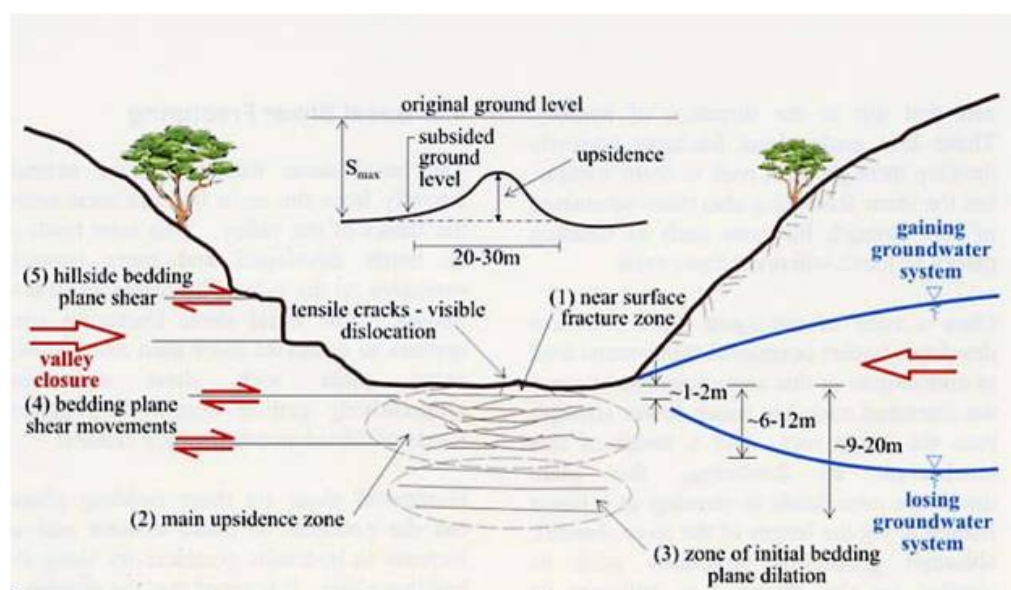


Figure1 - Upsidence fracture network determined from surface and subsurface monitoring, Waratah Rivulet at Metropolitan Colliery (source Mills, 2008)



Figure 2 - Buckling of near-surface, Waratah Rivulet, late 2004

Some significant observations regarding valley closure and upsidence are:

- both types of behaviour have been observed to occur up to several hundred metres beyond the conventional angle of draw, but at greatly reduced magnitude (including locations where the valley centres are not directly above the mining goaf);
- the movements develop incrementally with each panel extracted;
- incremental vertical subsidence leads to incremental upsidence and valley closure;
- both valley closure and upsidence are often greater in the presence of a headland; and
- the behaviours can also be associated with gently sloping valley systems and creek beds, albeit that the magnitudes of the closure and upsidence movements are less.

It is only in the last 15 to 20 years that the effects of underground mining on valley closure and upsidence, on a regional scale, have come to be widely recognised, particularly in the Southern

Coalfield where the nature of the surface topography leads to such effects. Whilst a conceptual fundamental understanding of the mechanisms which cause this type of behaviour has been developed, the detailed mechanism(s) and hence full extent of this type of behaviour requires further research.

Regional Far-Field Horizontal Displacements

In the last 20 years, mining induced, en-masse horizontal displacement of the surface has been detected in the Southern Coalfield for up to several kilometers from the limits of mining. These regional-scale movements are generally greatest at the goaf edge and decrease with increasing distance from the goaf. One of the first publications on the issue was by Reid (1998), who noted horizontal movements of some 25 mm up to 1.5 km from mine workings. Hebblewhite et al (2000) reported horizontal displacements in excess of 65 mm towards mine workings that were 680 m away (where mining was at a depth of approximately 450 m). These movements reduced to 60 mm at a distance of 1.5 km from the workings. Most of the horizontal movement takes place toward the gorges and active mining areas, although some has been recorded towards old goaf areas.

This behaviour is also not fully understood by subsidence engineers. A range of possible causes of valley closure, upsidence and far-field horizontal movements are under review. These causes include one or a combination of:

- simple elastic horizontal deformation of the strata within the exponential 'tail' of the subsidence profile that applies in conventional circumstances;
- influence of valleys and other topographical features which remove constraints to lateral movement and permit the overburden to move 'en masse' towards the goaf area, possibly sliding on underlying weak strata layers;
- unclamping of near-surface horizontal shear planes;
- influence of unusual geological strata which exhibit elasto-plastic or time dependent deformation;
- stress relaxation towards mining excavations;
- horizontal movements aligned with the principal *in-situ* compressive stress direction;
- valley notch stress concentrations;
- movements along regional joint sets and faults; and
- unclamping of regional geological plates.

It is important to note that where this type of far-field horizontal displacement has been detected to date, the levels of horizontal strain are very low. In other words, the differential horizontal movements over a particular length of surface are minimal. Consequently, there has been no evidence to date, of any significant adverse impacts on any natural features from this far-field behaviour. Nonetheless, the recognition of far-field horizontal movements is understood to have been the basis on which some community groups sought a protective buffer of 1 km between mining and rivers and significant streams.

Status of Prediction Capabilities (effects and impacts)

In consideration of the non-conventional subsidence effects and impacts discussed above, several points are worth discussing, arising from the review undertaken by the Panel. These include and relate to the following:

- the status of subsidence effects prediction capability;
- the status of subsidence impacts prediction capability;
- caution with the use of upsidence as a prediction parameter.

Prediction of subsidence effects

The Panel found that the status of subsidence effects prediction was well developed with respect to conventional subsidence behaviour, but less so in the case of some of the non-conventional behaviour discussed above. Having said that, there was a clear acknowledgement that there have been very significant gains in this type of prediction capacity over the last 10 to 15 years, particularly as a result of an extensive campaign of subsidence monitoring and related research by the mining industry – both collectively (through ACARP research programs) and by individual mining and consulting companies. This rapidly growing database of monitoring data has particularly encouraged the development of empirical prediction techniques. However, it is important to recognise the need for further

development in quantitative prediction capability, which will only occur with parallel development and research into a deeper understanding of the mechanism(s) involved in such forms of non-conventional subsidence.

Prediction of subsidence impacts

The Panel was critical of the current status of subsidence impact prediction capacity. This criticism was not because of a total lack of any such prediction, but because the current prediction in this regard lacked the quantitative rigour that might be expected in relation to particularly sensitive natural features such as particular valley floor rock bars, cliff lines, etc. Whilst current subsidence planning and management documents have again made great strides in this regard over the last decade, they remain more subjective or generic in regard to predicting subsidence impacts on significant natural features. There is of course good reason for this, since it is an extremely complex problem to deal with in any form of predictive model, where the number of variables and the actual variability of particular critical parameters can be extreme. However this should not detract from setting an objective to improve both the prediction capabilities in this regard, and the subsequent documentation of such predictions in any future mine planning submissions.

Caution in use of upsidence

A word of caution is required in the use of “upsidence” as a prediction or control parameter, relating to valley topography scenarios – for two reasons. Firstly, as defined above, upsidence is actually a relative parameter, rather than an absolute one. It relies on the measurement of absolute vertical displacement, but it then relies on the reduction of this figure by subtraction of the expected level of downward “conventional” vertical subsidence at the point in question. This immediately introduces an element of interpretation into the determination of upsidence – either in field monitoring, or in prediction. The second reason to express caution in the use of upsidence (at least in isolation) is that the actual manifestation of this phenomenon can vary enormously over short distances, depending on the individual layers of rock on the surface of the valley floor where it is measured. A far more reliable parameter to work with in relation to valley contributions to non-conventional subsidence is the actual horizontal valley closure. That is not to say that upsidence should be disregarded, but since it is a secondary effect of valley closure, then it should be used in this context, and not in isolation. Data presented to the Panel by the NSW Minerals Council (from MSEC) supported the view that there was a better correlation between actual and predicted behaviour for valley closure, than for valley floor upsidence.

Risk Management Zones

One of the key findings of the Panel was the fact that there was no scientific evidence or argument to support the view that an absolute “one size fits all” protective buffer region should be applied for protection of significant natural features from adverse subsidence impacts from underground mining. A number of community groups argued that a 1 km wide buffer, below which no mining was permitted, should be applied to all rivers in the Southern Coalfield. However the basis for either the 1 km figure, or the universal application of a single measure was not substantiated. For example, simple variations in depth, not to mention mining widths, seam thicknesses, and structural geology features all contribute to some degree of variability in the surface extent of any adverse subsidence impacts.

Furthermore, there was a clear misconception in the minds of some, in relation to the widely reported far-field horizontal displacement effects associated with the monitoring of the Lower Cataract and Nepean Gorges above Tower Colliery. The fact that horizontal displacements occurred at least 1 km away from the edge of mining (a subsidence effect) did not equate to an adverse subsidence impact (of which there was no evidence of any). In fact the greatest distance away from the edge of mining where any adverse impacts had been observed and reported across the Southern Coalfield was less than 400m.

Therefore, the approach taken by the Panel was to define a concept of Risk Management Zones (RMZs) in relation to a specific range and categories of natural features – not to prevent mining within such zones, but to require an increased level of prediction confidence and level of proof of such confidence (through previous case history back-analysis etc). The RMZ definition included an “angle from vertical” component, to recognise the role of depth, as well as a default minimum figure of 400m. It is expected that the definition and application of the RMZ will be the subject of ongoing review as the relevant databases and prediction techniques improve.

KEY RESEARCH RECOMMENDATIONS

The Southern Coalfield Panel Report made a number of recommendations for future research, including the following ones specifically relating to mine subsidence behaviour; prediction techniques; monitoring and remediation:

- The coal mining industry and Government should undertake additional research into the impacts of subsidence on both valley infill and headwater swamps. This research should focus on the resilience of swamps as functioning ecosystems, and the relative importance of mining-induced, climatic and other factors which may lead to swamp instability.
- The coal mining industry should undertake additional research into means of remediating stream bed cracking, including:
 - crack network identification and monitoring techniques;
 - all technical aspects of remediation, such as matters relating to environmental impacts of grouting operations and grout injection products, life spans of grouts, grouting beneath surfaces which cannot be accessed or disturbed, techniques for the remote placement of grout, achievement of a leak-proof seal and cosmetic treatments of surface expressions of cracks and grouting boreholes; and
- administrative aspects of remediation, in particular, procedures for ensuring the maintenance and security of grout seals in the long term.
- The coal mining industry should escalate research into the prediction of non-conventional subsidence effects in the Southern Coalfield and their impacts and consequences for significant natural features, particularly in respect of valley closure, upsidence and other topographic features.

ACKNOWLEDGEMENTS

This paper has made use of selected sections of the content of the Report prepared by the Southern Coalfield Panel. In so doing, the author acknowledges the significant contribution made by all Panel members to the preparation of the Panel Report, and the findings, conclusions and recommendations contained therein.

In drawing out and discussing some specific findings and issues in this paper, the views expressed (beyond the direct wording of the Panel Report) are solely those of the author of this paper and not necessarily of other Panel members, or the government Departments involved in commissioning the Panel of Inquiry.

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APPENDIX

Recommendations of Southern Coalfield Report

(The following is a direct extract from the published report of the Southern Coalfield Panel of Inquiry)

Assessment and Regulatory Processes

- 1) Risk Management Zones (RMZs) should be identified in order to focus assessment and management of potential impacts on significant natural features. RMZs are appropriate to manage all subsidence effects on significant natural features, but are particularly appropriate for non-conventional subsidence effects (especially **valley closure** and **upsidence**). Consequently, RMZs should be identified for all significant environmental features which are sensitive to valley closure and upsidence, including rivers, significant streams, significant cliff lines and valley infill swamps.
- 2) RMZs should be defined from the outside extremity of the surface feature, either by a 40° angle from the vertical down to the coal seam which is proposed to be extracted, or by a surface lateral distance of 400 m, whichever is the greater. RMZs should include the footprint of the feature itself and the area within the 40° angle (or the 400 m lateral distance) on each side of the feature.
- 3) RMZs for watercourses should be applied to all streams of 3rd order or above, in the Strahler stream classification. RMZs should also be developed for valley infill swamps not on a 3rd or higher order stream and for other areas of irregular or severe topography, such as major cliff lines and overhangs not directly associated with watercourses.
- 4) Environmental assessments for project applications lodged under Part 3A should be subject to the following improvements in the way in which they address subsidence effects, impacts and consequences:
 - a minimum of two years of baseline data, collected at an appropriate frequency and scale, should be provided for significant natural features, whether located within an RMZ or not;
 - identification and assessment of significance for all natural features located within 600 m of the edge of secondary extraction;
 - better distinction between subsidence effects, subsidence impacts and environmental consequences;
 - increased transparency, quantification and focus in describing anticipated subsidence impacts and consequences;
 - increased communication between subsidence engineers and specialists in ecology, hydrology, geomorphology, etc;
 - key aspects of the subsidence assessment (particularly in respect of predicted impacts on significant natural features and their consequences) should be subject to independent scientific peer review and/or use of expert opinion in the assessment process; and
 - increased use of net benefit reviews by both mining proponents and regulatory agencies in assessing applications.
- 5) Due to the extent of current knowledge gaps, a precautionary approach should be applied to the approval of mining which might unacceptably impact highly-significant natural features. The approvals process should require a 'reverse onus of proof' from the mining company before any mining is permitted which might unacceptably impact highly-significant natural features. Appropriate evidence should include a sensitivity analysis based on mining additional increments of 50 m towards the feature. If such mining is permitted because the risks are deemed acceptable, it should be subject to preparation and approval of a contingency plan to deal with the chance that predicted impacts are exceeded.

- 6) Approved mining within identified RMZs (and particularly in proximity to highly-significant natural features) should be subject to increased monitoring and assessment requirements which address subsidence effects, subsidence impacts and environmental consequences. The requirements should also address reporting procedures for back analysis and comparison of actual versus predicted effects and impacts, in order to review the accuracy and confidence levels of the prediction techniques used.
- 7) Part 3A of the *Environmental Planning & Assessment Act 1979* should be the primary approvals process used to set the envelope of acceptable subsidence impacts for underground coal mining projects. This envelope of acceptability should be expressed in clear conditions of approval which establish measurable performance standards against which environmental outcomes can be quantified. Once a project has approval under Part 3A, the Subsidence Management Plan approval should be restricted to detailed management which ensures that the risk of impacts remains within the envelope assessed and approved under Part 3A. In cases where a mining project approval under Part 3A of the EP&A Act does not yet exist, the SMP process should take a greater role in assessing and determining the acceptability of impacts.
- 8) The acceptability of impacts under Part 3A (and, in the interim, the SMP process) should be determined within a framework of risk-based decision-making, using a combination of environmental, economic and social values, risk assessment of potential environmental impacts, consultation with relevant stakeholders and consideration of sustainability issues.
- 9) Mining which might unacceptably impact highly-significant natural features should be subject to an increased security deposit sufficient to cover both anticipated rehabilitation costs (as at present), and potential rehabilitation costs in the event of non-approved impacts to the highly significant feature. The higher deposit should be commensurate with the nature and scale of the potential impact and should be attached to the mining lease by DPI under powers available to its Minister under the *Mining Act 1992*. If non-approved impacts occur and the feature is not able to be remediated by the mining company, then the deposit should be able to be forfeited as compensation for the loss of environmental amenity.
- 10) Consideration should be given to the increased use within Part 3A project approvals of conditions requiring environmental offsets to compensate for either predicted or non-predicted impacts on significant natural features, where such impacts are non-remediable.
- 11) Mining companies should ensure that they consult with key affected agencies as early as possible in the mine planning process, and consult with the community in accordance with applicable current industry and Government guidelines (eg NSW Minerals Council's *Community Engagement Handbook* and DoP's *Guidelines for Major Project Community Consultation*). For key agencies (eg DECC and SCA), this engagement should begin prior to the planning focus stage of a project application.
- 12) Government should provide improved guidance to both the mining industry and the community on significance and value for natural and other environmental features to inform company risk management processes, community expectations and Government approvals. This guidance should reflect the recognition that approved mining would be expected to have environmental impacts.

Subsidence Impact Management

- 13) The coal mining industry and Government should undertake additional research into the impacts of subsidence on both valley infill and headwater swamps. This research should focus on the resilience of swamps as functioning ecosystems, and the relative importance of mining-induced, climatic and other factors which may lead to swamp instability.
- 14) The coal mining industry should undertake additional research into means of remediating stream bed cracking, including:
 - crack network identification and monitoring techniques;
 - all technical aspects of remediation, such as matters relating to environmental impacts of grouting operations and grout injection products, life spans of grouts, grouting beneath surfaces which cannot be accessed or disturbed, techniques for the remote placement of

- grout, achievement of a leak-proof seal and cosmetic treatments of surface expressions of cracks and grouting boreholes; and
 - administrative aspects of remediation, in particular, procedures for ensuring the maintenance and security of grout seals in the long term.
- 15) Coal mining companies should develop and implement:
- approved contingency plans to manage unpredicted impacts on significant natural features; and
 - approved adaptive management strategies where geological disturbances or dissimilarities are recognised after approval but prior to extraction.
- 16) Government should review current control measures and procedures for approval and management of non-mining related impacts on Southern Coalfield natural features. These include various forms of discharge into rivers and streams, as well as water flow control practices. The impacts of such non-mining factors must be recognized when assessing the value of significant natural features in the region, and the assessment of appropriate control strategies.

Prediction of Subsidence Effects and Impacts

- 17) The coal mining industry should escalate research into the prediction of non-conventional subsidence effects in the Southern Coalfield and their impacts and consequences for significant natural features, particularly in respect of valley closure, upsidence and other topographic features.
- 18) Coal mining companies should place more emphasis on identifying local major geological disturbances or discontinuities (especially faults and dykes) which may lead to non-conventional subsidence effects, and on accurately predicting the resultant so-called 'anomalous' subsidence impacts.
- 19) In understanding and predicting impacts on valleys and their rivers and significant streams, coal mining companies should focus on the prediction of valley closure in addition to local upsidence. Until prediction methodologies for non-conventional subsidence are more precise and reliable, companies should continue to use an upper-bound, or conservative, approach in predicting valley closure.
- 20) Mining companies should incorporate a more extensive component of subsidence impact prediction with respect to natural features, in any future planning submissions. Such predictions should be accompanied by validation of the prediction methodology by use of back-analysis from previous predictions and monitoring data.

Environmental Baseline Data

- 21) Regulatory agencies should consider, together with the mining industry and other knowledge holders, opportunities to develop improved regional and cumulative data sets for the natural features of the Southern Coalfield, in particular, for aquatic communities, aquifers and groundwater resources.
- 22) Coal mining companies should provide a minimum of two years of baseline environmental data, collected at appropriate frequency and scale, to support any application under either Part 3A of the *Environmental Planning & Assessment Act 1979* or for approval of a Subsidence Management Plan.

ON MINING-INDUCED HORIZONTAL SHEAR DEFORMATIONS OF THE GROUND SURFACE

Gang Li¹, Robert Pâquet¹, Ray Ramage¹ and Phil Steuart¹

ABSTRACT: Horizontal shear deformations have not been commonly considered in subsidence engineering and risk management practices. This situation is quite different from many other engineering disciplines. This article presents the authors' initial findings of case studies from a number of collieries across all NSW Coalfields. The objective of this article is to highlight the significance of a ground deformation mode, that is, horizontal shear, and its implications to subsidence engineering and risk management. A Shear Index is suggested to facilitate studies of mining-induced shear deformations of the ground surface.

INTRODUCTION

This article presents an argument that conventional subsidence parameters specifying horizontal deformations, in particular, horizontal strains (i.e. change in length), are inadequate for subsidence engineering and risk management. The above-mentioned inadequacy can become practically important in areas where only low magnitude of conventionally defined horizontal strains is detectable due to deep cover depths (or relatively low "extraction width-to-cover depth" ratios).

There is clear evidence to suggest that the above-mentioned inadequacy is related to a lack of understanding of mining-induced horizontal deformations of the ground surface, in particular, horizontal shear deformations.

Despite theoretical definitions found in limited literature on mine subsidence (e.g. Peng, 1992), horizontal shear deformations have not been commonly considered in subsidence engineering and risk management practices. This situation is quite different from many other engineering disciplines.

HORIZONTAL SHEAR DEFORMATIONS

Indicators of horizontal shear deformations, as identified by this study, comprise:

1. Observed subsidence effects on civil structures indicating influence of shear deformations and significance of this deformation mode in terms of its impacts and frequency of occurrences. The shear effects at a particular site are demonstrated in Figure 1;



Figure 1 - Structures Affected by Horizontal Shear Deformations

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2. Statistical information suggesting a strong correlation between the shear-affected structures and strip footings, which have less capacity to resist or accommodate horizontal shear deformations as compared with that for other types of footings considered in this study;
3. Observed patterns of mining-induced surface fractures and deformations (in plan view) suggesting influence of shear, for example, i) en-echelon fractures near chain pillars where shear deformations were active or ii) occurrences of surface wrinkles where the effects of horizontal shear were clearly visible, and
4. Importantly, horizontal shear deformations of ground surface as indicated in 3D survey data obtained from a number of collieries across all NSW Coalfields (to be further discussed).

However, rigorous definition, in accordance with the principles of continuum mechanics (e.g. Jaeger, 1969), of horizontal shear strains is not possible using 3D survey data from a straight line of survey points.

It follows that if warranted considering the significance of the surface features and their capacity to resist or accommodate shear deformations, the current surveying practices may need to be changed to obtain properly defined horizontal shear strains (or principal strains).

To utilise the large amount of subsidence data in existence in the mining industry, an alternative (and approximate) Shear Index is suggested in order to gain an understanding of the general characteristics of mining-induced horizontal shear deformations. This Shear Index is derived based on the component of horizontal movements perpendicular to a survey line or a line of interest. The formula for deriving this index is the same as that for the conventionally defined tilt. Physically, this index reflects angular changes in the horizontal plane but it is not possible to tell what causes such changes, being either shear or rigid body rotation or both. However, the distribution pattern of this index can help to understand the development of shear deformations and to find "trouble spots" (refer to further discussions presented in the Section below).

FURTHER DISCUSSIONS ON HORIZONTAL SHEAR DEFORMATIONS

Figure 2 shows the distribution pattern of horizontal movements perpendicular to a survey line across a longwall panel and the corresponding Shear Index as discussed above.

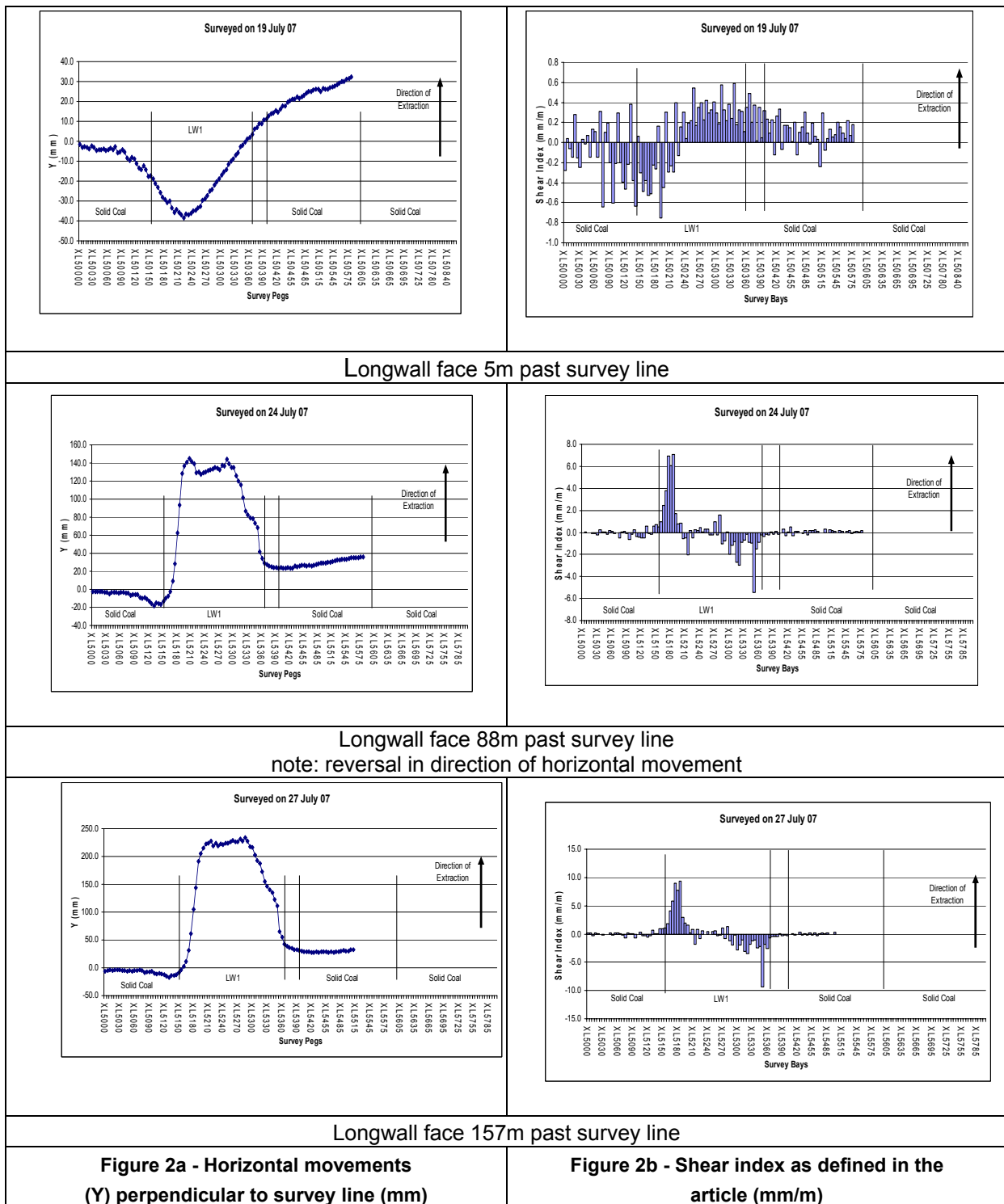
Although the site is located in the Hunter Coalfield with shallow cover depths, this case is selected as it provides a clear demonstration of the following observations common to the studied cases from all NSW Coalfields:

- A complex history of the horizontal movements perpendicular to the cross line (Figure 2a) involving a reversal of movement direction after the extraction face passed the survey site by a certain distance. This distance varied from site to site. Similar findings were reported by Holla and Thompson (1992) and Mills (2001);
- Indications of horizontal shear deformations (near both solid ribs in this case, as shown by the Shear Index plotted in Figure 2b), noting the reversal in the sense of shearing after the extraction face has passed the survey site. The reversal in the sense of shearing has a potential to enhance the effects of shear deformations, and
- The occurrences of permanent horizontal deformations.

IMPLICATIONS

From the 3D survey data collected from a number of collieries across all NSW Coalfields, the characteristics (i.e. the magnitude, nature, distribution and timing of occurrences) of the conventionally defined subsidence parameters are compared with those of the following horizontal deformational parameters:

- (i) Mining-induced horizontal movements perpendicular to survey grid lines, and
- (ii) The corresponding Shear Index as discussed above.



Implications from the findings of the current study so far are summarised as follows.

1. Horizontal Shear Deformations – There is a need to recognise horizontal shear deformation as a significant mode of mining-induced deformations at the ground surface. Specific attention should be paid to surface features with inadequate shear resistance and to areas with deep cover depths (or relatively low “extraction width-to-cover depth” ratios) where the conventionally defined horizontal strains predicted may suggest low risks.

2. Assessment of Subsidence Impacts on Civil Structures – Further to Point (1) above, there is a need to recognise the limitations of subsidence models based on conventionally defined horizontal strains and AS 2870-1996 (Standards Australia, 1996) when predicting subsidence impacts on civil structures. Consequently, there is a need to identify areas where changes and improvements to these models are required.
3. Civil Structures on Sloping Ground – Further to Point (1) above, specific attention should be paid to civil structures on the sloping ground. In this case, there is a potential for enhanced shear deformations due to the participation of down-slope movements. In addition, the performance of any footings to resist or accommodate shear deformations in this environment needs to be investigated and understood.
4. Capacity of Surface Features to Resist or Accommodate Shear Deformations – This is an area where knowledge has not been clearly established for subsidence engineering and management. The situation here, again, is different from many other engineering disciplines when shear deformations are concerned. There is a need to undertake necessary research into this area.
5. Mining-induced Surface Wrinkles – Mining-induced surface wrinkles (Figure 3), or compression humps, are one of the significant factors for subsidence impacts on civil structures. Where these deformational features occurred in areas with low predicted horizontal strains according to conventional subsidence models, geological structures were often blamed for their occurrences resulting in unpredicted or higher-than-predicted impacts on civil structures. However, recently conducted field investigations have not been able to provide a clear link between geological structures and such surface wrinkles, while there is a continuing need for an improved understanding of these features to develop effective early warning and risk management systems. The identification of horizontal shear deformations can offer an explanation (Figure 4), additional to geological structures and the conventionally defined compressive horizontal strains, for the occurrences of these deformational features.
- 6.



Figure 3 - Mining-induced surface wrinkles

6. Management of Subsidence-related Risks to Linear Infrastructure Items – The results of this study suggest a need to review the adequacy of risk management systems for important linear infrastructure items such as roads, rails, canals or pipelines, if these management systems have been developed based primarily on conventional subsidence models taking into consideration parameters predicted or measured along the lengths of such infrastructure items and/or if the features in questions do not have sufficient capacity to resist or accommodate lateral movements or shear deformations.
7. Survey Practices - As discussed above, to obtain properly defined shear strains or principal strains, the survey practices need to be changed. The suggested change is related primarily to the layout of survey grids, for example, 3D surveys of two (or multiple) parallel grid lines separated by an appropriately defined distance.

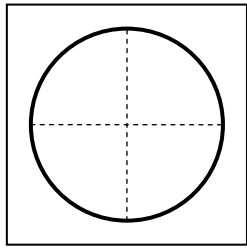


Figure 4a - Strain circle prior to shear deformation

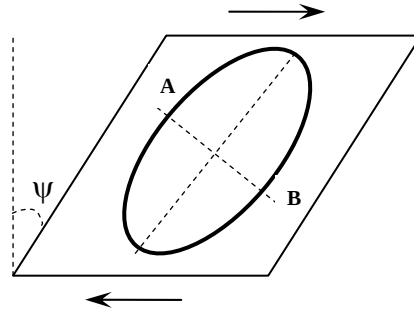


Figure 4b - Strain ellipse after shear deformation (After Ramsay, 1980). Surface wrinkles may occur due to shortening along the AB axis

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PREDICTION OF SURFACE SUBSIDENCE DUE TO INCLINED VERY SHALLOW COAL SEAM MINING USING FDM

Kourosh Shahriar¹, Sina Amoushahi² and Mohammad Arabzadeh³

ABSTRACT: Surface subsidence as an inevitable consequence of underground mining can cause problems for the environment and surface structures. Subsidence due to mining two shallow panels from an inclined coal seam C_1 of the Parvadeh (Tabas) coalfield, located in the eastern part of Iran, was predicted by finite difference method (FDM) using FLAC^{3D} software. The predicted subsidence profiles were compared favourably with both the measured values as well as the profile functions method. Using the parametric analysis, the position of maximum subsidence area was predicted over the panel rise side, which was completely in contrast with deep coal seam mining. The range of critical width to depth ratio (W/H) for both panels was determined between 1.0 and 1.4.

INTRODUCTION

Longwall mining of coal seams causes the formation of subsidence troughs which lead to a range of damages to the environment and surface structures. In order to protect the environment and structures from these damages, relatively accurate subsidence prediction is essential. The shape of subsidence trough due to horizontal coal seam mining is symmetric, whereas it is asymmetric for inclined ones. Most of the research on this subject has been validated for deep panels, while subsidence prediction for shallow and very shallow coal seams has not been given adequate attention. The position of maximum possible subsidence point (S_{max}) due to inclined deep seam mining shifts toward the panel dip side as illustrated in Figure 1 (Peng, 1992; Whittaker and Reddish, 1989).

Surface subsidence can be final (static), dynamic (progressive) and creep (delay or time-dependent). The final subsidence trough is that which exists long after the mining has been completed and its magnitude and shape are quite different from the dynamic subsidence trough formed during the face moving. For longwall coal mining, creep subsidence in fairly short time (4 to 12 months) will be completed and its magnitude is between three to five per cent of maximum subsidence. This period becomes even shorter with decreasing depth (Peng, 1992). In this study, creep subsidence will be neglected with a good approximation due to very shallow depth of objective panels (below 50 m).

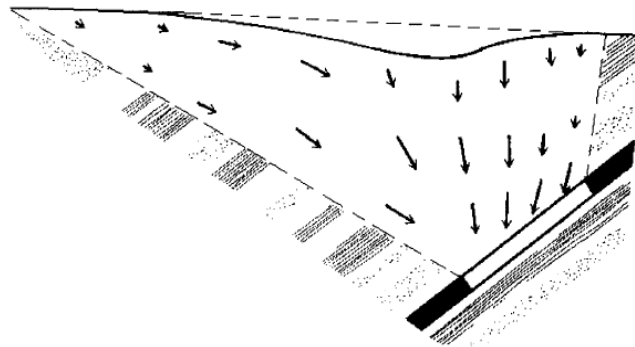


Figure 1 - Strata movements in inclined deep seam mining (Whittaker and Reddish, 1989)

There are three types of subsidence trough, ie; subcritical, critical and supercritical, depending on the width to depth ratio (W/H) of the opening. In subcritical conditions, subsidence does not reach to full development or maximum possible subsidence (S_{max}). When both the width and length of the opening have increased to critical conditions, subsidence reaches the maximum possible value. Thereafter, though both the width and length of the opening continue to increase, the maximum possible subsidence (S_{max}) does not increase, but spreads laterally into an area (Peng, 1992; Whittaker and Reddish, 1989).

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In this paper, a 3D numerical model of the panel No 28 located at Madanjou coal mine is developed using FLAC^{3D} code (Itasca, 2002) which is based on finite difference method (FDM). Then subsidence due to inclined shallow coal seam mining is predicted and compared to profile function developed by Asadi, Shahriar and Goshtasbi (2004) for this coalfield. The proposed numerical model is validated in another coal mine of Parvadeh coalfield (Negin coal mine).

SUBSIDENCE PREDICTION METHODS

Controlling measures in surface subsidence can be considered in three stages including prediction, prevention and protection. The accuracy of subsidence prediction greatly influences the effectiveness of preventative and protective measures (Afsari Nejad, 1999).

Subsidence prediction methods can be categorised into empirical methods (SEH graphical method, profile and influence functions), physical models and numerical methods (National Coal Board, 1975; Alejano, Ramirez-Orangemen and Taboada, 1999; Whittaker and Reddish, 1989).

Empirical methods are designed based on a large number of field measurements. Profile functions are based on a curve fitting procedure that uses a mathematical function to match the measured subsidence profile. When this mathematical function is established by use of actual field data then it can be used for the future prediction of surface subsidence in the mining area (Peng, 1992; Whittaker and Reddish, 1989). Asadi, Shahriar and Goshtasbi (2004) and Asadi et al (2005) developed some profile functions for Parvadeh coalfield (Table 1) which will be compared with numerical method obtained in this paper. Influence functions are based on superposition principle and are suitable only for supercritical conditions (Peng, 1992; Whittaker and Reddish, 1989).

Physical models are helpful for understanding the subsidence mechanism, but are not a good tool for estimating displacements (Alejano, Ramirez-Oyanguren and Taboada, 1999). Numerical methods are different from the other methods in that both the geological and geotechnical aspects of the mine working can be taken into account. Among numerical techniques, FDM is the most suitable method for solving highly nonlinear and large strain problems like subsidence phenomena. Therefore the code FLAC^{3D} which is based on FDM and explicit solution technique was chosen for simulating the subsidence in this study.

The application of numerical methods to real cases has to be accompanied by three processes: calibration of real data, validation and sensitivity analysis (Alvarez Fernandez et al, 2005).

MADANJOU COAL MINE

Madanjou coal mine is a part of Parvadeh 3 coalfield which is located at the south of Tabas city, Yazd province, Iran. Panel No 28 of Madanjou coal mine is selected in order to simulate the subsidence. Geometry and characteristics of this panel are shown at Table 2. Geological column and geomechanical properties of coal seam and surrounding strata are presented in Figure 2 and Table 3 respectively.

SUBSIDENCE SIMULATION

Modelling was carried out with FLAC^{3D} code (Itasca, 2002) which is based on finite difference method and it was performed in following five steps:

1. Determination of boundaries, material behaviour model and material properties.
2. Formation of the model geometry and meshing.
3. Determination of the boundary and initial conditions; Initial running of the program and monitoring of the model response.
4. Re-evaluation of the model and necessary modifications.
5. Interpretation of the results.

In order to avoid disturbance at boundaries and considering the face length of 60 m according to Table 2, a block with dimensions of x=350 m, y= 200 m and z= 160 m was selected as the initial geometry (Figure 3).

Table 1 - Profile functions developed for Parvadeh coalfields by Asadi, Shahriar and Goshtasbi (2004) and Asadi et al (2005)

Location	Developed profile function
Madanjou coal mine	$S(x) = -0.798 \left[ce^{-1.9 \left[\frac{-x}{19.15} \right]^{2.1}} + de^{-1.75 \left[\frac{x}{31.6} \right]^{2.2}} \right]$
Negin coal mine	$S(x) = -0.7457 \left[ce^{-8.8 \left[\frac{-x}{60} \right]^{2.17}} + de^{-7.4 \left[\frac{x}{100} \right]^{2.11}} \right]$

Table 2 - Geometry of panel No 28 in Madanjou coal mine

face length	60 m
dip angle of coal seam	20 °
dip side depth	28 m
rise side depth	7 m
average depth	17-18 m
extracted C ₁ seam height	1 m
mining method	unmechanised shortwall mining with caving
direction of mining	along the strike

It has been found that the elasto-plastic constitutive models are the most suitable ones for the simulation of surface subsidence (Peng, 1992; Whittaker and Reddish, 1989; Alejano, Ramirez-Oyanguren and Taboada, 1999; Afsari Nejad, 1999). The elastic models underestimate the maximum subsidence ($S_{\max}$) and mislead the position of maximum subsidence point. Therefore, the elasto-plastic Mohr-Coloumb behaviour model was chosen for simulating the surface subsidence. It is pointed out that the correct determination of $S_{\max}$ position is very important in inclined seams. In flat seams, the position of $S_{\max}$ locates over the panel center but in inclined deep seams, this point shifts toward dip side of the working (Whittaker and Reddish, 1989).

Table 3 - Laboratory properties of coal seam and surrounding rocks in Madanjou coal mine

Formation	Density (kg/m ³)	Poisson's ratio	Cohesion (MPa)	Internal friction angle (degree)	Modulus of elasticity (GPa)
coal seam	1500	0.26	0.4	22	0.7
roof sandstone	2700	0.32	5.1	38	4
roof siltstone	2700	0.31	2.1	30	2.2
floor sandstone	2700	0.31	3	35	3.6
floor siltstone	2700	0.31	1.2	28	1.6
floor shale	2000	0.26	0.5	25	0.8

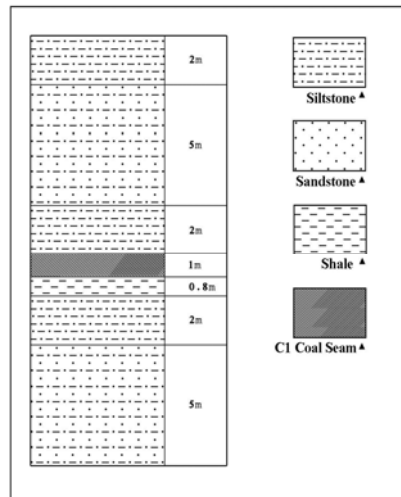


Figure 2 - Geological column at Madanjou coal mine

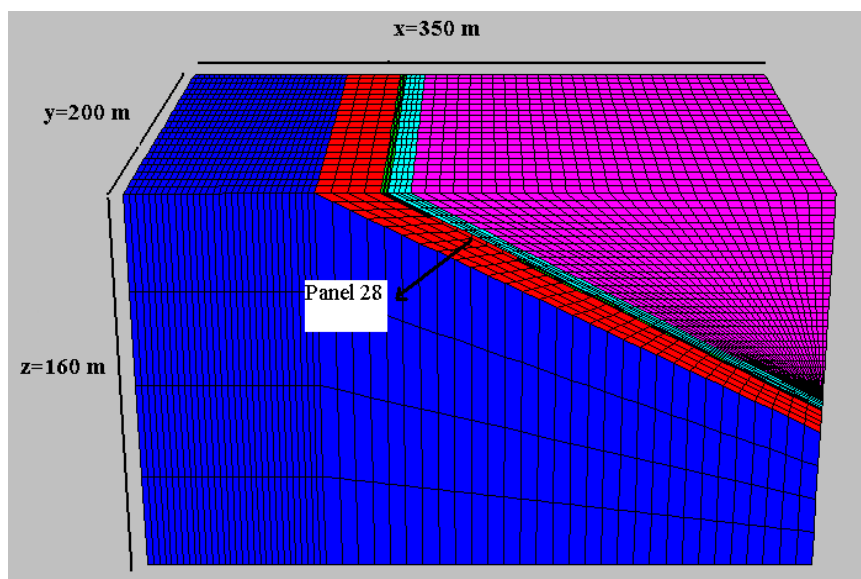


Figure 3 - Model boundaries

Assessment of input parameters

The results of numerical modelling are very sensitive to input parameters. Different methodologies are available in order to achieve them. The concept of reduction factor (RF) has been used successfully by several researchers especially in subsidence problems (Peng, 1992; Alejano, Ramirez-Oyanguren and Taboada, 1999 ; Afsari Nejad, 1999).

Different models are based on different assumptions and may account for different factors, so that rules for deriving parameters for one model may not be valid for another model. For example, one model may be purely elastic and use a Young's modulus that best reflects the rock failure that may occur. The rule to obtain this value from measurements would not be valid in a model that did account for rock failure. Thus, parameter selection for a model requires significant calibration work and experience with that model before there can be confidence in its prediction (Kelly, Luo and Craig, 2002).

Input parameters are classified into stiffness (deformability) and strength parameters. Deformability parameters consist of modulus of deformation (E) and Poisson's ratio. Experiences have shown that Poisson's ratio is little affected by size and does not change appreciably with rock mass scale effects. Therefore in this study, in situ magnitudes of this parameter are approximated equivalent to laboratory ones.

Research has revealed that shape and magnitude of the subsidence trough are strongly dependent on both Young's (E) and shear (G) moduli. Thus, in this analysis, characterisation is performed in two steps. The first one, based upon empirical relationships, allows one to estimate the values of parameters roughly. The second one requires a benchmarking numeric procedure to estimate the final values.

There are some common empirical formulae for estimating the rock mass deformation modulus (E_{RM}) from rock mass rating (RMR) and intact rock deformation modulus (E_i) which are shown in Table 4 (Alejano, Ramirez-Oyanguren and Taboada, 1999 ; Afsari Nejad, 1999; Sonmez et al, 2006).

Starting from an intact rock Young's modulus of 2.2 GPa up to 4 GPa for the immediate and main roof (Table 3), and having RMR=30, then Ramamurthy equations have better agreement in comparison with others. Alejano, Ramirez-Oyanguren and Taboada (1999) used these formulae successfully. Ramamurthy equations result in reduction factors of one-fifth and one-fifty second for horizontal and inclined stratification, respectively. Therefore the range of one-fifth up to one-fifty second is selected as the initial reduction factor of Young's modulus. After back analysis and benchmarking, reduction factor of one-twentieth is considered in order to achieve the *in situ* pre-failure Young's module; ie according to Table 3, $E_{RM} = 0.1$ GPa up to 0.2 GPa.

According to different studies, the shear modulus of a stratified rock mass must be a small value. For instance, Afsari Nejad (1999) used $G=E/15$, Alejano, Ramirez-Oyanguren and Taboada (1999) used $G=E/24$ and Yao, Reddish and Whittaker (1993) used a value somewhat smaller. In this study, G was measured $E/50$ after running several models and using back analysis.

The Mohr-Coloumb behaviour model is isotropic, while in fact coal measures are anisotropic bodies. Furthermore, due to bedding planes in the coal measures, the post failure values of shear modulus (G) decrease more than the modulus of elasticity and consequently, the bulk modulus (K). Obviously a unique reduction to shear and bulk modulus for derivation of post failure properties can not explain the anisotropic behaviour of rock materials (Lloyd, Mohammad and Reddish, 1997). Thus, two different reduction factors were applied to bulk and shear modulus. After running several models, reduction factors of one-tenth for bulk modulus and one-fiftieth for shear modulus gave the best results.

Initial stresses

From the information held on the world stress map project it can be concluded that the principal horizontal stress direction is likely to be in a north east-south west (NE-SW) direction at Tabas coalfield. No information was available on the magnitude of in situ stresses except that K (ratio of horizontal stresses to vertical stresses) is larger than one. Therefore sensitivity analysis was carried out in order to approximate the horizontal to vertical stress ratio ($K = \frac{\sigma_H}{\sigma_V}$) for this region.

It is found that $K=1.5$ and $K=2$ have similar trend with each other while for $K=2.5$, S_{max} reduces significantly and its position shifts to the panel center, besides uplift of surface becomes abnormal. Therefore K is considered 1.5 with a good approximation in model (Figure 4).

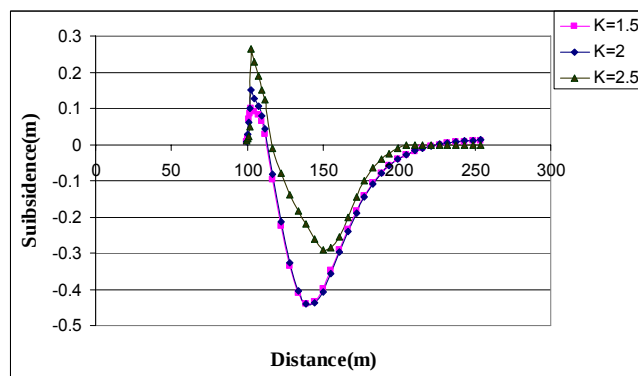


Figure 4 - Sensitivity analysis to initial stress

The similar results between $K=1.5$ and $K=2$ might be due to horizontal stresses in practice being anisotropic and maximum horizontal stresses are nearly 1.4 times the minimum horizontal stresses, but this issue is not considered in the model because of data deficiency.

Interpretation of the results

The program was run up to obtaining the final results which is shown in Figure 5. It is observed that the predicted limit angle by FDM over the rise side has a good agreement with measured limit angle (nearly 40°) while at the dip side, FDM predicts wider subsidence trough in comparison with measured one ie 57° vs 49° which is illustrated in Figure 6.

Table 4- Empirical equations suggested for estimating the rock mass modulus of deformation (Sonmez et al, 2006; Alejano, Ramirez-Oyanguren and Taboada, 1999)

Originator(s)	Required parameters	Limitations	Equation
Bieniawski(1978)	RMR	$RMR > 50$	$E_{RM} = 2RMR - 100 \text{ (GPa)}$
Serafim and Pereira(1983)	RMR	$RMR \leq 50$	$E_{RM} = 10^{\left(\frac{RMR-10}{40}\right)} \text{ (GPa)}$
Ramamurthy(1986)	E_i, RMR	horizontal stratification inclined stratification	$E_{RM} = E_i e^{(0.0217 RMR - 2.17)}$ $E_{RM} = E_i e^{(0.0564 RMR - 5.64)}$
Nicholson and Bieniawski(1990)	E_i, RMR	-----	$E_{RM} = E_i \left[0.0028 RMR^2 + 0.9 \exp\left(\frac{RMR}{22.82}\right) \right]$
Mitri et al(1994)	E_i, RMR	-----	$E_{RM} = E_i \left[0.5 \left(1 - \left(\cos\left(\frac{\pi \cdot RMR}{100}\right) \right) \right) \right]$
Sonmez et al(2006)	E_i, RMR	-----	$E_{RM} = E_i \times 10^{\left[\frac{(RMR-100)(100-RMR)}{400 \exp\left(\frac{-RMR}{100}\right)} \right]}$

Figure 7 compares predicted subsidence profiles by FDM, surveying and profile function method. The position of predicted S_{max} by FDM completely coincides with surveying and profile function method. Therefore in shallow workings like this case (average depth=17.5 m), the position of S_{max} shifts to rise side (shallower part) of the panel. This phenomenon is totally in contrast with deep seams in which point of S_{max} shifts toward dip side of the panel. From this point of view, Parvadeh (Tabas) coalfields are exceptional.

Furthermore, predicted S_{max} by FDM is nearly three per cent less than the predicted S_{max} by surveying and profile function. Actually FDM neglects residual subsidence so it underestimates S_{max} while the profile function predicts final subsidence basin.

Residual or time-dependent subsidence in this mine is roughly three per cent of maximum subsidence. On account of low depth, ground movements reach to the surface sooner than usual. Generally in longwall mining with caving, especially in shallow mines, the residual subsidence is almost negligible; vice versa in room and pillar method it has an outstanding role in creating the final subsidence profile (Peng, 1992).

Some uplift or upsidence (less than 10 cm) is created over the rise side and panel floor. It can be due to sequences of sandstone strata that behave like a beam in which one similar case has been reported in one of the Columbia's mines (Donnelly et al, 2001). In addition, due to very low depth of panel, cover load pressure may not be high enough for reconsolidation of gob material and accordingly uplift results. One of the advantages of FDM in comparison with profile function is its ability to figure the uplift at the surface or panel floor. No uplift is observed in measured profile provided by Asadi, Shahriar and Goshtasbi (2004) and Asadi et al (2005) because of their efforts were just focused on measuring downwards subsidence.

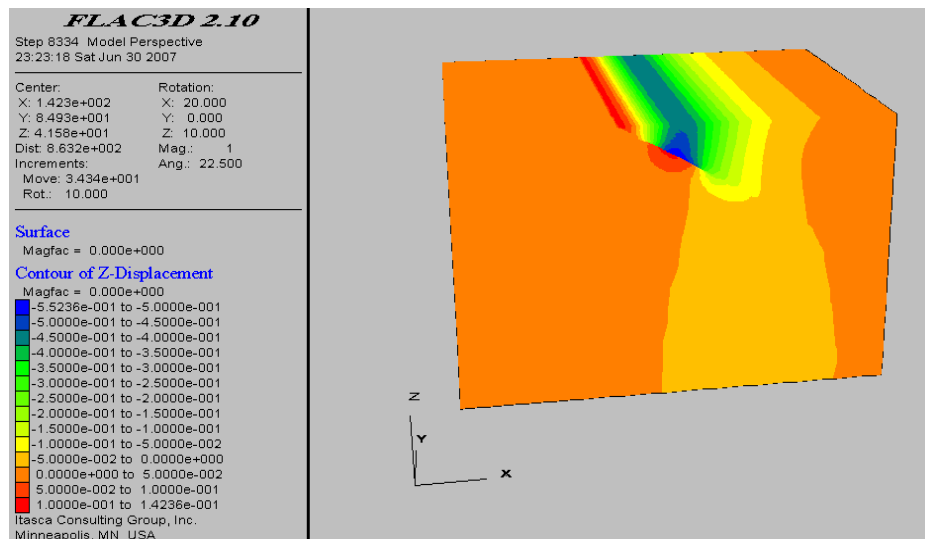


Figure 5 - Ground subsidence over panel No 28

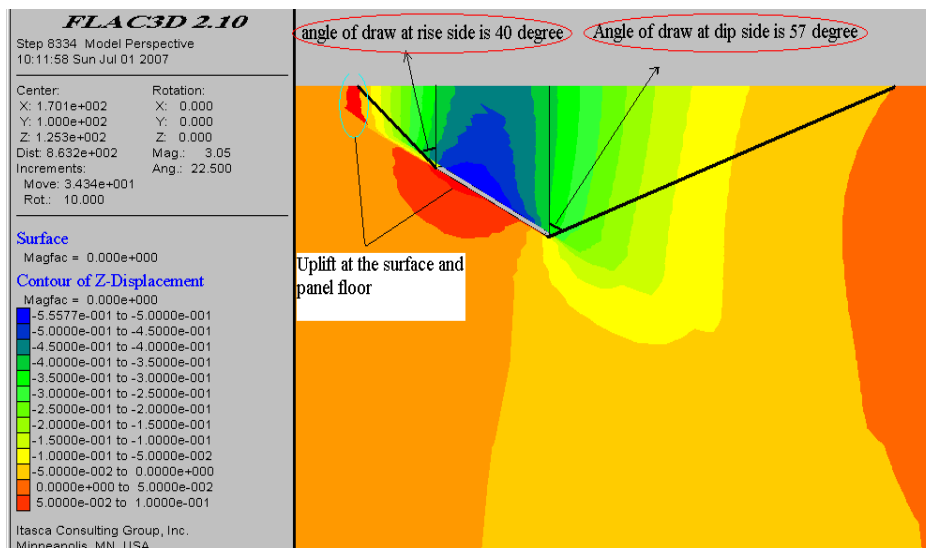


Figure 6 - Angles of draw at the sides of panel No 28 located at Madanjou coal mine

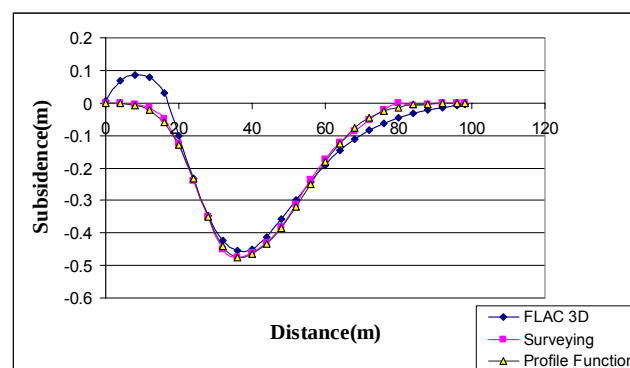


Figure 7 - Predicted subsidence profiles by FDM and profile function vs measured ones by surveying

VALIDATION

In order to ensure the reliability of the proposed numerical model, it has to be validated somewhere else in Parvadeh coalfield. For this purpose, Negin coal mine which is located north of Parvadeh 2 coalfield is selected. Geometry and characteristics of the simulated panel is shown in Table 5. Figure 8 shows the angles of draw at panel sides as well as flat bottom of subsidence trough due to supercritical dimensions of opening.

Figure 9 compares predicted subsidence profiles by FDM, surveying and profile function method. It is observed that similar to Madanjou coal mine, the predicted angle of draw at rise side has a good agreement with survey and profile function method. Conversely FDM predicts wider profile at the dip side. Furthermore FDM shows again some uplift at the surface which does not appear in surveying and profile function method. The position of $S_{\max}$ predicted by FDM has been shifted a little toward the rise side and does not coincide exactly with profile function method. It seems that for steeper coal seams the model has to be calibrated.

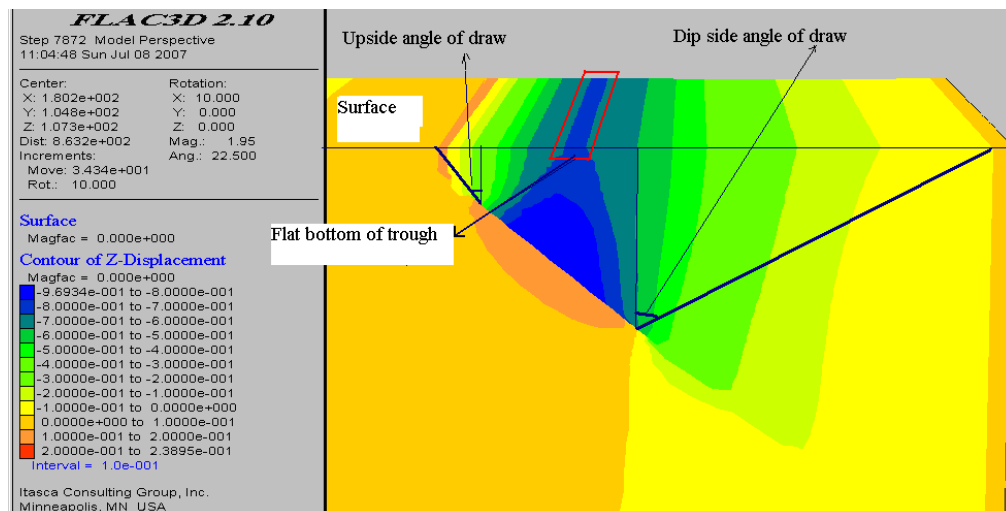


Figure 8 - Angles of draw at the sides of panel

SENSITIVITY ANALYSIS

In order to apply numerical models to real cases, in addition to calibration and validation, sensitivity analysis must be carried out on the parameters affecting the shape and magnitude of the subsidence trough (Alvarez Fernandez et al, 2005).

Sensitivity analysis to panel width

According to Figure 10, when the panel width is 8.5 m ($W/H = 0.85$), subsidence is about 200 mm and as the panel width increases to 17 m ($W/H = 1.47$) subsidence reaches to 440 mm. By increasing the width to 25.5 m ($W/H = 1.96$) it does not cause any increase in the $S_{\max}$ and just subsidence profile spreads laterally. It is concluded that critical width to depth ratio (W/H) is between 0.8 and 1.4. Furthermore increasing the panel width causes subsidence profile to be widen.

Sensitivity analysis to seam depth

Sensitivity analysis was done for three depths of 17 m ($W/H = 2.91$), 50 m ($W/H = 1$) and 64 m ($W/H = 0.77$) which is shown in Figure 11. It is observed that by increasing the depth, the ground surface uplift is reduced, and the subsidence profile becomes wider due to widening the area of influence. In addition, it is obvious that the critical width to depth ratio (W/H) is larger than one. Therefore according to results obtained from sensitivity analysis on depth and width of the panel, critical width to depth ratio range is between 1.0 and 1.4. According to Figures 10 and 12, subsidence due to mining of panels with similar W/H is equal.

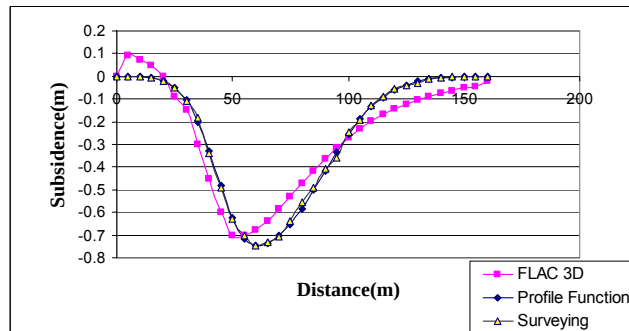


Figure 9 - Predicted subsidence profiles by FDM and profile function method vs surveying

Table 5 - Geometry and characteristics of the first panel of Negin coal mine

face length	90 m
dip angle of coal seam	30 °
dip side depth	62 m
rise side depth	17 m
average depth	40 m
extracted C ₁ seam height	1.7 m
mining method	unmechanised shortwall mining with caving
direction of mining	along the strike

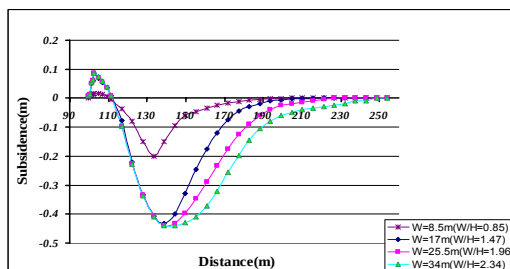


Figure 10 - Sensitivity analysis on panel width

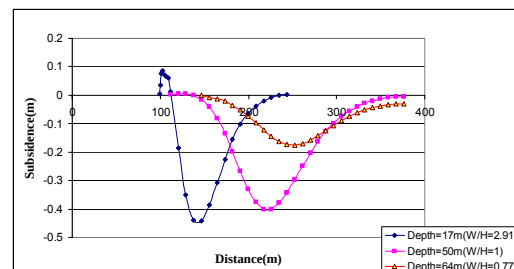


Figure 11 - Sensitivity analysis on seam depth

CONCLUSIONS

In this study, surface subsidence due to inclined very shallow coal seam mining of two underground coal mines in Parvadeh (Tabas) coalfield was simulated by FLAC^{3D} code which is based on finite difference method (FDM). FDM results were compared with measured profile and profile function method. FDM underestimated S_{max} up to three per cent in comparison with surveying and profile function. The reason is that the residual subsidence is neglected in this research but the profile function method predicts final subsidence trough. Furthermore in both cases, FDM in contrast with measured profiles obtained by surveying and profile function method, predicted uplift over the panels rise side at the surface in which was confirmed by local observations. The reason that no uplift was observed in measured profile provided by Asadi, Shahriar and Goshtasbi (2004) and Asadi et al (2005) was due to their efforts just have been focused on measuring downwards subsidence.

The Position of S_{max} in shallow coal seams shifted towards panel rise side which was totally in contrast with deep seam mining. Sensitivity analysis showed that by increasing the depth, this point gradually shifts toward the panel dip side. It was also found that critical width to depth ratio range is between 1.0 and 1.4 for both panels. This range is a little lower than the range of critical W/H ratio which has been found by National Coal Board of UK (1975). This might be related to very low depth situation of both panels.

Numerical methods can illustrate subsidence mechanism better than profile function due to taking into account the geomechanical material properties. Accordingly profile function results can hardly be extrapolated from one coal mining area to another, and even sometimes from panel to panel. Empirical methods have their own advantageous because of their simple and inexpensive applications.

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CURRENT STATUS AND FUTURE PROSPECTS OF MINING SUBSIDENCE AND GROUND CONTROL TECHNOLOGY IN CHINA*

Wenbing Guo¹, Youfeng Zou¹ and Yixin Liu¹

ABSTRACT: Mining of coal alters the ground stressfields, causing strata deformation, ultimately leading to surface subsidence. For the past 40 years some significant experience on mine subsidence control technology has been acquired in China, particularly when mining under buildings, railways and water bodies, which are known as "3-bodies". Current emphasis is in the evaluation of appropriate mining methods and other correlative technologies that can control or reduce the ground surface subsidence and protect surface structures. The current status of coal mining subsidence and ground control technology in China are discussed, including partial mining, backfilling, bed separation grouting, and harmonic mining. The partial mining methods include; strip pillar mining, the room and pillar method, and limited mining thickness. The backfilling mining method uses the traditional backfill material, like gypsum, paste-filling, and so on. Bed separation grouting in overburden strata is a new and patented technique which can reduce the surface subsidence to some extent, and is used in some Chinese coalmines.

INTRODUCTION

Coal is the most abundant energy resource in China, supplying about 70% of primary energy consumption (See Figure 1). China is the world's largest coal producer and produces nearly 35 percent of the world's annual coal production.

Due to a great deal of coal resources being extracted from underground, the environmental hazards resulted from mining activities are becoming a serious problem. Coal mining subsidence not only destroys the ecological environment, but also causes surface structure damage (See Figure 2). For example, in Yaojie Mining Bureau of Gansu Province, a large surface area subsided abruptly due to coal mining in 1993 causing fatalities.

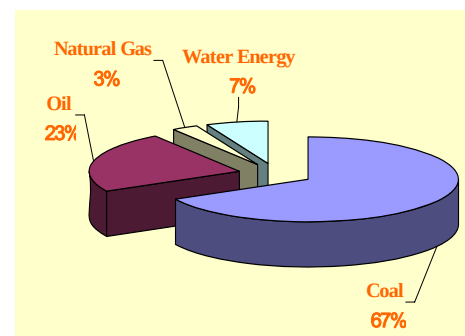


Figure 1 - Energy consumption in China

Reserves of coal under buildings, railway lines and water bodies (called "3- body") are huge in China. Based on the official data, the areas of subsidence prone land due to coal mining are about 600,000 ha. There are more than 1094 villages with 110,000 inhabitants residing in the area situated in the provinces of Henan, Hebei, Shandong, Anhui and Jiangsu. About 10 000 tonnes of coal is mined between 0.2-0.28 ha. In plain mining area with dense villages, about 2000 persons need to be relocated when 10,000,000 tonnes coal is extracted (Zhang Huaxing et al, 2000). At present, most coal mines that have coal reserves under 3-body conditions have serious problems with mining layout. Although moving villages before mining need not change mining methods, nevertheless, the compensation cost of moving villages away is continuously increasing. Also it is very difficult to find new fertile land to resettle villagers affected by mining subsidence

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In order to control mining subsidence and to protect surface structures, water bodies and railway lines from damages, it is necessary to research mining subsidence and ground control technology. Through several decades of studying and practice, China has accumulated an abundant experience on mining subsidence control technology. The objective is to study appropriate mining methods that reduce or control surface subsidence, and protect structures from damage.



Figure 2 - Farmland and structures damaged by mining

MINING SUBSIDENCE AND GROUND CONTROL TECHNOLOGY

The following underground mining techniques are practiced in China to control subsidence and prevent hazards.

Partial mining methods

Strip pillar mining method

Strip pillar mining is the most widely used method to control mining subsidence in China. In strip pillar mining the coal reserve is divided into regular strips with alternate strips being extracted. The strips left behind, called strip pillars, are designed to support the overburden and prevent surface subsidence. This is one of the important methods in "Green Mining Technology" and has become an effective method to mine those coal reserves lying under village structures (1,2,3,4,5 and 6) (Guo Wenbing, et al. 1998).

The advantage of strip pillar mining is to reduce the surface subsidence effectively without changing mining technology. The strip pillar mining method was first employed in 1976, and currently a large amount of coal reserves under "3-body" have been extracted by the strip pillar mining method. In China, the roof control method of strip pillar mining is almost by caving method. The mining depth of strip pillar mining is less than 500 m; mining height is mostly less than 6m, and the recovery ratio ranges between 40% and 68%. The surface subsidence factor depending on the recovery ratio is mostly less than 0.2, (See Table 1) (Guo Wenbing, Deng Kazhong, Zou Youfeng. 2004).

Table 1 - Relationship between surface subsidence factor and recovery ratio of strip pillar mining

recovery/%	hard stratum	medium-hard stratum	soft stratum
60	0.09□0.11	0.13□0.17	0.17□0.21
50	0.05□0.06	0.08□0.10	0.10□0.12
40	0.026□0.032	0.03□0.05	0.05□0.06

Room and pillar method

The room and pillar method is widely used in America, Australia, Canada, India, and South Africa. Its subsidence factor ranges between 0.35 and 0.68. This method has also been employed in recent years in China, but on a small scale. Huangling Coalmine of Shanxi province is the first coal mine to employ room and pillar mining using continuous miners. Other mines with room and pillar mining include those at Nantun coal mine of Yanzhou Mining Group and Daliuta mine of Dongsheng mining district.

Limiting thickness mining

Limiting thickness-mining method can reduce the effects of surface subsidence on the surface structures. It is rarely used, because its recovery ratio will be low if no surface structure damage is allowed. The permitted mining height (thickness) M is calculated by,

$$M \leq \frac{\varepsilon_y \cdot H}{1.52 \cdot b \cdot q \cdot \tan \beta}$$

Where: ε_y is the permitted surface horizontal strain; H -mining depth in meters; q - subsidence factor; b , horizontal movement coefficient; and $\tan \beta$ - tangent of major affected angle.

Backfilling mining methods*Traditional backfilling method*

The traditional backfilling method includes hydraulic backfilling, pneumatic backfilling, mechanical backfilling, and coal gangue sliding backfilling. In the process of mining, filling materials such as sands, coal gangue or fly ashes are filled in the gob behind the working face in order to support the overburden strata.

The hydraulic backfilling method is the most effective way to control surface subsidence. Its subsidence factor in general ranges from 0.1 to 0.3. When water supply is lacking or underground working face is damp or watery, the pneumatic backfilling method should be employed. Coal gangue sliding backfilling method is only used in steeply inclined and inclined coal seams; The subsidence factor of the coal gangue sliding filling method is roughly 0.3~0.4.

Table 2 shows different gob backfilling methods used in some coalmine, such as Fushun, Jiaohe, Jixi, Liaoyuan, Jiaozuo, and Huainan in China (Sui Huiquan, WANG Shaojun, Wang Hu. 2004).

Paste backfilling method

Backfilling materials and backfilling technology have made great progress in China in recent years. The paste backfilling method is a new method in China, and one of the key measures in "green mining technology". This method can increase the coal recovery ratio, protect groundwater and surface structures from damages, improve mining area environment and make use of solid waste materials. Jinchuan Company in China set up the paste backfilling production system in 1996. Now some coalmines in Shandong province are using this method.

Table 2 - Examples of backfilling mining method in China

No.	Mining method and backfilling method	Site	Coal condition	Protection object	Subsidence factor
1	Longwall mining along the dip with ascending hydraulic backfilling method	Shengli Coalmine, Fushun	Inclined very thick coal seams	Structures	0.1~0.22
2	Longwall mining along the strike with hydraulic backfilling method	Suncun Coalmine, Xinwen	Gently inclined medium thick coal seam	River (water)	0.15~0.2
3	Longwall mining along the strike with hydraulic and gangue backfilling	Wulin Shaft, Jiaohe	Gently inclined thick coal seam	Riceland	0.21
4	Longwall mining along the strike with pneumatic backfilling method	Yanmazhuang Coalmine, Jiaozuo	Gently inclined thick coal seam	Villages	0.3~0.4

After paste backfilling materials are filled in the gob, they become non-water-yielding material aggregate, and its solid constituents, commonly varies from 76% to 85% (Qian, Minggao, Xu, Jialin, Miu, Xiexing. 2003). The preparation process of paste filling is as follows: first, materials such as coal gangue, fly ash, industrial slag, bank sand are processed into hydrated paste like toothpaste. Then the toothpaste mixture is transported underground and pump filled into the gob. Once the paste fill is hardened and set in the gob, the cementation fill body can support overburden strata and control surface subsidence.

The main features of the past filling method are: the concentration of slurry made by solid waste material is high; the paste can segregate before setting. The paste doesn't pollute the underground working face and thus no drainage equipment is needed at the working face. Paste filling provides good support to Guo Wenbing, Deng Kazhong control overburden strata and surface subsidence, thus contributing to improvement in production and efficiency.

Grouting of bed separations in overburden strata

During mining, strata separations occur frequently in the overburden especially under strong and thick strata. Industrial waste materials such as fly ash are injected into the voids of strata separations under high pressure.

Introduction of bed separation in overburden strata

In recent years, a number of journal articles have published papers on ground subsidence by bed separations and grouting technology. Grouting of the bed separation caused by mining is achieved by injecting into fractured strata, a mixture of fly ash and industrial waste materials based slurry. The injection is carried out from surface boreholes as shown in Figures 3 and 4. The extent of the slurry injection depends on the structure and characteristics of overburden strata. In general, bed separations in overburden strata are formed under competent and thick strata (so called the key strata).

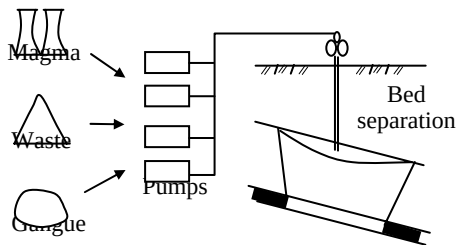


Figure 3 - Bed separation grouting technology under

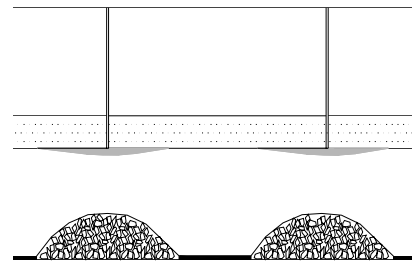


Figure 4 - Bed separation grouting the key strata

Mechanism of reducing subsidence by bed separation grouting

There are various reasons that bed separation grouting can reduce surface subsidence (Sui Huiquan, Wang Zhonglin. 2001). First, fly ash slurry material has better binding strength. When injected into the bed separation, fly ash can occupy the fractured space and slowly bind the separated layers, thus preventing the overburden strata from subsiding thus minimising surface subsidence. Thus grouting of the fractured weak layers will cement, reinforce, and improve the strength of strata.

In comparison with gob backfilling, bed separation grouting is conducted in upper overburden strata and will have less filling effect. Grout location should be in accordance with the following conditions.

- (1) Grouting slurry should not reach the working face from the overburden strata. Therefore, the grouting location must be above the caved zone caused by mining.
- (2) The depth from the surface of the grouted separation space must not be less than $0.4 \sim 0.6H$ (H is mining depth). Also for longwall mining, the length of the working face must be less than $0.3H$.
- (3) Where the overburden strata is strong and thick, the lower interface layer will be the best place for grouting.

Table 3 shows some examples of bed separation grouting in China. It can slow down surface subsidence to some degree (Zhao Deshen, Su Zhongjie, Sui Huiquan. 1998).

Table 3 - Examples of bed separation grouting in China

Mine	Seam thickness	Angle of coal seam	Mining depth	Subsidence factor
Laohutai	25.7m	26°	602m	0.278
Xuzhuan	2.6m	20°	529m	0.24
Dongtan	5.4m	3°	545m	0.032

Harmonic mining method

Harmonic mining is a technical measure to reduce surface deformation, but usually it can't reduce surface subsidence. During mining, adjusting the progress of two or more working faces and making structures to be located at the center of the subsidence trough, so that the structures are only subjected to dynamic deformation and final uniform subsidence. In China, Fengfeng mining group has made an experiment on extracting coal reserves under Sinsi village by harmonic mining technology. Seven longwall faces were simultaneously conducted, and coal reserves under the village was simultaneously extracted by two longwall faces. Because there was no gob edge under the village, the surface deformation in the village was reduced.

THE TREND OF GROUND CONTROL TECHNOLOGY

- (1) The paste backfilling method will be the developmental trend of strata and surface movement control technology and mining under "3-body". Paste backfilling is being studied both in the laboratory and field. It represents the development trend of subsidence control technology and mining "under 3-body". As one of the important measures of "green mining technology", it will become an important measure for protection of surface structures from damage. Although the paste backfilling has some disadvantages, this method is a favourite approach to realize "green mining". On the one hand, paste backfilling method can make use of solid waste material so that environmental pollution problem caused by solid waste material is solved to some degree. On the other hand, some problems of mining environment and safety can be solved to some extent. So paste backfilling technology using solid waste material represents the developing direction of strata and surface movement control and mining coal reserves under villages.
- (2) The strip pillar mining method can reduce surface subsidence effectively, and protect surface structures from damage. Since strip pillar mining is an important technical measure to control mining subsidence and to extract coal under "3-bodies", it will continue to be widely used in China's coal mines. At present, the research on deep-strip pillar mining is not enough. As the mining depth increases, it is increasingly urgent to research deep strip mining design and surface subsidence prediction. When the mining depth of strip pillar mining is more than 500m, there isn't a good method to select prediction parameters. As to multiple seams, the theory of surface movement and deformation design for strip pillar mining is still not perfect, so it results in larger differences between predictive results and field observation. Therefore, it is necessary to use new theories to study the mechanism and laws of mining subsidence.
- (3) There is a series for the difficulties of traditional backfilling method, for example, backfilling technology is complicated, the cost is higher, the coal production decreases, mining production is affected by the backfilling method. Therefore, this method is rarely used at present in China. Bed separated grouting technology can reduce the surface subsidence. But it still needs to be studied in many aspects. Now, its application is not extensive. Although the room and pillar method is employed in many foreign countries, the percentage using room and pillar mining method gradually decreases. Because there are some differences in geologic and mining conditions between China and foreign countries, room and pillar is rarely used. Harmonic mining method has many disadvantages, for example, it can only reduce surface deformation and can't reduce surface subsidence. Thus harmonic mining technology cannot become the main measure of subsidence and ground control, and therefore its application is limited in China.

CONCLUSIONS

China is a big country of abundant coal resources and coal production. But the problems of surface subsidence and environmental hazards due to coal mining are becoming increasingly serious. It is necessary to study mining subsidence and damage prevention technology. The current status of the coal mining subsidence and ground control technology in China are analysed and discussed in this paper. The measures to control subsidence and prevent damage in China mainly include: the partial mining method, the backfilling mining method, bed separation grouting in overburden strata and the harmonic mining method. The partial mining methods include the strip pillar mining method, the room and pillar method and the limited thickness (height) mining method. The traditional backfilling mining methods include hydraulic backfilling, pneumatic backfilling, mechanical backfilling, and coal gangue sliding filling.

The Strip pillar mining method is regarded as an important measure to control mining subsidence and to extract coal reserves "under 3-body". It is and will be used widely in China's coal mines. Paste backfilling technology and strip pillar mining technology are both very important measures. They are suitable for mining practice in China. They represent the development trend of damage control technology.

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USING HELIUM AS A TRACER GAS TO MEASURE VERTICAL OVERBURDEN CONDUCTIVITY ABOVE EXTRACTION PANELS

Yvette Heritage¹ and Winton Gale¹

ABSTRACT: The potential of helium injection into goaf and overburden strata as a tool to determine and measure goaf to surface connectivity is discussed. Laboratory studies investigated the flow mechanics and flow velocities of injecting helium through fractures by goaf injection technique and applied the laboratory findings to the field. From the laboratory studies, it was found that the mechanics of helium flow through fractures is by bubble flow. A relationship between gas velocity and fracture aperture was found allowing the determination of fracture conductivity through helium injection, which was comparable with previous works. Field trials of helium injection into the goaf were successfully conducted to determine whether a connection exists between the surface and the goaf. Helium injection process was carried out in two stages; the measurement of background helium, and injected helium. The average fracture aperture was determined from the arrival time of the first injected helium pulse, which takes the most direct path to the surface. The equivalent average conductivity was calculated from the average fracture aperture. Another technique of borehole helium injection was used to determine connection in the fracture network of the overburden. The borehole helium injection technique is a more direct approach of injecting helium into the fracture network of the overburden. With a borehole drilled into the highly permeable caved zone of the goaf, then borehole helium injection can demonstrate more quickly if a connection to the surface exists.

A repeatable technique of helium injection into the goaf or borehole has successfully been developed and demonstrated to prove connectivity between the goaf and surface of a longwall coal mine. These techniques will prove an effective tool for monitoring of environmental and hydrological problems.

INTRODUCTION

Longwall coal mining at shallow depths has many mining and environmental issues related to subsidence. The issues range from mining impacts caused by ventilation and spontaneous combustion to environmental impacts created from the release of goaf gas and hydrological issues. A primary cause of these impacts is subsidence which can form a connection between the goaf and surface, initiating these impacts. Determining the magnitude and extent of surface to goaf connection can assist in reducing the impact mining has on the environment and improve mining efficiency.

A connection is formed when the overburden subsides and new joints are created in addition to the reactivation of old joints. This opening up of joints creates a connected fracture network from the goaf to the surface which varies in tortuosity and conductivity. Creating a tool and method of determining connectivity and conductivity from surface to goaf would be invaluable to the coal mining industry.

This study aimed to assess the potential of helium injection into goaf and overburden strata as a tool to determine goaf to surface connectivity. Laboratory studies investigated the flow mechanics and flow velocities of helium through fractures. Field experiments trialled the helium goaf injection technique and applied the laboratory findings to the field.

BACKGROUND

The flow mechanics of gas and liquids has been extensively researched (Fourar and Bories, 1995; Viana et al, 2003; Odling et al, 1999; Liefer, Patro and Bowyer 2000; Sarkar, Toksoz and Burns, 2004; Ranjith, Choi and Fourar, 2006). Gas flow is described as laminar or bubble flow.

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Darcy flow only describes laminar flow however flow is not always laminar due to flow structures, pressure gradients and gas concentrations (Fourar and Bories, 1995). Non laminar flow, such as bubble flow, follows a different set of parameters. Bubble flow is traditionally discussed as gas flow in a liquid or solid in a liquid, but rarely gas flow in gas. Characteristics of this study however indicate bubble flow of gas in gas, helium in air.

The literature looks at helium and radon (which migrates similarly to helium) detection in soil gas samples in a variety of fields (Kristiansson and Malmqvist, 1982; Reddy et al, 2006; Ciotoli et al, 1999; Agarwal et al, 2006; Gascoyne and Wuschke, 1997; Lineham et al, 1996). This study is unlike the other studies as it involves the detection of helium directly from surface cracks formed from subsidence in longwall coal mining. The helium injection technique also differs from previous techniques as the helium is injected into the goaf and rises up under its own buoyancy through primary and secondary fracture networks to the surface.

HELIUM INJECTION INTO FRACTURED ROCK

To investigate the mechanics of helium flow to see whether helium flows through fractures as bubble or Darcy flow a laboratory experiment was constructed. The nature of the flow mechanics will affect the interpretation of the results.

The set up of the experiment involves an enclosed system including a helium injection chamber at the base, a vertically fractured core sample in the middle, and a helium collection chamber at the top. Core samples were cut and set at various apertures and placed in a mould. The mould is open at the base and the top of the core sample, sealed at the sides, with the fracture set on the vertical plane. The helium is injected into the basal chamber, where it rises with the same pressure differential each time and measured at the top of the sample.

Helium was detected at the top of the sample in intermittent bursts or pulses. The first pulse was smaller in concentration than the following pulses indicating the shortest travel distance, while the larger pulses that follow indicate the average travelling distance. The large pulses continue until the helium diminishes. This pulsed style of flow indicates bubble flow. Bubble flow for helium is supported by the literature. Subsequent analyses and calculations will be based on bubble flow mechanics.

The calculation of fracture aperture from the helium rise velocity provides a link between gas flow and water flow, as conductivity is calculated from the fracture aperture. Cored sandstone samples were cut vertically in half and separators were placed between the cored halves at different thicknesses. These artificially fractured rock samples represent natural vertical joints in which helium is to be injected into. The joint surface is not completely flat as the samples were not perfectly cut. Therefore the spaced apertures may not necessarily be represented by the separated thickness and may be larger than expected.

The experiment was set up to time how long it took the first pulse to travel the shortest distance through the sample. The average fracture aperture of the shortest path can then be calculated using both Stoke's Law and Davies and Taylor's rise velocity formula. Six samples, each with set apertures of approximately 0.05 mm, 0.1 mm, 0.15 mm, 0.2 mm, 0.25 mm and 0.5 mm, were injected with helium and the time of the first pulse recorded. This was repeated as many times as possible for each sample. All the samples yielded reasonably within the expected range.

The samples were also subjected to water flow tests where water flowed through the fractured samples under constant head. The aperture of the samples was calculated using the cubic law. The water tests were used to verify the set apertures and form a comparison for the helium results. The results are tabulated in Table 1.

The water tests show the aperture to be larger than the set aperture as expected. This is due to the extra space created between the rough fracture surface and the spacers. The helium method yielded smaller apertures than the water test results. This may be due to a factor of surface tension or static between the helium and the rock wall of the fracture that was not accounted for.

The set apertures calculated from the results of helium and water flow are plotted in Figure 1. For smaller apertures, Stoke's Law fits more closely with the set and water derived apertures, while the

Davies and Taylor equation fits more closely for larger apertures above 0.15 mm. This is supported by the literature where Stoke's Law is used for smaller apertures and Davies and Taylor is used for larger apertures.

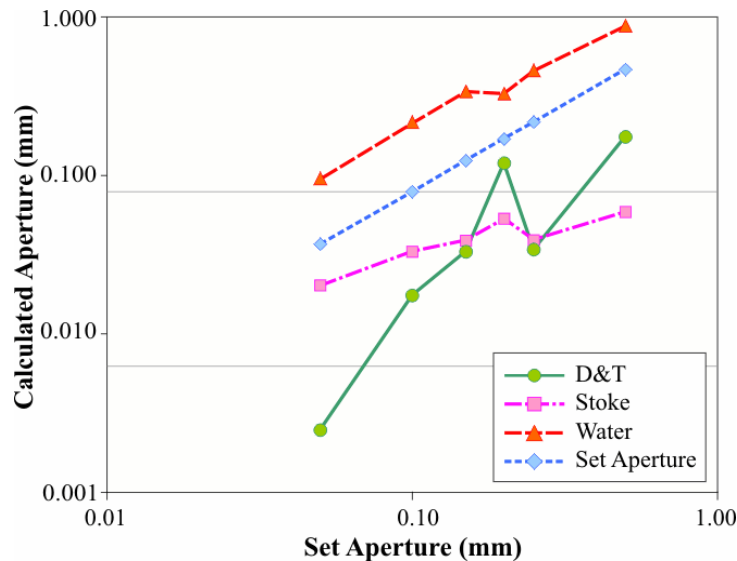


Figure 1 - Comparison of calculated apertures for Stoke's and Davies & Taylor's and water test methods.

The conductivity (K) of a fracture is calculated from the fracture aperture (e) using the cubic law. The kinematic viscosity (ν) is $1.01 \times 10^{-6} \text{ m}^2/\text{s}$ for water as follows (Indraratna and Ranjith, 2001):

$$K = \frac{ge^3}{12\nu b}$$

The calculated apertures, from the helium bubble flow and Darcy water flow, have been used to calculate the equivalent conductivities (Table 1). These calculations will be useful when injecting into the goaf to determine average conductivities of the overburden.

There are limited experiments worldwide that illustrate the flow rates of helium through fractured rock. Etiope and Martinelli (2002) have summarised examples of bubble flow through fractured rock which are represented in Figure 2. The examples include radon in igneous rocks (1) and helium bubble flow through low-permeability saturated faulted clays (2), medium-permeability clays (3) and high-permeability saturated faulted granite (4).

The flow per aperture has an upper boundary of Stoke's Law for small apertures, Davies and Taylor's equation for larger apertures and the terminal velocity of helium in air, as determined in this study. Factors that reduce the flow rate from the theoretical results include surface roughness, aperture variability, external stress and infill (Ranjith, 2000), which explains the trend of the experimental data. The data is also closer to the theoretical gas velocity for the larger apertures as the surface effects reduce with the increase in aperture width.

HELIUM GOAF INJECTION

This section looks at the injection of helium into the goaf of a longwall and the helium detection methods in surface cracks for the purpose of determining surface to goaf connectivity and characterising the overburden conductivity. It follows our chronology of determining the best technique for helium goaf injection and surface detection through field trials at Beltana and Ashton Mines.

Table 1 - Experimental results with aperture and conductivity calculations

Set Aperture mm	Sample Length m	Mean Time s	Rate m/s	Calculated Aperture mm			Calculated Conductivity m/s		
				Davies & Taylor	Stokes's Law	Water	Davies & Taylor	Stoke's Law	Water
air	1.14	7.5	0.152						
0.5	0.655	33.33	0.02	0.206	0.076	0.889	7.0E-06	3.6E-07	5.7E-04
0.25	0.58	62	0.009	0.047	0.053	0.495	8.2E-08	1.2E-07	9.8E-05
0.20	0.185	11.2	0.017	0.145	0.070	0.364	2.5E-06	2.8E-07	3.9E-05
0.15	0.245	26.6	0.009	0.045	0.052	0.375	7.5E-08	1.2E-07	4.3E-05
0.10	0.235	34	0.007	0.025	0.045	0.250	1.3E-08	7.5E-08	1.3E-05
0.05	0.205	72	0.003	0.004	0.029	0.119	6.5E-11	2.0E-08	1.4E-06

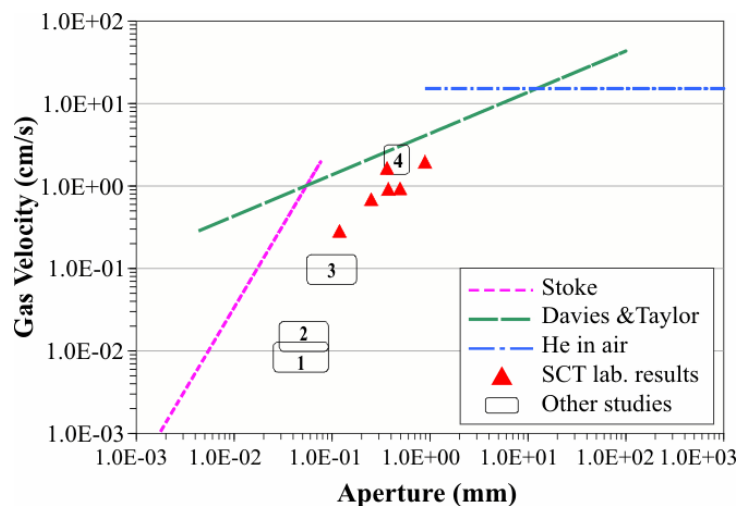


Figure 2 - Gas velocity versus aperture

Realistically, for helium to migrate from the goaf to the surface, the helium will travel through a network of fractures. The shallower the depth of cover over a mined seam, the more direct the fracture path is to the surface. Therefore the thicker the depth of cover, the more tortuous the flow path will be. There are other local factors that influence the tortuosity and aperture of the fractures, one of them being lithology.

The methodology for helium injection into the goaf is as follows. The goaf helium injection stage requires one person underground to release the gas and another person on the surface to monitor the surface cracks. To inject the helium into the goaf, tubing must be installed in the cut-through seal to the edge of the goaf, otherwise the helium may pocket in the cut-through. The helium is then released and the time of the release recorded. On the surface, the cracks are monitored for the injected helium. The time and concentration is recorded for each helium pulse. Detected helium results are compared with background helium results. A Uson Qualichek 1210 Leak Locator was used to detect the concentrations of helium. The detector has a remote sensor with a long nozzle to detect the helium emerging from the surface cracks.

Beltana Trial

Trials were conducted at Beltana No. 1 Mine in Singleton, where helium injections were carried out in the Whybrow Seam goaf in longwall Panels 6 and 7. The seam ranges in depth from approximately 50 m to 220 m. Subsidence cracks at Beltana form an arc from the maingate to the tailgate. This trial involved the injection of high purity helium into the goaf of the longwall panel where it would be monitored in surface subsidence cracks.

The geological section of the site consists of an interbedded sedimentary sequence of sandstone, siltstone, mudstone and coal with part of the sedimentary sequence overlain with sill. The current working seam is the Lower Whybrow.

Trials were conducted at three sites with depths of cover of 50 m, 105 m, and 220 m. During preliminary trials background helium was also observed emerging from the surface cracks. This created another element to determining whether a surface to goaf connection exists. The need for background helium to be characterised was now necessary to isolate injected helium from background helium. The background helium may not come from the mined seam but may come from smaller coal seams higher up in the sequence. The variability of whether there is background helium can also change daily with the change in air pressure. For this reason, background tests must be conducted on the day of the injection.

In the subsequent trial, the background and injected helium was detected from the surface cracks in pulses. The pulses indicated bubble flow while the variability of the pulses indicated a tortuous path. The flow mechanism resembled laboratory results.

50 m site - The surface cracks at this site were large and continuous and could be traced for at least 20 m, up to and over 50 m. The 50 m site was not tested for background helium. However large quantities of helium were detected 5 minutes after the injection. Background readings were not measured at this site; however, as the site is only 50m deep there are no major coal seams above the mined seam which would imply that any reading of background or injected helium would be sourced from the goaf.

105 m site - The subsidence cracks at the 105 m site were smaller than the 50 m site and less continuous on the surface. After helium injection, small pulses of helium were detected in a number of cracks which did and did not exhibit background helium. The cracks with no background helium and a helium injected result indicated a connection between the surface and the goaf. This situation is ideal as no background comparisons are needed. The low intensity of the peaks indicates that the average aperture is very small and only a small amount of helium migrated along this path.

The other scenario was cracks that had background helium results that needed to be compared with the injected helium results. As the background helium has the same pulse flow characteristics as the injected helium, a comparison of the rate and peak intensity is needed. The magnitude of background peaks was generally less than 30 sccm. After 12:58:40 pm a definite increase in intensity of helium concentration is observed with average peaks increasing to approximately 50 sccm. The helium was injected at approximately 12:30 pm indicating the injected helium reached the surface approximately 30 minutes after injection. The variability of average peak height two hours after injection indicates the migration path is a tortuous fracture network.

220 m site - The 220 m site showed cracks similar in size to those at the 105 m site. The crack continuity is about 1-10 m. The surface continuity at this site was reduced by surface features such as vineyards and roads. Background helium levels were observed at the 220 m site early on in the trial process and reduced dramatically over the period of the trial. In the final injection at this site most cracks monitored did not show a helium response except for one crack which noticeably showed a helium response with no measured background helium. This may indicate a surface to goaf connection. The peak intensity and frequency similarly to the 105m site again indicating a small average fracture aperture with a finite volume of helium injected into the fracture system. Other cracks showed a small helium response during background monitoring but did not show any response after helium goaf injection. This shows the variability of helium detection at this site and may indicate that the measured post injection helium response described above may not be indicative of a surface to goaf connection.

The 105 m site was used to determine fracture aperture and conductivity. The time taken for the injected helium to reach the surface at the 105 m overburden site was 30 minutes. At this overburden thickness the caved zone is approximately 50 m, resulting in 55 m of upper strata with reduced flow velocities. Subtracting the time and distance of free helium flow allows the calculation of average velocity which in turn determines the average aperture. For this instance, the helium flowed at 150 mm/s for 50 m which equates to 5.5 minutes. The remaining time (30-5.5 minutes) of 24.5 minutes is the time it takes to travel the last 55m. This equates to a flow rate of 4cm/s assuming a direct path. Extrapolating the gas velocity to the aperture in the bubble migration graph illustrated in Figure 2 we get an aperture of approximately 1mm. It is unlikely that the flow path is direct so an element of tortuosity would increase the distance of flow and the flow rate. Therefore the calculated aperture is a minimum aperture.

Conductivity is calculated from the aperture of a fracture. The calculated conductivity is an average conductivity throughout the overburden. For the 105 m site above, the aperture was calculated at a minimum of 1mm for the non caved zone. The conductivity formula for an individual fracture using the cubic law is as follows (Indraratna & Ranjith, 2001):

$$K_f = \frac{ge^3}{12vb}$$

Where the hydraulic conductivity of a single fracture is K (m/s), g is gravity (m^2/s), aperture is e (m), kinematic viscosity (v) is $1.01 \times 10^{-6} m^2/s$ for water and b is the spacing between fractures (m). An aperture of 1mm, using this formula and assuming 1 fracture in a 1m cube, equates to a hydraulic conductivity of $8 \times 10^{-4} m/s$ (assuming water as the fluid). Therefore the average hydraulic conductivity of the non caved zone above the goaf at 105 m overburden is greater than $8 \times 10^{-4} m/s$.

The advantage of using the helium injection technique to estimate hydraulic conductivity is that it focuses on the vertical conductivity as opposed to other methods, such as borehole packer testing, that focus on horizontal conductivity.

From the Beltana helium injection trials, a technique for injecting helium into the goaf and detecting the helium in surface cracks was developed. The trials revealed a limitation where background helium from both the working seam and seams in the overburden was emerging from the surface cracks. To overcome this background helium detection, both pre and post helium injection levels were required to be monitored. If the background levels are too high then it is difficult to distinguish the injected helium from the background helium.

Ashton Trial

After field trials at Beltana, the technique was tested at another site. Further injections took place at Ashton Underground Mine to determine whether a surface to goaf connection existed in Longwall 1. Injections were conducted at 95 m and 75 m overburden thicknesses.

The geological section at Ashton consists of an interbedded sedimentary sequence with numerous coal seams. The current working seam is the Pikes Gully Seam which in Longwall 1 ranges from 40 m to 95 m deep. The surface cracks at the 95m overburden site were up to 0.3 m wide with vertical continuity of at least 5-7 metres. The cracks at this site were large due to the influence of the sloping topography. After performing background checks, helium was injected into the goaf at 21 Cut-through. No helium was detected in surface cracks at this site. The cracks at the 75 m overburden site were much smaller with a maximum of only approximately 0.05 m wide. After performing background helium checks, helium was injected into the goaf at 17 Cut-through. No helium was detected in surface cracks at this site.

Helium may not have reached the surface for two reasons: i) the gas may have risen into a pocket with no surface connection, and not reached the cracks that do have a connection, or ii) there is no surface to goaf connection. Therefore, this method of helium injection does not confirm that there is a surface to goaf connection however it does not mean that a connection does not exist.

In the two trials, a method of helium injection and detection has successfully been used to determine whether a surface to goaf connection exists. A repeatable technique of helium injection into longwall goaf was developed to determine whether a surface to goaf connection exists (See ACARP Report C15010).

BOREHOLE HELIUM INJECTION

Borehole helium injection was trialled in addition to helium injection into the goaf. This method was created to reduce the limiting factors found using goaf injection, such as the inability to direct the helium into the continuous vertical cracks in the goaf. This method using borehole helium injection was trialled at Ashton Mine after goaf helium injection observed a null result. A borehole was drilled into the goaf of a longwall panel. Subsidence cracks on the surface adjacent to the borehole were the targets for detecting the helium.

The lithology of the overburden consisted of interbedded sandstone, siltstone and coal. The mined Pikes Gully Seam sat approximately 90m below the surface. The 50 m borehole was drilled to approximately 40 m above the mined seam. Difficulty in retaining water return during drilling in addition to attempted packer testing indicated a highly conductive fracture network around the borehole.

The method for borehole helium injection is as follows. The intervals were defined with inflatable packers at the top and the base of the borehole. Helium was injected into the strata at 500 kPa. The helium was injected into the packed off interval and injected horizontally into the strata. The injected pressure quickly decays from the borehole centre. Then due to its buoyant nature, the helium rises up to the surface via the interconnected fracture network.

Background tests conducted on the two tested surface cracks detected zero helium. All three tests at 20 m, 30 m, and 50 m recorded a connection with pulses of helium observed up to 6 minutes apart however mostly about 1 minute apart (Table 2).

Table 2 - Ashton borehole helium injection results

Depth (m)		Time Between Pulses (mins)		
From	To	Min.	Max.	Mean
18.32	21.56	~1	~1	~1
30.00	31.31	1	6	~1
49.4	50.6	0.2	1.7	~0.5

The conductivity calculated from the rate of helium injected into the borehole was found to be at least 1×10^{-5} m/s, similar to that of the packer testing results. However, once the applied pressure was turned off, the pressure down the borehole dropped off immediately indicating that the down-hole pressure was much less than the injection pressure. This is due to the helium in the hole discharging faster than it was recharging, due to resistance in the inflation tube. This may indicate a much higher conductivity than calculated.

The rate of helium flow to the surface indicates a much higher conductivity than the volume injected into the strata. At the 50 m deep site, helium took 75 s to reach the surface, which equates to a rate of approximately 650 mm/s. Referring to the gas velocity vs. aperture graph of other test results (Figure 2), the gas velocity in this study corresponds to free flowing gas with apertures greater than 5mm. The conductivity of a fracture with 5 mm aperture equates to 1×10^{-1} m/s, which is much higher than the

conductivities calculated from the water and helium volume flow rates due to not measuring the exact down hole pressure.

The results of the borehole injection test indicated that the overburden above the test locations was conductive and indicated that the helium injected into the goaf at the two underground sites may not have had a clear pathway to the fractured overburden above the monitoring sites. The borehole injection method did not preclude the presence of an impermeable layer below the borehole that could explain the lack of helium migration from the goaf injection to the surface.

The borehole helium injection technique is a more direct approach of injecting helium into the fracture network of the overburden. With a borehole drilled into the highly permeable caved zone of the goaf, then borehole helium injection can demonstrate more quickly if a connection to the surface exists.

An advantage of the borehole helium injection is that the helium is injected directly into the fracture network as opposed to the injection of helium into the goaf which does not guarantee that the helium will rise into the fractures that have a connection to the surface. Another advantage is the time taken to monitor the cracks during borehole injection is only a matter of minutes rather than the hours it takes to monitor the goaf injection.

The borehole helium injection can be used in conjunction with packer testing to parallel the results. Packer testing shows the characteristics of lateral conductivity around the borehole while helium injection shows the characteristics of vertical conductivity adjacent to the borehole.

LIMITING FACTORS

A few limiting factors were found during this study. The limiting factors may affect the results of the test by inhibiting flow or detection of the helium.

Helium is found in goaf gas at some mines and can be detected prior to helium injection. If the background helium levels are too high then it is difficult to differentiate the injected helium from the background helium. However, as helium dissipates with time, the test can be delayed until the background helium levels are low enough.

Rain can cause a direct problem by increasing the moisture content in the soil which in turn expands the soil resulting in the reduction of crack aperture. Rain can indirectly restrict flow through small cracks by silting them up. Therefore, if the test cracks are small, it is ideal to test before rain events if possible. However if this is the case, then it is advantageous for the mine.

CONCLUSION

From laboratory experiments it was found that the mechanics of helium flow through fractures is by bubble flow. A relationship between gas velocity and fracture aperture was found allowing the determination of fracture conductivity through helium injection, which was comparable with previous works.

Field trials of helium injection into the goaf were successfully conducted to determine whether a connection exists between the surface and the goaf. The determination of whether a surface to goaf connection exists requires a two stage process consisting of the measurement of background helium and injected helium. The average fracture aperture is determined from the arrival time of the first injected helium pulse which takes the most direct path to the surface. From the average aperture an equivalent average conductivity can be calculated.

Another technique of borehole helium injection was used to determine connection in the fracture network of the overburden. The borehole helium injection technique is a more direct approach of injecting helium into the fracture network of the overburden. With a borehole drilled into the highly permeable caved zone of the goaf, then borehole helium injection can demonstrate more quickly if a connection to the surface exists.

A repeatable technique of helium injection into the goaf or borehole has successfully been developed and demonstrated to prove connectivity between the goaf and surface of a longwall coal mine. These techniques will prove an effective tool for monitoring of environmental and hydrological problems.

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MOVING THE VENTILATION REPORT INTO THE 21ST CENTURY

John Rowland¹

ABSTRACT: Over the last three decades there has been significant technological change in both coal mine monitoring and ventilation modelling capabilities in the underground coal industry. During this time there have been substantial changes in statutory ventilation monitoring requirements to reflect such changes in our industry's monitoring capability. Somewhat contrastingly, there has been very little change in the mandatory regulatory requirements relating to ventilation quantity and gas contamination readings that need to be taken on a monthly basis. On top of that there has been no official mention made of any requirements to model or predict ventilation circuit changes in the mine office, before attempting such procedures underground.

Using the NSW coal mines regulations as the basis for the opinion it is the writer's belief that whilst the skill level requirements of minesite ventilation officers has been substantially elevated over the last quarter of a century there has been no step change in the mandatory output requirement of the monthly statutory ventilation survey and accompanying report.

In the 20th century it was not uncommon for the mine ventilation survey to comprise a handful of readings which were then entered into a pro-forma book and then hidden away until the next survey. The data measured was insufficient to accurately delineate the circuit and little or no thought was given to utilising the data to build a mimicked model of the ventilation circuit. Unfortunately in the 21st century there has been insufficient regulatory change to date to ensure such basic practices are elevated to a higher level.

Current ventilation modelling software programs are very user friendly and relatively simple to maintain and update but even the most recent changes in the regulations have all but ignored the value of such tools.

With some careful planning the ventilation surveyor should set up both his quantity and pressure survey stations in a manner which will afford not only statutory compliance but allow him to accurately define the ventilation circuit to a level that will allow him to maintain an accurate ventilation model of the circuit. The maintenance of such a tool will afford far safer change procedures relating to ongoing circuit adjustments.

The final ventilation report that is assembled should be a useful tool and ideally communicated to the frontline supervisors to assist them with their ongoing understanding of the circuit in which they work.

It is hoped that some of the tips contained in this paper relating to report inclusions and representation of such data may help at least some ventilation officers streamline their monthly data collection/collation process and with a minimum amount of work. The output is an accurate snapshot of the circuit to utilise for model validation that should double as a useful training tool.

The Australian coal industry has an outstanding safety reputation and we must ensure we properly utilise all available technologies to keep it that way.

INTRODUCTION

There have been significant changes in technology over the last 30 years along with an elevation in the educational level and importance placed on the role of the Ventilation Officer (VO) in underground coal mines in Australia, and particularly in NSW. From a time in the 1970's when a ventilation model conjured images of the scaled down steel ducting in the ventilation laboratory at Wollongong TAFE to the remarkable tool it is known as today there has been relatively little change in the prescribed routine duties of minesite ventilation officers.

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Even though modern regulatory theorists espouse the virtues of self regulation and are admitting to be moving away from specified or prescribed legislation, the current NSW regulations still prescribe minimum requirements relating to the regular collection of air quantity and gas determinations around the mine.

Unlike the self regulation expected relating to the use of auxiliary fans underground which was all but left out of the 1999 NSW underground regulations, the locations and frequencies that air and gas determinations must be taken are still prescribed by legislation. Any preference towards prescribed or unprescribed legislation will probably be debated ad infinitum but the fact remains that the ventilation regulations relating to physical duties carried out by the ventilation officers have changed little over the last three decades. Also, if only the minimum prescribed readings are taken as detailed in the regulations, then insufficient information will have been collected to foster any real intimacy with the operating circuit.

The level of knowledge and skill of the current incumbent group of ventilation officers however, is thankfully far higher than in times past even if the specific regulations demand little more from them regarding a ventilation survey and report, than it did in the 1970's. Fortunately most ventilation officers have moved with the times and have a far greater level of understanding and control over their ventilation circuits than in the old days when it was basically an add on job, and the standard of the average modern ventilation report reflects that.

There are still however, a number of sites without operating or active ventilation models with which to carry out predictive analysis of circuit changes and the law makes no mandatory prescription in this regard. The regular maintenance of a ventilation model though is a relatively simple task given some careful planning about the data required to be collected on a monthly basis. As such the maintenance of a validated model should be encouraged as a high priority.

It is a fact that with some key preparation in the office, most of the hard work is done by the Excel spreadsheet, and the underground duties required to collect the data should be well in reach of even the busiest of ventilation officers.

The writer has hit some hurdles and obstacles over the last decade whilst measuring and modelling a myriad of ventilation circuits. Many lessons have been learned during this time and the suggested paper inclusions and techniques to assist with circuit delineation detailed herein will hopefully be of some assistance, at least to the more uninitiated.

Without doubt one of the most valuable learning's however is not assuming that front line supervisors either won't understand or don't want to learn more about the ventilation circuit for which they are the minders. They are like sponges and at all costs should be afforded access to the information to make both the ventilation officers' job an easier one and more importantly the minesite a much safer place.

A 21ST CENTURY MINDSET VERSUS A 20TH CENTURY ONE

Assembling a ventilation report in the 21st Century should be:

1. Where the total underground flow distributions can be determined.
2. Where the efficiency of individual splits can be determined and tracked.
3. Where the condition of groups of appliances can be assessed and remedial works prioritised.
4. Where the total distribution of particular contaminants such as gas, heat or dust can be identified and managed.
5. Where the mine resistance/equivalent orifice can be determined and trended.
6. Where the operating point on the fan is recorded and used to substantiate the accuracy of existing curves or even build an operational curve if there is not one available.
7. Where the power and efficiency of fans can be validated against claimed performance.
8. Where the base data enables the maintenance of a ventilation model that can accurately predict ventilation change results.
9. Where the model accuracy is such that intricate and complex ventilation changes can be done using pressures alone with the only flow recordings being validation readings on completion.
10. Where management see the benefit of distributing the report to the underground workforce and in particular the supervisors that can learn greatly from the contents.

11. Where the report itself becomes a communication vehicle for any ventilation matters of interest such as "how to take a bag sample" or "how to build a stopping" and so on.
12. Where the report becomes entrenched as a long term training tool for front line supervisors, Deputies and Undermanagers.
13. Where people are heard to remark "How come I didn't get a copy of the vent report last month?"

Assembling a Ventilation Report in the 20th century was:

1. Carrying out the survey to only satisfy the minimum regulatory requirements.
2. Having insufficient data to maintain even a rudimentary ventilation model.
3. Carrying out ventilation changes using the "gut feel" method.
4. Having no real idea of where all the flow is distributed throughout the mine.
5. Not knowing where all the gas contributions come from throughout the circuit.
6. Not using the opportunity to slowly espouse your ventilation knowledge to the people who maintain the integrity of the mine in your absence.
7. Burying the report under a pile of books until the next survey
8. Still hearing people say "what are you doing with that wand thingy??"

It is extremely doubtful that a ventilation model could be maintained in a satisfactory state of tune by taking only the specified readings in the ventilation regulations in either state, but this paper focuses particularly on the requirements in NSW coal mines.

A "satisfactory state of tune" implies the model needs to be of a standard that will enable the ventilation officer to accurately predict resultant changes to minewide pressure and flow distributions throughout the circuit. Obviously without such knowledge there is no way of accurately predicting resultant changes in gas contaminations throughout the circuit pursuant to a major ventilation change. Ventilation changes in the absence of an accurate ventilation model are ad hoc, risky and almost always subject to some rework causing them to drag out over a substantially longer time frame, impacting negatively on both safety and production.

If a mine's routine monthly ventilation survey affords the ability to both comply with the mandatory regulations and to maintain a model which will accurately predict the results of any intended circuit changes then the work done will be well worthwhile. All the data can be easily compiled into an interesting and informative report which is then distributed as a powerful training tool to statutory supervisors or anyone else that cares to learn more about the ventilation circuit.

If only the regulatory minimum data is collected to do nothing more than to satisfy the regulations and the report is subsequently hidden under a pile of books in the report room, then it is likely that the intricacies of the ventilation circuit will not be understood by either the management or the workforce.

If persons reading this paper "resemble" the above remark they should not feel too threatened because at one stage we all used to do it that way and the regulatory changes have been altered insufficiently to ensure an entrenched cultural change has occurred at all sites over the last 30 years.

It is still quite legal and accepted in some circles and arguably sufficient in the most benign of mines to comply with only the mandatory minimum requirements on a monthly basis. Unfortunately though all mines are basically treated equally and to regard the minimum number of mandatory determinations sufficient for a gassy mine would be risky indeed.

In the 1970's, taking the ventilation readings was a job which was normally annexed to the undermanger in charge's other duties and when running a mine with 6 CM units and 500 people the process of delegation almost always ensured that the taking of the readings fell onto other shoulders. The author is well aware of this having been the subject of such delegation when employed as a colliery surveyor in a south coast mine in the 1970's when he would take the required readings basically for reasons unknown then fill in the report and file it away until the next month.

Still now though in the 21st century there are a number of mines that take little more than the readings as required under the current regulations only to file the report away without properly understanding and utilising the data.

With a simple circuit diagnosis, appropriately positioned quantity and pressure stations can be established around the mine which will not only satisfy the mandatory regulations but afford the Ventilation Officer the control that he needs over the ventilation circuit, and with only a little extra work.

INDUSTRY CHANGES AND EVENTS IN THE LAST 30 YEARS

The author is of the opinion that the ventilation regulations have not gone close to reflecting industry changes in technology over the last 30 years. There have in fact been only relatively minor changes to the mandatory quantity and gas readings that are still required to be periodically taken. The advent of computer programs including Excel and obviously modelling programs such as Ventsim have been largely ignored in many circles, and especially during the regulatory updates. It is a fact that the regulations have specified a dramatic elevation in the knowledge base of the site Ventilation Officers and also given them a mandate to focus on ventilation related issues as a priority, which is admirable and sensible. Why though with such an increased focus from one perspective has there been so little change to the specified regulations that are designed to ensure the safety and control of the mines' ventilation systems. Regulatory authorities will no doubt claim that "*we have the ventilation arrangements, what more do we need?*" The fact that there is no clear evidence of a mandatory requirement to properly and adequately delineate the circuit makes the "Ventilation Arrangements" something of a toothless tiger.

There have been numerous warnings given the disasters that have befallen the industry over the last 30 or so years. 17 men were lost in a disaster at Box Flat in Ipswich in 1972 and then a further total of 37 lives were lost in the Moura area in central Queensland between 1975 and 1994 in three separate catastrophic ventilation related disasters.

In the NSW fields there were 14 fatalities in the 1979 Appin mine disaster, which was supposedly pivotal in the upgrade of the regulations, pursuant to that event. On top of that a further 30 people were fortunate to escape when Endeavour Colliery (formerly Newvale No. 2 mine) exploded in 1995. Unfortunately, due to the lucky outcome rather than the inherent potential, this incident was largely ignored. It is no coincidence or surprise that these worst of occurrences that have been encountered in over 30 years have all been ventilation related.

CHANGE HISTORY OF THE NSW VENTILATION REGULATIONS

The NSW CMRA of 1912

The 1912 Act and its pursuant regulations were still in force in NSW up until it was replaced in 1984 by the regulations made pursuant to the NSW Coal Mines Regulation Act 1982 No. 67. Figure 1 depicts an example generic mine layout showing ventilation and gas readings that were required to be taken under the 1912 regulations up until 1984, on a monthly basis. Obviously all acts of parliament require a high degree of interpretation but the writer and a number of colleagues are of the belief that in summary the following minimum tasks were required to be carried out at that time, on a monthly basis.

1. Measurement of air quantities in the main intakes near the mine entry.
2. Measurement of air quantities in the panel intakes 100 m outbye the first working place.
3. Determinations of inflammable gas (methane) in district returns.

The NSW Coal Mines Regulation (Ventilation-Underground Mines) Regulation 1984

The 1984 upgraded regulations required quantities to be measured every 28 days at the same sites from the 1912 regulations as detailed above and also at the intake side of the face machines including miners and longwall and shortwall units. Extra methane gas determinations in excess of the 1912 requirements were required at the commencement of each hazardous zone in the mine. These extra inclusions are shown in Figure 2.

Historical regulatory requirements

In relation to air quantities and gas determinations the 1912 (pre 1984) Act and regs required the following prescribed monthly readings to comply

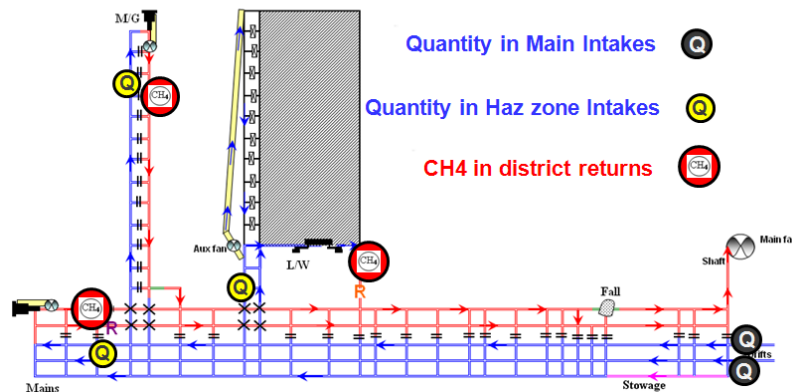


Figure1 - Historical regulatory requirements under the 1912 legislation

Historical regulatory requirements

EXTRA requirements from 1984 regs to comply

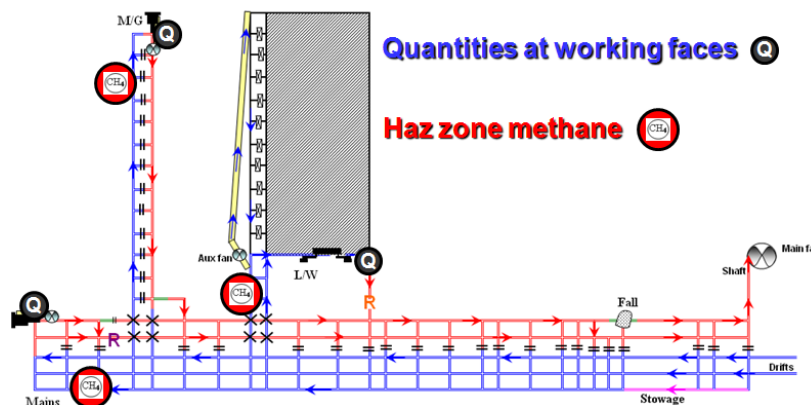


Figure 2 - Additional regulatory requirements under the 1984 legislation

The NSW Coal Mines (Underground) Regulation 1999 under the Coal Mines Regulation Act 1982

The upgraded 1999 regulations were the first pursuant to the Moura No 2 mine disaster which changed the demographic of the ventilation profession in that a far higher level of training and ventilation knowledge was required to fill the role of a VO at a NSW coal mine (and a similar result with the rewrite of the Queensland regulations). With this change in the required knowledge base of the ventilation professional, there was also the advent of a documented Ventilation Control System (VCS) in NSW. This would ensure that a well documented management plan would be assembled to manage the various system components. This was seen as a proactive move and a shift away from prescriptive regulations. Still though, and in some contrast, the regulations continued to stipulate the mandatory readings required to be taken around the circuit on a monthly basis and many sites continued to summarise their circuit performance by measuring at little more than these locations. Only minor changes were made to the stipulated determinations that were required to be measured and recorded under the new regulations.

They included a monthly determination of the carbon monoxide, carbon dioxide and oxygen content in the ventilation district returns described as parts of the mine ventilated by a separate air split. These extra requirements are detailed in Figure 3.

Historical regulatory requirements

EXTRA requirements from 1999 regs to comply

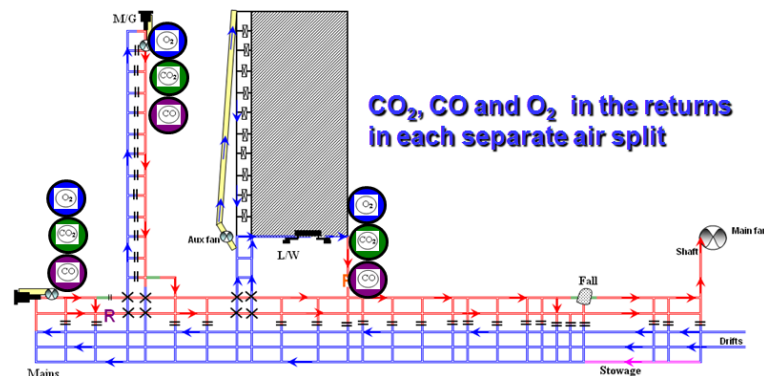


Figure 3 - Additional Regulatory requirements under the 1999 legislation

The NSW Coal Mine Health and Safety Regulation 2006 under the NSW Coal Mine Health and Safety Act 2002

This regulation was rewritten and declared in force in late 2006 and included some changes. In particular the mandatory ventilation officers (VOs) qualification from UNSW was still a requirement but an amendment was made to include the allowance of a certificate of competency to be a manager of a mine, as an appropriate qualification. This was due mainly to the industry's inability to attract people into the VO's roles which was, and still is, an industry wide problem.

Another significant change was the inclusion of a modified management plan which was to be known as the mine's "Ventilation Arrangements (VA's)". These arrangements were an upgrade of the former "VCS" from the previous regulations and focused more heavily on the control and maintenance of the mines ventilation system, amongst other issues. This was again seen as a proactive move and well complimented by a vastly improved monitoring control system.

Another change involved a formal audit of the "Ventilation Arrangements" system on an annual basis. A copy of these audits was intended to be forwarded to the department in some early draft versions but this initial requirement was surprisingly omitted in the final legal document. Despite this, these mandatory audits have become a constructive reality check for mine operators, to assist them with both the intention and compliance of their arrangements, on an annual basis.

Notwithstanding the fact that the new 2006 regulations had given the sites far more flexibility to organise and control their own ventilation systems around a practical framework, they continued to specify the mandatory locations that ventilation quantity and gas determinations would be taken. There was minor wording changes made in relation to these locations but the requirement remained fundamentally the same as the previous regulations. The minimum specified total quantity and gas determinations required under the current regulations as generally interpreted by VOs is shown in Figure 4.

MINE AIRFLOW MONITORING VERSUS MINE AIRFLOW MEASUREMENT AND CONTROL

There have been substantial technological advances in telemetric capabilities over the last 30 years and there has been a metamorphosis in the monitoring requirements around the mine as a result. In the 1912 regulations the only gas monitoring referenced dealt with the mandatory requirement to monitor the return side of a longwall face along with a directive to monitor continuous miners, if seen necessary by the chief inspector. In the current legislation there are vastly improved requirements on the locations and requirements of monitors and detectors and of course a complete section surrounding the requirements of the mines' "Monitoring Arrangements". One can only wonder then why the technological advances in ventilation pressure measurement and ventilation modelling has not brought about a similar scale of change during the same period.

The total required data is shown below.

It is a start but will not help with properly quantifying:

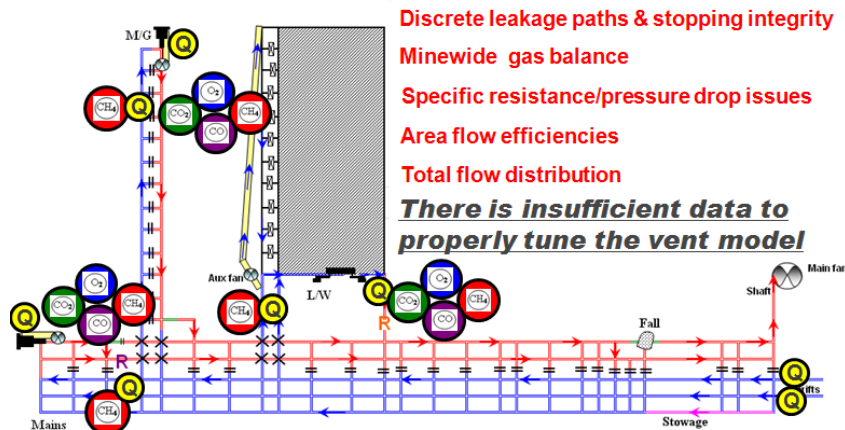


Figure 4 - Total Required data under current legislation

Monitoring regimes are somewhat reactive protecting the miner after a malfunction or event. This is not dissimilar to how personal protective equipment fits into the hierarchy of controls in safety systems at the bottom of the control process. Just as a methane alarm trip may protect a live electrical installation, a dust mask will protect a worker in a dusty uncontrolled atmosphere. Conversely it could be argued that to ensure that ventilation circuits are accurately delineated and then properly modelled to control the process and hopefully eliminate a hazard before it manifests itself is a far more proactive approach. In this manner the reliance on the more reactive and protective sentinel is lower, but still available as required. It is likely that the circuit that is measured, modelled and validated will surely have less reactive monitoring trips or events than one that is not.

A balance between accurate circuit measurement and control and the more reactive safety net of a monitoring system is a sensible approach.

REQUIREMENTS TO BUILD A USEFUL VENTILATION REPORT AND MAINTAIN A MODEL

If one examines these mandatory minimum quantity and gas requirements as summarised pictorially in Figure 4 there is no way it could be argued that this alone would be sufficient to maintain anything but the most elementary of ventilation models. The mandatory required data under the regulations will highlight both total and face ventilation flows and thus overall volumetric efficiency but will not afford the proper determination of discrete leakage paths, nor pinpoint where specific gas contamination is being contributed throughout the circuit.

Ventilation models themselves are acutely sensitive to the resistances of the roadways and it is these resistances that disburse the flow magnitudes to the particular branches. The branch resistances simply cannot be validated without taking pressure determinations. The only reference to pressure in the 2006 NSW ventilation regulations relates to a pressure gauge on the main fan. This fact suggests that the determination of not only regulator pressures but ventilating pressures in general is a requirement that is seen to be of minor importance as long as some air flows and some gas concentrations are measured. This point is well substantiated by the complete lack of the words "model", "predict" and "simulate" in the 2006 NSW ventilation regulations which demonstrates that the expected maintenance of a validated model with which to do risk based assessments on circuit changes is definitely not demanded, and possibly not even expected.

What data is actually required to properly measure the circuit?

The answer to this question is a relatively simple one.

Quantities:

- A sufficient quantity of data must be collected to delineate the total circuit flows. I.e. The ventilation surveyor needs to establish stations so that all the air entering the mine is

accounted for as either intentional flows or unintentional leakage and the magnitudes and locations of those flows is properly identified.

Contaminants:

- Gas determinations if taken at the above sites will provide a mine wide gas balance such that the gas contribution in litres/sec or litres/min from all areas can be identified and then monitored on an ongoing basis. (NB. This does not need to be done with chromatographic analysis if one (or better still two) accurate hand held instruments are utilised which makes it very quick and easy and accurate enough for routine trending)

Pressures:

- Pressure determinations at various sites are required to allow the Ventilation Officer to update the ventilation model. Without a model the pressures are of little use but definitely required if branch resistances need to be adjusted during the regular validation of an accurate ventilation model.

Where should the quantity and gas data be measured?

(Note that the suggested sites to measure flows and contaminants listed below ignore any mandatory statutory requirements. These must be done by law but are not dealt with again here. Only the most practical and suitable locations to enable the maintenance of a ventilation model are listed)

Regardless of the layout of the mine the selection of appropriate survey stations follows a similar pattern and a similar set of rules but a careful analysis of each individual circuit is always required.

Using the previous generic longwall layout the quantity and gas determinations should be kept simple and to a minimum number and taken at the following sites as summarised in Figure 5.

These locations detailed in Figure 5 include:



Figure 5 - Summary of proposed quantity and gas station locations

Pt 1 - includes a flow determination in both main returns near shaft bottom which combined with a determination of all gases provides the total mine flow and total individual gas makes reporting to the mine fan.

Pt 2 - includes a flow determination in both main returns just outbye the L/Wall corner which provides the total leakage in this outbye area of the mains and thus the average resistance of appliances in this zone as well. The gas make determination here quantifies the gas contribution in the main returns between the outbye side of the L/W and the shaft.

Pt 3 - includes a flow determination in the Longwall tailgate which also provides the flow from the mains headings inbye the Longwall by difference. The Longwall tailgate gas make determination similarly identifies the total gas make from the mains inbye the wall by difference.

Pt 4 - includes a flow determination in both main returns just outbye the development gate panel return which provides the total mains leakage between here and the Longwall and thus the average resistance of appliances in this zone of the mains can be determined also. The gas make determination here will calculate the gas emission between here and the longwall by difference.

Pt 5 - includes a flow determination in the last line of cut throughs in the mains along with a gas make determination in the intakes which highlights the total intakes gas emission from the surface to that point. A gas determination is taken in the return which will highlight face area gas emission at time of survey.

Pt 6 - includes a flow determination in the maingate panel intake which by difference will highlight the mains leakage between Pt 4 and Pt 5 and appliance resistance averages can then be determined in that area. A gas determination here in the intake and a gas determination in the return will highlight both total intake emission to the start of the maingate and total panel emission in the maingate district as well.

Pt 7 - includes a flow determination in the maingate panel intake around the last line of cut-throughs which by difference will highlight the maingate panel total leakage magnitude and again appliance resistance averages can then be determined in this area. A gas determination in both the face intake and the return behind the fan will determine both the maingate intake gas emission and the face area gas emission by difference and also the total return heading emission by difference also.

Pt 8 - includes a flow determination in the blind companion road which, with a gas determination will highlight seal leakage rates and overall gas emission if any.

Keep It Simple Stupid (KISS)

Although it is an oversimplified Longwall mine circuit it should be noted that the total quantity distribution in Figure 5 has been ascertained at eight different sites with a mere eleven velocity determinations at, with the exception of the face area readings, what should be established pre-existing stations. The total minewide gas balance has been ascertained with only fourteen individual readings per gas of interest. This is not a lot of underground work with most of the calculation work done not by the surveyor but by "Excel".

The learning here is "keep it simple stupid" as you will note in Figure 6.

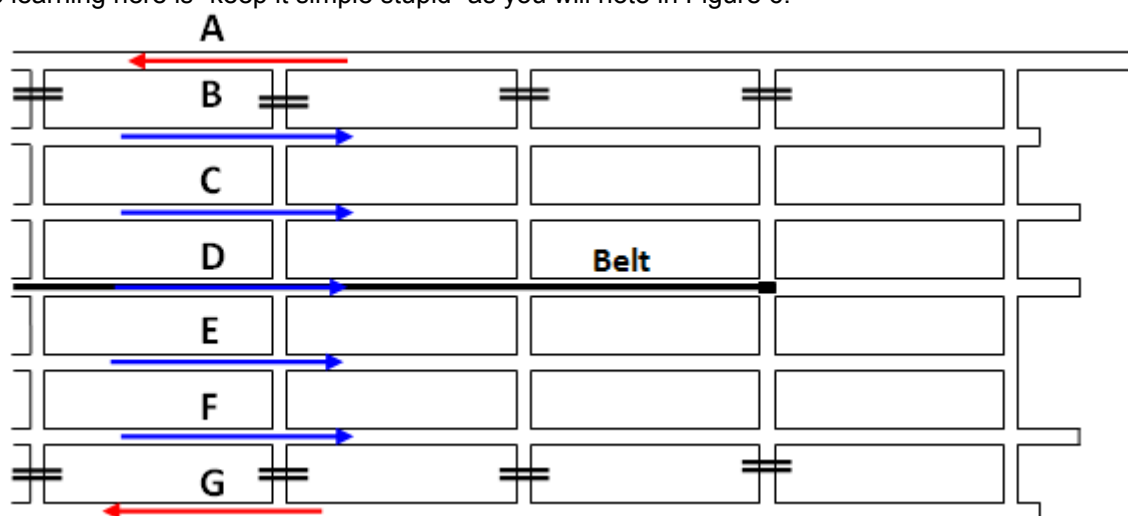


Figure 6 - Keeping data collection fast and simple

The writer has seen ventilation surveys with well over twice as many readings taken than were actually required which can make the difference between a one and a two day survey and the difference between sufficiently accurate data and possibly too much data to properly reconcile. If the total flow in Figure 6 is required, then Kirchhoff's 1st law says the total intake flow equals the total return flow with no contribution from other splits. Why measure all seven or even the five intake roadways (one with a conveyor in it!!) when two accurate return flow determinations at A Hdg and G Hdg will arguably provide the most accurate determination of the total flow.

Why and what pressure data should be measured?

1. *Pressure determinations to quantify key resistance values in the initial establishment of the ventilation model.*

Critical branch resistance values that normally require a one off determination may include:

- a) The Longwall face including the BSL and gate end shock losses.
- b) Shafts and unusually shaped mine entries.
- c) The resistance of the various styles of overcasts that are in use.
- d) Resistances of any falls, areas or stowage or other immovable hindrances to flow.

Such frictional pressure losses generally need to be measured once unless the resistance changes for any reason. They are not included in the set of pressure readings required monthly to assist with the maintenance of the ventilation model.

2. *To determine in conjunction with measured quantities the frictional resistances of the various changing or dynamic circuit features at the time of survey.*

With reference to the generic longwall circuit shown in Figure 7 locations would include:

- a) All regulators that are offering any measurable resistance to flow including supposedly "open" regulators examples of which would be at Pt6 and Pt7 in Figure 7.
- b) Any other dynamic resistance such as "weekend" style bag that may be erected across an auxiliary fan to promote duct flow during a panel flit. The writer has measured these at well over 100Pa which will substantially alter the validity of a ventilation model if missed. Another example of a critical resistance would be in a high pressure gassy mine that utilises tight tailgate corner bag to sweep air to the back of the T/G chock. The pressure on this appliance may conceivably represent 5% or greater of fan pressure on a face utilising high volumes of air. These are labelled as Pt8 and Pt9 in Figure 7.

3. *To assist with the general validation of the ventilation model.*

Strongly advised locations include;

- a) Fan pressure which should be measured as close to the blades as practicable by direct measurement or trusted monitoring. This indicates total circuit pressure at the time of survey and the mine resistance/equivalent orifice by $P = RQ^2$, as shown at Pt 1 in Figure 7.
- b) Across selected stoppings which will validate total combined exhaust and intake losses directly inbye that point and outbye it by difference. Examples of these sites are shown at Pt 2 and Pt 3 in Figure 7.
- c) Across the L/W panel entry which will determine the L/W circuit resistance including the mains returns between the M/G and T/G as shown at Pt 4 in Figure 7.
- d) Across the maingate panel entry as shown as Pt 5 in Figure 7. This is the most important pressure on the most dynamic circuit and will enable the M/G panel mesh resistance to be determined and updated. This mesh resistance can be altered on the model using a minor adjustment to the roadway dimensions or k factors to ensure the modelled panel entry differential pressure is the same as that measured across the panel entry stopping during the survey.

Routine Pressure Stations

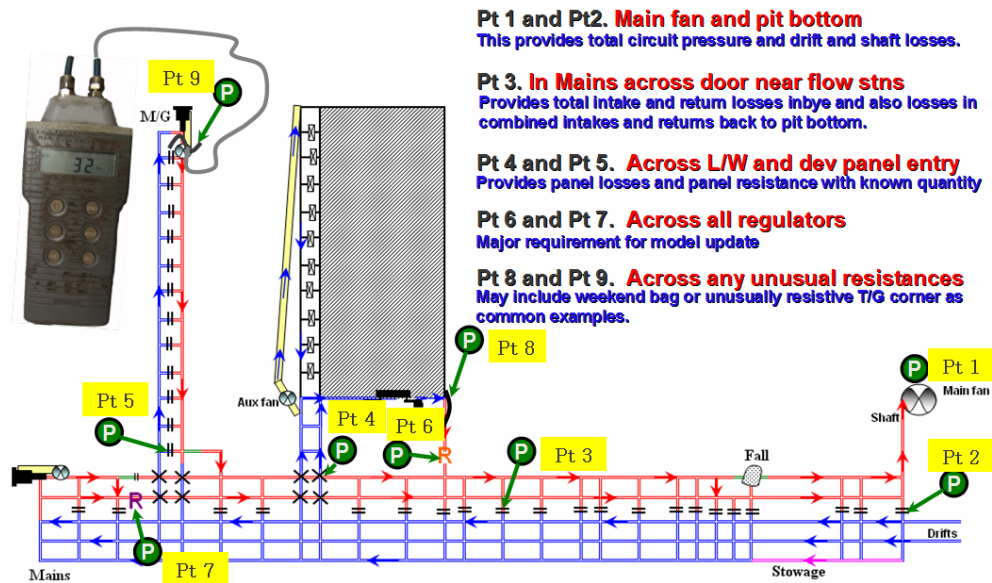


Figure 7 - Summary of proposed pressure station locations

- e) Random differential pressures should be ascertained at various sites around the circuit to assist with general model validation. As long as a good cross section of locations is chosen, their positions are relatively unimportant. When the returns are accessed it is wise to measure the differential pressure across the access door, which can be done in an instant with an electronic manometer. Further to this if pressure differentials are determined adjacent to quantity stations then these values can be used to validate and /or adjust the resistances of the total roadways (both intakes and returns) between these quantity/ pressure station locations.

How much total raw data needs to be collected?

Obviously it depends on the size and layout of the mine. An analysis of monthly routine surveys at 10 operating mines the author has surveyed (one of which would be arguably the most complex circuit in the country) showed the following results:

- Nine out of the ten pits could be surveyed in one day by one person and the other was two full days.
- The average number of quantity/gas determinations per mine was 31.
- The average number of pressure determinations per mine was 16.
- The highest number of quantity/gas and pressure readings was 63 and 31 respectively.
- The lowest number of quantity/gas and pressure readings was 21 and 10 respectively.

All surveys gathered sufficient data to accurately validate and maintain all site ventilation models on a monthly basis.

Generation of the ventilation report after the raw data is collected.

After the collection of the raw data the rest of the report can be prepared relatively easily using linked Excel cells to update most of the pages, some examples of which are detailed below.

VENTILATION REPORT INCLUSIONS TO CONSIDER

Raw Data Notes/Base Pages

These notes form the basis of the whole report and make excellent reference material if a flow needs to be checked remotely or a ventilation change is carried out under instruction. If this is available to

the deputy underground it is a simple task to direct him to any station if required. An example of an excerpt from a base page is shown in Table 1, which may include details such as:

- Station location and cross-sectional area
- Station Identification
- Last measured average velocity
- Previous gas makes at each station
- Temperatures
- Regulator details and previous pressures

Mine Schematic

This is an extremely functional plan. It serves as a useful multipurpose tool in that when included in the ventilation report it shows details such as overcasts and regulators and summarises all the flow, pressure and even gas data collected on the base pages which is a powerful training tool for front line supervisors, deputies and undermanagers.

Both Figures 8 and Figure 9 are contrasting examples of circuit schematics generated in Excel. The level of detail is a personal choice but most of the measured quantities and pressures should be depicted on the schematic.

Table 1 - Example of raw underground data

STN	DATE	LOCATION	AV VEL m/s	AREA m ²	QUANT m ³ /s	CH ₄ %	CH ₄ l/sec	CO ₂ %	CO ₂ l/sec	Temp °C		COMMENT
										W	D	
316 PANEL RETURNS												
10	1/06/2000	316 RETURN AA HDG 19-20 C/T	1.66	16.50	27.39	0.25	68	0.90	247	26.00	28.00	Stn 21m I/B 19 line
11	1/06/2000	316 RETURN A HDG 19-20 C/T	0.93	14.20	13.21	0.30	40	0.75	99	25.00	26.00	Stn 18m I/B 19 line
12	1/06/2000	316 RETURN B HDG 19-20 C/T (belt)	0.87	12.50	10.88	0.20	22	0.85	92	26.00	28.00	Stn 20m I/B 19 line
13	1/06/2000	316 RETURN C HDG 19-20 C/T	2.30	15.02	34.55	0.40	138	1.10	380	26.00	27.50	Stn 20m I/B 19 line
		TOTAL			86		268		818			
323 PANEL												
14	1/06/2000	323 B HDG RETURN	1.50	13.00	19.50	0.45	88	0.00	0	25.00	26.00	Stn 25m I/B D Hdg 317
15	1/06/2000	323 C HDG RETURN	0.05	13.00	0.65	0.00	0	0.00	0	26.00	28.00	Stn 17m I/B D Hdg 317
		TOTAL			20		88		0			Total 323 Ret to 3 Area

Regulator Details

STN	Date	Regulator	Reg Area (m ²)	Roadway Area (m ²)	Quant (m ³ /s)	Pressure Drop (Pa)	Comments
P1	1-Jun	L/W M/G	Louvered	12.5	25.30	1238	Control Readout
P2	1-Jun	519	1.45	11.9	48.8	220	measured
P3	1-Jun	323	2.42	13.2	21.0	560	measured

From a technically functional viewpoint the schematic acts as the reference tool to update the ventilation model. The model simply cannot be updated without a pictorial interface to compare the changing model to, whilst updating it.

Depending on the model validity there may be anything from fifty to a couple of thousand mouse clicks to get the model running and every single mouse click changes all the flows and pressures throughout the circuit. It is an understatement to say this would be tricky if trying to do it from a data sheet instead of a plan that is similar in layout to the model being updated. It is true that all measured pressures and flows could simply be transcribed onto a mineplan for the update process but the value of the schematic as both a communication and model update tool cannot be overstated and, as such, should be included.

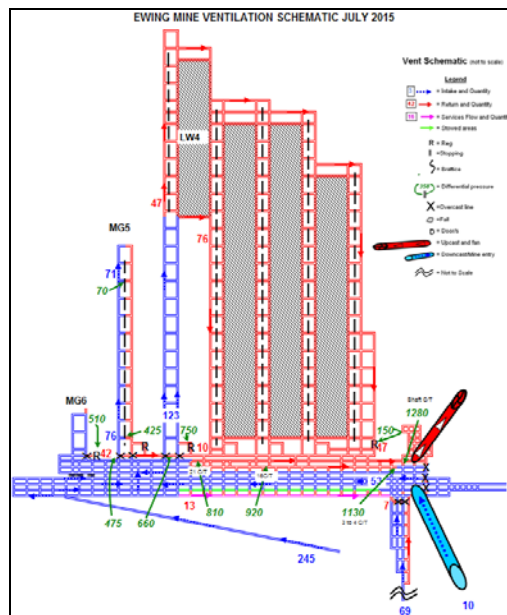


Figure 8

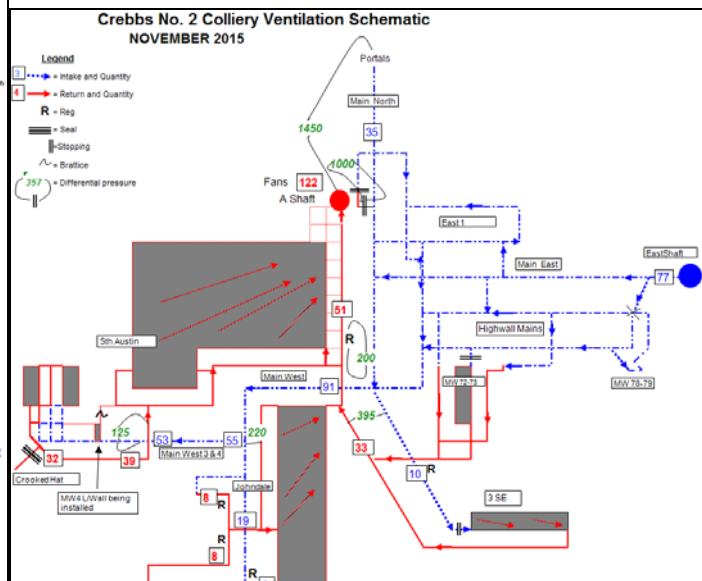


Figure 9

Figures 8 and 9 - Examples of Excel circuit schematics

Model Validation Data and Accuracy

When the ventilation model is tuned and validated, the flow and pressure displays should be inserted in the report as evidence of the tool being updated, as detailed in Figure 10. This diagram represents the modelled flows and Figure 11 represents the modelled pressures. A sensible approach is to highlight only the flows and pressures that were taken during the survey onto the model display. This allows a simple comparison of the “as read” and “as modelled” data to assess model accuracy pursuant to the update.

Persons need not panic about the accuracy to the n^{th} degree however as the model is never exactly a “clone” of the measured data. Its accuracy depends not just on the ability of the Ventilation Officer to update the software but also on the quality of the data set that was collected, in what was an operating mine at the time of survey.

The model update process should be likened to tacking into the wind in a sailing boat. Whilst the boat is never going exactly toward the desired direction it is always travelling in the correct general direction.

In a similar fashion the model will never exactly replicate the underground data. It is adjusted on a monthly basis to be as close to the measured data set as practicably possible. Obviously some months it will appear healthier than others and this point needs to be remembered.

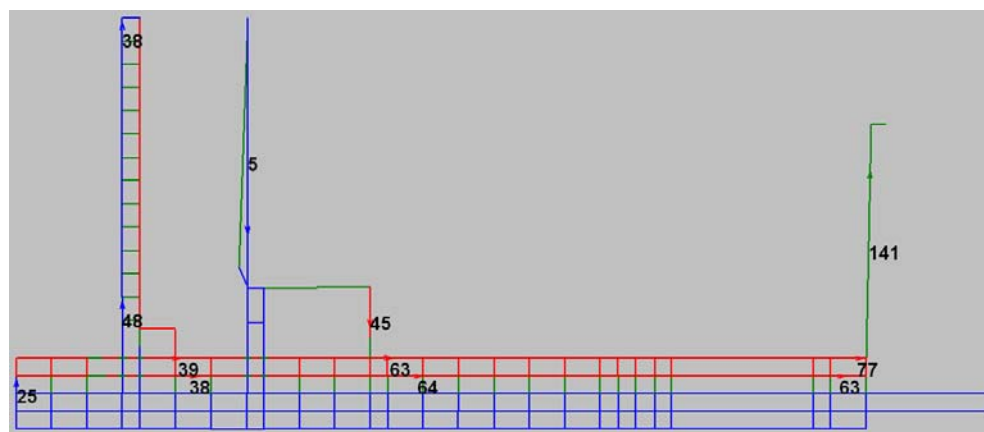


Figure 10 - Modelled flows at survey stations

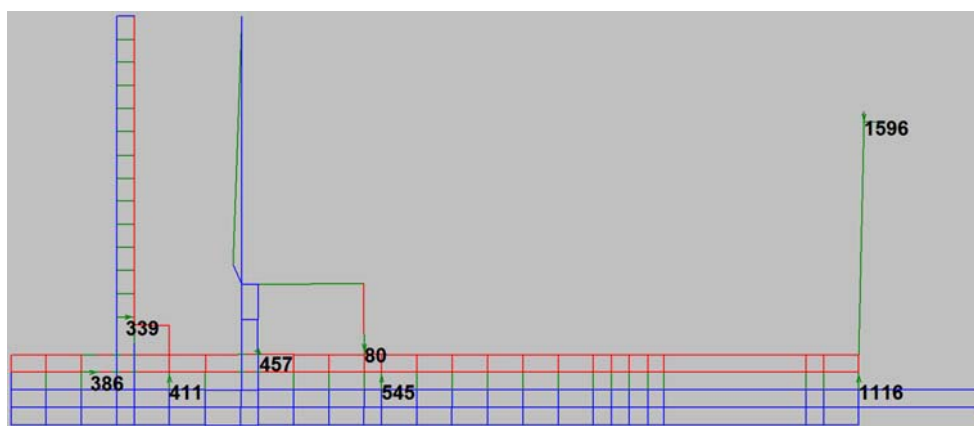


Figure 11 - Modelled pressures at survey stations

Flow Distribution and Leakage Reduction:

This is the evidence that sufficient data has been collected to properly delineate all circuit flows.

The flow distribution sector charts are the living breathing proof that the ventilation surveyor knows exactly where all the air is reporting to underground, as he should.

If the route of travel and the magnitude of all underground airflows are not known then the ventilation model cannot be properly updated nor can an accurate gas balance be executed.

Figure 12 shows the intentional flows on the left which should be monitored and occasionally adjusted as required to ensure they comply with intended requirements. The unintentional or leakage flows on the right are of much more interest. This sector chart is used to continually pareto or prioritise the areas of leakage that should be addressed next.

The primary intention of leakage identification and reduction is to elevate face flows if current magnitudes are deemed insufficient.

If face flows are satisfactory then the improved efficiency brought about by the leakage improvements may afford a reduction in the main fan operating point which will save power and money. This factor is being looked at more closely due to the world economic downturn and more importantly the onset of the Rudd governments emissions trading scheme in 2010 which will see main fan power reduction become a far higher priority. The biggest drivers affecting main fan power usage are both leakage and resistance reduction. As such the VO will play an increasingly critical role in such initiatives.

Mine Gas Balance:

The total gas contribution around the circuit is made up of the individual gas makes determined around the circuit in the mine ventilation air as depicted in the simple example chart in Figure 13. The gas concentration data is simple to collect during the survey and the individual makes are flushed straight out of the raw data and the graph updated accordingly. Further diagnosis may involve plotting the changing gas contributions against seam gas contours or production rates if and as required. The gas data is of pivotal importance when estimating resultant airflows required after a ventilation change to manage the expected gas makes in the individual splits. The data collection also acts as a cross check against the mine monitoring system data.

SOUTHFORK MINE Leakage/Flow Summary

Intentional Flows	% of total flow	Quantity (m3/s)
Mains last line	21	25
M/G panel last line	29	35
L/W face flow	37	45
Diesel bay	7	9
Charge Stn	5	6

Total Return from Survey = 193

120

Total Flow accounted for = 193

Flow/Leakage as % of Total Intake = 100

MARCH 2015

Unintentional Leakage Flows	% of total flow	Quantity (m3/s)
Pit bottom leakage flow	27	20
Mains Leakage P/B to L/W	20	15
Mains Leakage L/W to M/G	17	13
M/G panel leakage	16	12
Leakage M/G to Mains face	17	13
Leakage from M/G 3 entry to last line	1	1

73

Intentional flow (face efficiency) = 62%

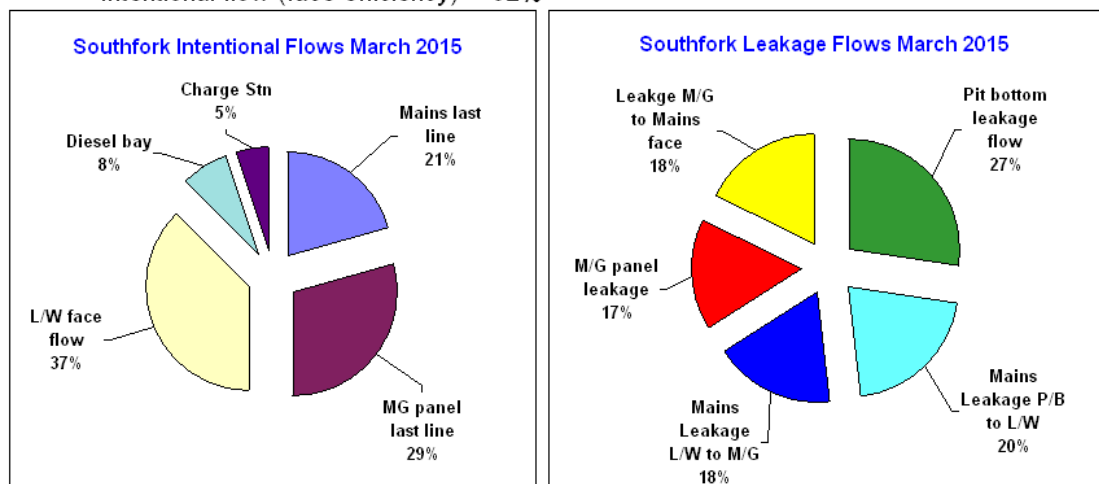


Figure 12 - Intentional and unintentional flow distribution

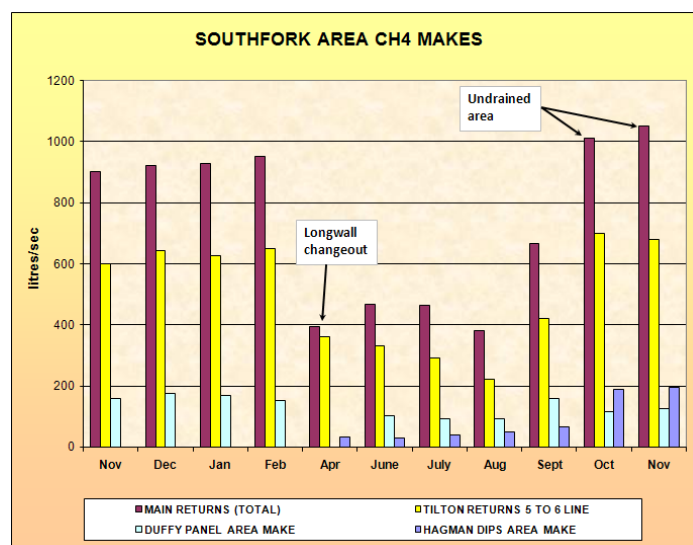


Figure 13 - Gas contributions by district

Mine Resistance Records

The resistance data can be easily graphed so that ongoing improvements and circuit changes can be properly assessed as shown in the example chart in Figure 14. The comments on this graph help with workforce awareness of the resistance reduction campaign and the effects of the removal of circuit restrictions such as the falls and stowage detailed on the chart. Also of note is the increasing mine resistance during the leakage reduction process, which at first glance looks detrimental until the improvement in face area efficiency is taken into account during the same period. Increased face area efficiencies equate to higher face flows if required or a reduction in fan duty and subsequent cost reductions, if face quantities are adequate.

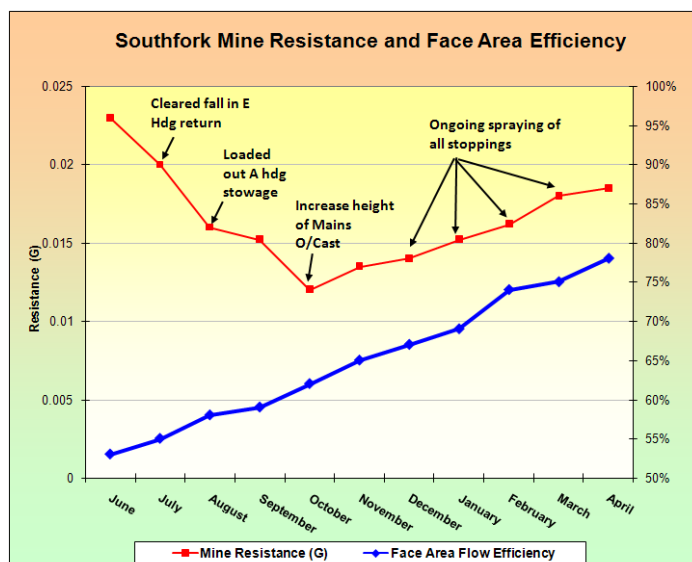


Figure 14 - Mine resistance and face area efficiency

Fan Performance Records

These details are often discounted as irrelevant but the data can be read straight from the monitoring screens at most mines and forms a detailed history of fan flow, fan pressure, input power and mine resistance at any point in time. Typical data is detailed in an example in Table 2. More importantly if there is no fan curve available the data can substantiate the operational part of the curve over time as shown in Figure 15.

Table 2 - Fans and flow summary

SOUTHFORK MINE			MARCH 2015		
MAIN FAN INFORMATION			MINE AIRFLOW SUMMARY		
No 1 Fan Running				m ³ /s	Comments
Airflow	98	m ³ /s	Total Airflow	195	From Monitoring
Fan Speed	450	rpm	Total Intakes	184	Return flow assumed equal this month
Current	341	Amps	Panel last lines	131	= Total LWV, M/G and Mains last lines
Voltage	311	Volts	and unintentional	53	= Total Intake minus total panel last line quantities
Power	149	Kw			
No 2 Fan Running			FACE AREA VENTILATION EFFICIENCY		
Airflow	97	m ³ /s			Comments
Fan Speed	455	rpm	Last line Total	71%	= (Total Panel flows)/total intakes x 100
Current	300	Amps			
Voltage	313	Volts			
Power	138	Kw			
			PANEL AIRFLOW EFFICIENCIES		
Shaft Collar Pressure	1550	Pa			Comments
Mine Resistance	0.04578	Gauls	M/G Efficiency from entry to last line	82%	As measured

Ventilation Gas and Drainage Gas Contributions

If the mine has an active gas drainage system and the drainage circuit flow data is readily procurable then the values of the gases detected in both the ventilation circuit and the drainage circuit can be easily tracked on an ongoing basis as shown in Figure 16. Further to this, the performance of the drainage circuit can be monitored in a similar manner to the mine resistance performance as shown in Figure 17 "Drainage capture by %."

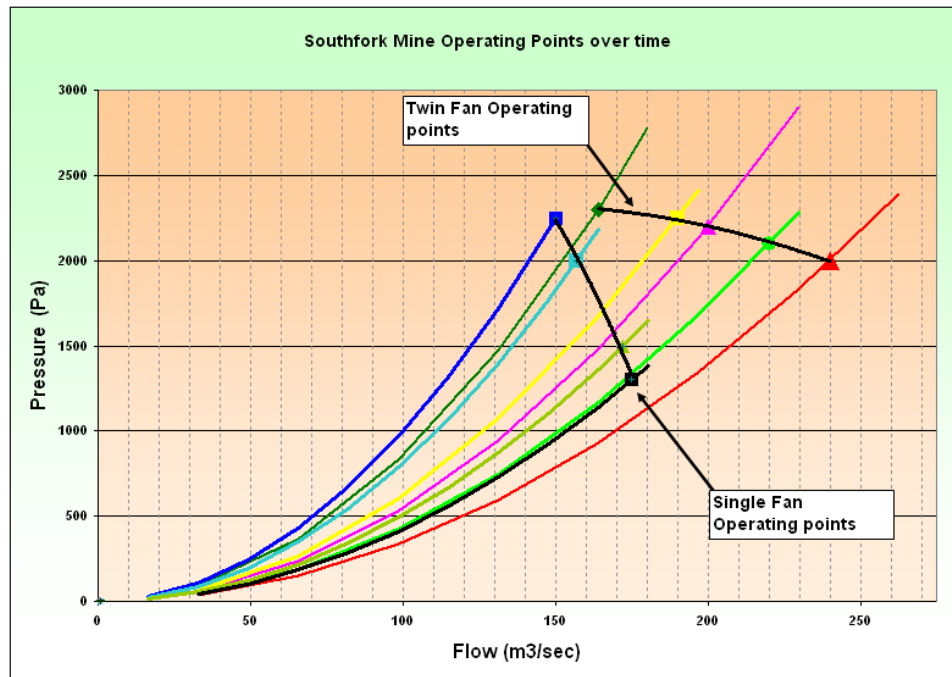


Figure 15 - Operating zone of mine fans over time

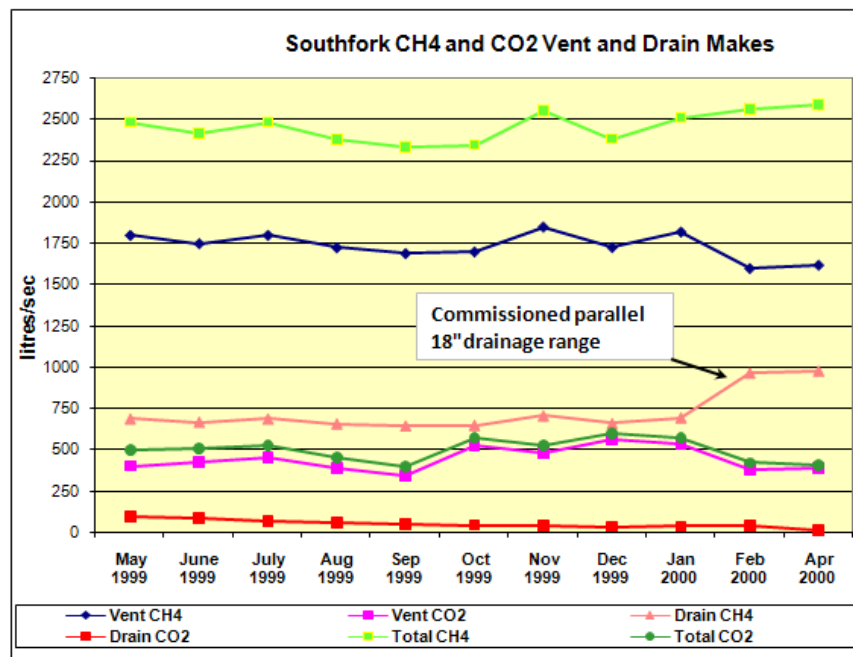


Figure 16 - Ventilation and drainage circuit contributions

Other Inclusions of Interest

Any interesting ventilation related information should be included in the report to maintain the interest of the reader and to assist with their understanding of a particular topic.

Such inclusions are only limited by the Ventilation Officer's imagination but some examples are shown grouped together in Figure 18. The value of such inclusions cannot be overstated given their influence on the overall "readability" and thus ongoing circulation of the finished document.

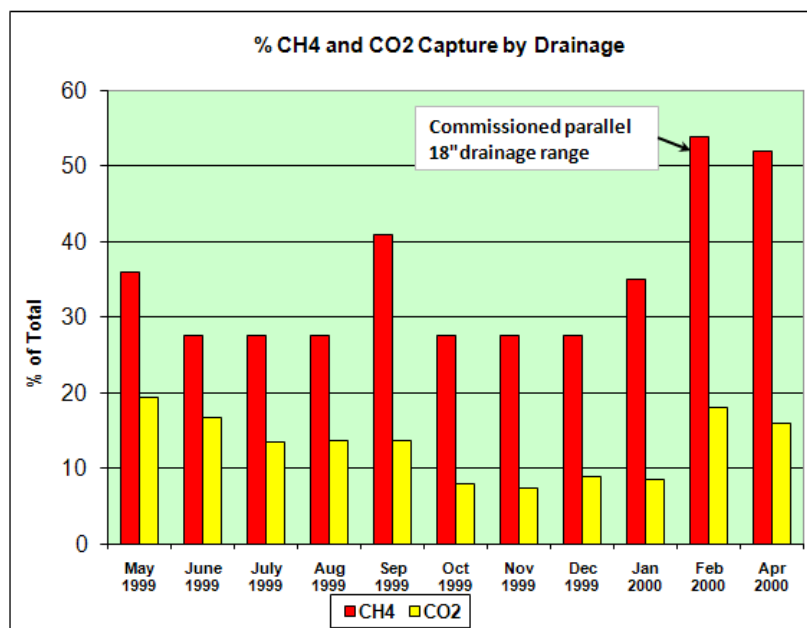


Figure 17 - Drainage capture by percentage

THE VALUE OF SHARING THE INFORMATION WITH THE WORKFORCE

The supervisors such as deputies and undermanagers are the people with the frontline responsibility of ensuring the safety of the workforce underground. Management must supply them with the appropriate resources to do this and training is a significant part of that. If management decides it is worthy to freely distribute the report, the challenge then is to make the report interesting enough for people to actually read it. It may not be totally appropriate from a protocol point of view but personalising the report to a level where individuals are sometimes mentioned along with such things as commenting on raised or lowered standards and attributing that to particular groups guarantees that the report will be very widely read. If it is widely read and contains information that teaches people particular skills and knowledge, then it becomes an ongoing free training tool that will substantially enhance the ventilation knowledge of the employees over time.

The writer is convinced of this value having seen its worth over nearly six years at a very transparent Bowen Basin mine that clearly has no secrets. Whilst difficult to objectively measure it is very apparent that the general ventilation knowledge and resultant respect for the circuit from the supervisors at that site has been well elevated during that period.

CONCLUSION

Regulatory updates have made significant inroads into elevated safety levels in our mines over the last 30 years and the resultant changes to ventilation control processes and systems have been very beneficial in that regard.

Unfortunately though, varying levels of interpretation and understanding have clouded those regulatory intentions to some degree and as such there is a myriad of different interpretations about what constitutes a reasonable compliance with adequate process control in relation to the ventilation systems in our mines. Some operations measure and delineate their circuits on a monthly basis and use an operational ventilation model as a tool with which to do risk based assessments on any intended circuit adjustments. Other more normally benign operations collect the minimum of data and utilise no ventilation modelling at all and the variability is as distinct as that.

Calculating Gas Makes

$$\text{Gas made (litres/sec)} = \text{Airflow (mL/sec)} \times \text{methane concentration (\%)} \times 10$$

Example: You measure a cross-sectional roadway area of 12m^2 and a velocity of the air in that roadway is measured at 12m/sec and there is a gas concentration in the XAM 200 of 1.0% CH₄. What is the gas ratio?

Declarations

Quantity

- + 7 x A
- + 2.2 x 12.6
- + 27.7 m³/year surface

One male = 23.7 ± 1.6 × 10³
= 463 thousands of mortars

I have installed a Metasploit Meterpreter over for the Degraded to see

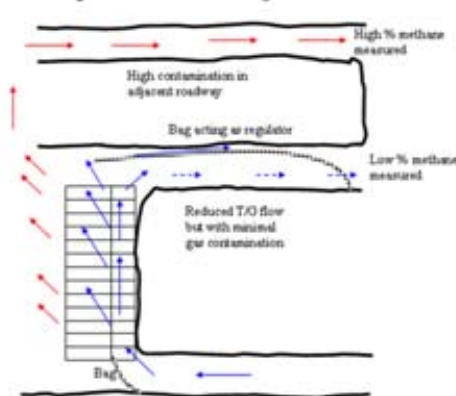
CO Make Chart (values only as shown) © Created by Julie Radford www.julieradford.com.au

[illegible]

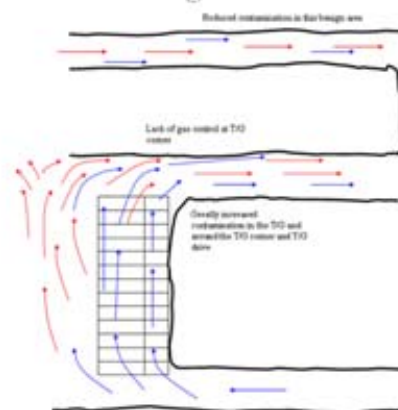
Gas make calculator (litres/second)
Methane Gas Percentage Detected (%)

	0.25	0.50	0.75	1.00	1.25	1.50	1.75	2.00	2.25	2.50	2.75	3.00	3.25	3.50	3.75	4.00	4.25	4.50	4.75	5.00	5.25	5.50	5.75	6.00	6.25	6.50	6.75	7.00	7.25	7.50	7.75	8.00	8.25	8.50	8.75	9.00	9.25	9.50	9.75	10.00	10.25	10.50	10.75	11.00	11.25	11.50	11.75	12.00	12.25	12.50	12.75	13.00	13.25	13.50	13.75	14.00	14.25	14.50	14.75	15.00	15.25	15.50	15.75	16.00	16.25	16.50	16.75	17.00	17.25	17.50	17.75	18.00	18.25	18.50	18.75	19.00	19.25	19.50	19.75	20.00	20.25	20.50	20.75	21.00	21.25	21.50	21.75	22.00	22.25	22.50	22.75	23.00	23.25	23.50	23.75	24.00	24.25	24.50	24.75	25.00	25.25	25.50	25.75	26.00	26.25	26.50	26.75	27.00	27.25	27.50	27.75	28.00	28.25	28.50	28.75	29.00	29.25	29.50	29.75	30.00	30.25	30.50	30.75	31.00	31.25	31.50	31.75	32.00	32.25	32.50	32.75	33.00	33.25	33.50	33.75	34.00	34.25	34.50	34.75	35.00	35.25	35.50	35.75	36.00	36.25	36.50	36.75	37.00	37.25	37.50	37.75	38.00	38.25	38.50	38.75	39.00	39.25	39.50	39.75	40.00	40.25	40.50	40.75	41.00	41.25	41.50	41.75	42.00	42.25	42.50	42.75	43.00	43.25	43.50	43.75	44.00	44.25	44.50	44.75	45.00	45.25	45.50	45.75	46.00	46.25	46.50	46.75	47.00	47.25	47.50	47.75	48.00	48.25	48.50	48.75	49.00	49.25	49.50	49.75	50.00	50.25	50.50	50.75	51.00	51.25	51.50	51.75	52.00	52.25	52.50	52.75	53.00	53.25	53.50	53.75	54.00	54.25	54.50	54.75	55.00	55.25	55.50	55.75	56.00	56.25	56.50	56.75	57.00	57.25	57.50	57.75	58.00	58.25	58.50	58.75	59.00	59.25	59.50	59.75	60.00	60.25	60.50	60.75	61.00	61.25	61.50	61.75	62.00	62.25	62.50	62.75	63.00	63.25	63.50	63.75	64.00	64.25	64.50	64.75	65.00	65.25	65.50	65.75	66.00	66.25	66.50	66.75	67.00	67.25	67.50	67.75	68.00	68.25	68.50	68.75	69.00	69.25	69.50	69.75	70.00	70.25	70.50	70.75	71.00	71.25	71.50	71.75	72.00	72.25	72.50	72.75	73.00	73.25	73.50	73.75	74.00	74.25	74.50	74.75	75.00	75.25	75.50	75.75	76.00	76.25	76.50	76.75	77.00	77.25	77.50	77.75	78.00	78.25	78.50	78.75	79.00	79.25	79.50	79.75	80.00	80.25	80.50	80.75	81.00	81.25	81.50	81.75	82.00	82.25	82.50	82.75	83.00	83.25	83.50	83.75	84.00	84.25	84.50	84.75	85.00	85.25	85.50	85.75	86.00	86.25	86.50	86.75	87.00	87.25	87.50	87.75	88.00	88.25	88.50	88.75	89.00	89.25	89.50	89.75	90.00	90.25	90.50	90.75	91.00	91.25	91.50	91.75	92.00	92.25	92.50	92.75	93.00	93.25	93.50	93.75	94.00	94.25	94.50	94.75	95.00	95.25	95.50	95.75	96.00	96.25	96.50	96.75	97.00	97.25	97.50	97.75	98.00	98.25	98.50	98.75	99.00	99.25	99.50	99.75	100.00																																																																																																																						
Relative Humidity (at 145)	1.00	0.99	0.98	0.97	0.96	0.95	0.94	0.93	0.92	0.91	0.90	0.89	0.88	0.87	0.86	0.85	0.84	0.83	0.82	0.81	0.80	0.79	0.78	0.77	0.76	0.75	0.74	0.73	0.72	0.71	0.70	0.69	0.68	0.67	0.66	0.65	0.64	0.63	0.62	0.61	0.60	0.59	0.58	0.57	0.56	0.55	0.54	0.53	0.52	0.51	0.50	0.49	0.48	0.47	0.46	0.45	0.44	0.43	0.42	0.41	0.40	0.39	0.38	0.37	0.36	0.35	0.34	0.33	0.32	0.31	0.30	0.29	0.28	0.27	0.26	0.25	0.24	0.23	0.22	0.21	0.20	0.19	0.18	0.17	0.16	0.15	0.14	0.13	0.12	0.11	0.10	0.09	0.08	0.07	0.06	0.05	0.04	0.03	0.02	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.0

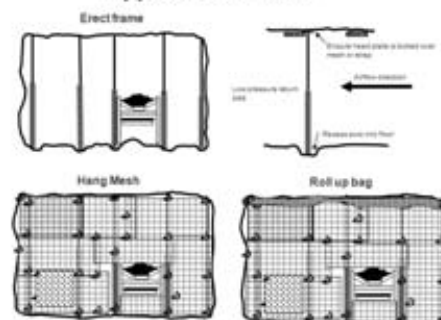
Importance of Bag Standards



and the danger without them



Appliance Erection



Methane Decay Curves

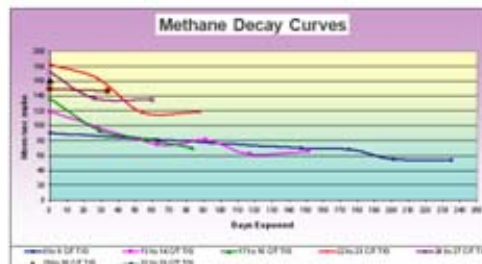


Figure 18 - Various inclusions of interest

One intention of this paper was to demonstrate that the data collection can be a lot simpler than some may think and whilst the specifics of model updates were barely touched on, most ventilation officers will have more than enough ability to maintain their own models if given the time and resources to do so.

It should be remembered also that if the ventilation report is properly configured a considerable amount of useful output will be created for what is not a begrudgingly large amount of input. In a short period of time the mine will be controlling the ventilation circuit rather than the ventilation circuit controlling the mine.

Finally one of the key learning's in thirty years of waving the wand is that site people really are interested in how it all works. It is not a black art and nor should be treated as such. Take the opportunity at all cost to utilise the ventilation report as a training tool and spread the word. Sure they find mistakes in it!! The author has been notified about those on many occasions but they have been the most satisfying conversations ever because it is concrete proof that it is actually being read, by the people that really matter.

INNOVATIVE APPROACH TO MAINTAINING MINE VENTILATION DURING FAN UPGRADE AT CARBOROUGH DOWNS MINE, JUNE 2008

Owen Morgan¹ and Martin Watkinson²

ABSTRACT: Carborough Downs Coal Mine was developed from a box cut using three entries, one for men and materials, one for the conveyor and one return portal. The initial drivage was conducted by drill and blast and each access was ventilated by a Flaktwoods 27 m³/s 1058 50 kW fan. The box cut fan installation was transferred from the sister mine in NSW and installed on the portal. After 18 months the shock losses in the fan to adit adaptor were severely affecting the ability to increase the number of development units operating in the mine. It was decided to perform a fan upgrade prior to the installation of the main fans and fan shaft in March 2009. The fan housing adaptor was modified to reduce the shock losses and the motors and gearboxes were upgraded at the same time. The planning processes, minor ventilation changes and supervision which were conducted to maintain mine ventilation during the overhaul period are discussed. The three Flaktwoods 1058 fans were bolted into a shipping container and monitored for flow, pressure and vibration to suit the requirements of the Queensland legislation and the ventilation maintained within the mine at 75 m³/s down from 145 m³/s with the original fans. The upgraded ventilation capacity achieved was 245 m³/s. Two of these fans are currently being used in series to ventilate the new conveyor drift with a design capacity of 28 m³/s at 1200 m through 1218 mm ducting.

INTRODUCTION

Carborough Downs coal mine is situated approximately 140 km south west of Mackay and 35 km east of the township of Moranbah in central Queensland.

Development of the mine commenced in 2006 by the excavation of a box cut to simplify the seam access. Seam access was by three stone drifts: conveyor access, men and materials and return. Each of these roadways was driven by drill and blast to access the seam and ventilated by a Flaktwoods 27 m³/s 1058 50 kW fan through 1218 mm steel ducting exhausting to atmosphere. Once the coal seam was accessed and ventilation connections made the surface fan arrangement comprising two Richardson 1965 CY and a plenum chamber attached the fans to the portal in parallel. These fans and portal adaptor had been lying idle at the sister mine of the Integra Coal Mine (Glennies Creek) in New South Wales (NSW) and had been fully overhauled with new Toshiba 250kW motors and Hansen 3.11:1 gearboxes rotating the impellers at 473 rpm. Once installed they supplied a total of 145 m³/s to the pit, 40 m³/s below the original modelling predictions that had been carried out during the feasibility study stages prior to the development of Carborough Downs Mine. The original planning was to have only two bord and pillar sections with a nominal total ventilation requirement at around 120 m³/s, so the initial modelling showed ample quantities to be able to cope with the expected conditions. Approval of the expansion plan to include longwall mining required the working of four development panels to achieve the desired schedule. This requirement necessitated the upgrade of the mine fans in the box cut until the main fans could be installed in the planned return shaft. The approach used to maintain mine ventilation during the change out of the fan adaptor and fan upgrade is the subject of discussion in this paper.

DEFINITION OF THE PROBLEM

In 2007 the mine expansion plan was approved which required the operation of up to four working panels to achieve longwall start-up in June 2009. Ventilation investigations carried out by John Rowland of Dallas Mining Services, Wollongong, revealed that the existing fan installation had unacceptable shock losses, around 500 Pa in the adaptor, and ventilation modelling also revealed that a fan upgrade would also be required to increase available ventilation quantities to achieve

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longwall start-up of June 2009. The upgrade required an increase in the size of the motors from 250-280 kW and the installation of new gearboxes with a ratio of 2.5:1 giving a projected impeller speed of 595 rpm and capacity of 245 m³/s. The long-term plan was to install a return shaft and fan installation to provide life of mine ventilation. The location of the fan shaft is adjacent to the first longwall block shown in Figure 1.

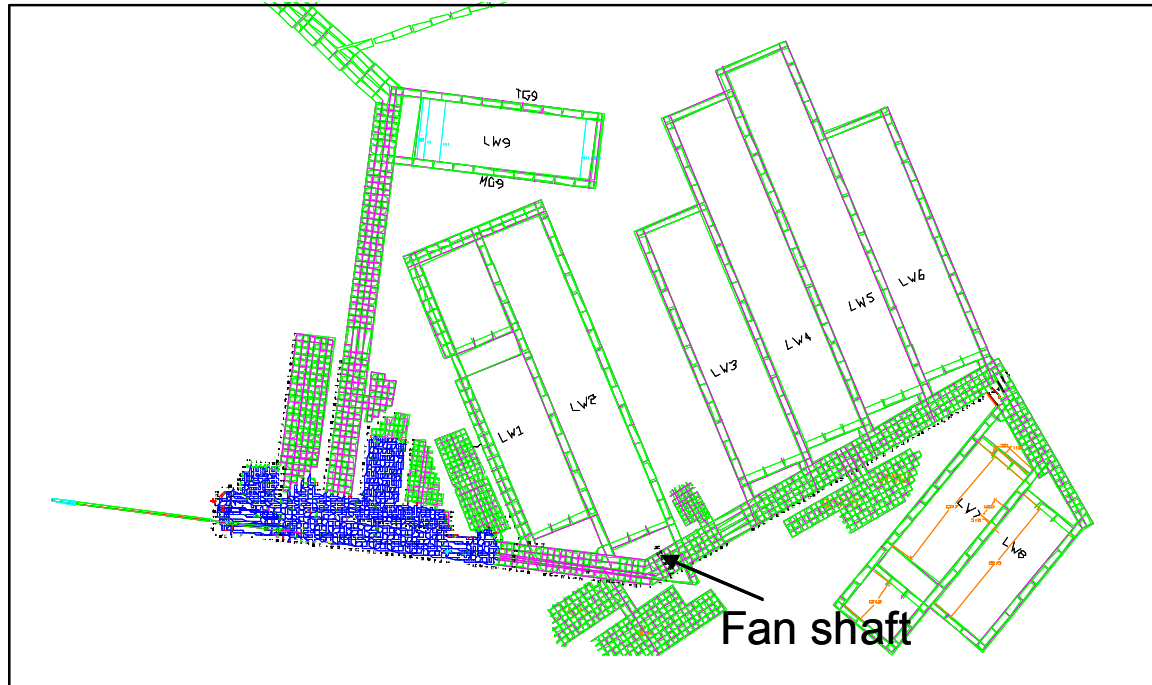


Figure 1 - Longwall layout plan for the first 9 Panels

Once it was identified that the adaptor change out and fan upgrade was required the search commenced for suitable temporary fan to maintain the mine ventilation. The mine was driving longwall access roads with typical insitu gas content over 5 m³/t and up to 6.2 m³/t post drainage, therefore maintaining mine ventilation was critical.

The work on the fan upgrade and adaptor modification was planned to coincide with a required statutory annual high voltage switch testing and the opportunity was also going to be taken to install the conveyor belts for maingate 1 development.

PLANNING FOR MINE FAN UPGRADE

It was not possible to obtain a suitable fan for maintaining the temporary ventilation of the mine for the three day period required (12th to 14th June 2008). At this time attention was paid to the three Flaktwoods fans that had been "mothballed". Whilst on their own they could not produce sufficient ventilation, but modelling analysis indicated that in parallel they could produce up to 90 m³/s.

A quick and effective solution was now sourced to fabricate suitable temporary fan housing. It was decided to simply bolt the three Flaktwoods fans in parallel in a steel shipping container. The shipping container was then adapted to fit the mine using mesh and brattice at the conveyor portal. Each of the fans was monitored for vibration and pressure in the mines Citect monitoring system to comply with the Queensland coal mining act and regulations. Full ventilation modelling was undertaken using Ventsim to predict the anticipated mine ventilation during the change over process.

Detailed work lists were prepared to site temporary ventilation structures which could be quickly commissioned on 12th June 2008. Work included the preparation of temporary stoppings and the installation of brattice rolls which could be quickly deployed and removed during the ventilation changeovers. Another working procedure was prepared for the sequence of events required to

establish the temporary ventilation circuit on ceasing the operation of the main fans. Maintaining the mine ventilation enabled the tailgate to complete its panel extension, the installation of the maingate conveyor belt in preparation for panel production and the continuation of in-seam methane drainage operations.

ESTABLISHING TEMPORARY VENTILATION

The stages of the ventilation change over were:

1. Ensure all appliances are pre-erected as required before commencing change and ensure any air leaking through or past the tubes at the road-header fan site has been sealed.
2. No road the E heading road header district.
3. Shut down main fans and ensure underground power is tripped.
4. Enter the mine via fan drift and close stopping appliance at bottom of fan drift. (See Figure 2 for appliance locations and steps)
5. Move the four gas tree at 3 line D heading to ensure the it is in the ventilation stream from the northern return.
6. Open D heading 3 to 4 cut-through double doors and chain back. Information tag must be attached with reason for doors being open recorded.
7. Close both brattice stoppings in B to C heading 4 and 5 cut-through.
8. Drop bag around temporary coffin seal D 5 to 6 line.
9. Start the three temporary portal fans (Flactwoods 50 kW fans) and check airflow is established.
10. Close the regulator in 6 line E-F heading to allow 25 m³/s to return along F heading 6-12 cut-through. (The target regulator pressure drop is around 60 Pa)
11. Proceed to 11 cut-through and open the doors at 11 C to D to get air flowing inbye in the belt road.
12. Check ventilation status on travelling inbye into the mine.
13. Adjust the brattice stopping outbye of the overcast in 53 cut-through B heading to allow 10 m³/s to ventilate the drill stub at 54 cut-through
14. Measure the return quantity at tailgate 01 regulator and the last open line (6A) and note general body gas levels.
15. Measure return quantity for the East Mains panel at the last line and note general body gas level.
16. Notify the control room officer that the pit is open and designated work areas are safe.
17. Conduct full ventilation survey sufficient to update the Ventsim model.

Conducting these ventilation changes meant the mine return roadways connected to the conveyor drift which then became the temporary return drift. All other ventilation circuitry remained the same in the mine and therefore only minor changes were required to maintain the gas monitoring system, consisting of a 4 gas tree (CO, CO₂, O₂ and CH₄) at the shipping container and the minor relocation of the 3 cut-through gas tree (around 5 m).

A sensor change from Explosion Risk Zone/Non-Explosion Risk Zone (ERZ/NERZ) status to environmental status was carried out at 16 cut-through as the return for the outbye drift drivage was closed off during the fan shut down and the face area no-rooded. This negated the requirement to have the ERZ/NERZ boundary in place and re-directed the ventilation that was normally sent to this return further into the mine to the critical face areas. As the drift area was being driven in stone and the face area was only 20 m in, the no-roading of the area was predicted not to cause any concerns with gas accumulations

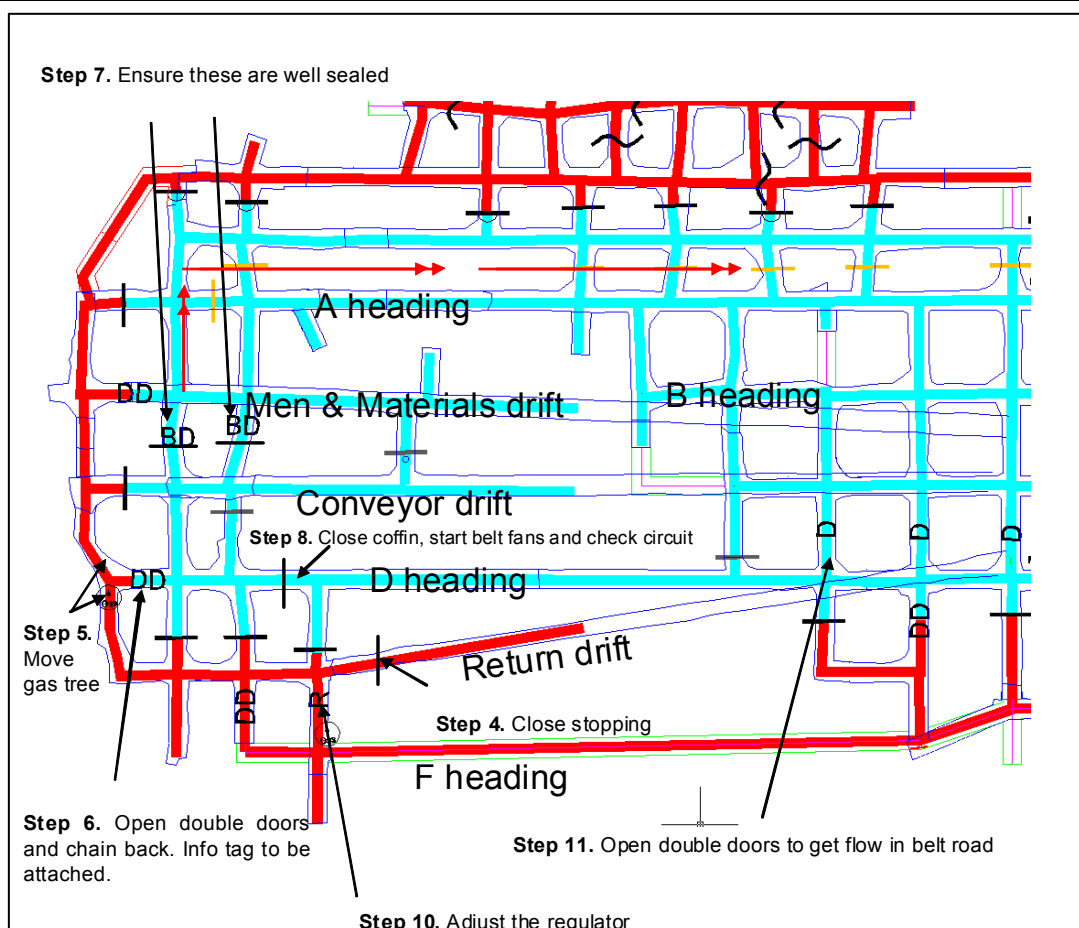


Figure 2 - Plan detailing ventilation changes on stopping of the main fans

The mine ventilation officer carried out the ventilation changes, conducted ventilation surveys and inspected the underground districts to monitor methane levels and make minor adjustments to the district ventilation quantities. The maximum methane percentage found during the temporary ventilation arrangements was 1.2 % in the tailgate return and was as per the predictive modelling had shown. Full details of the methane percentages and ventilation quantities pre-restarting the upgraded fans are provided in Table 1.

Table 1 - Ventilation Quantities and methane percentages before main fan start-up after re-commissioning

Panel	Heading or C/T	Intake / Return	Quantity m ³ /s	CH ₄ %
Belt Drift	Vent station	Return	93	0.9
East Mains Return	F 57 to 58	Return	21	1.1
TG return	A 0 to 1 line	Return	20	1.2
Drill rig site	B52 to 53 line	Return	10	1.0
Return	F Hdg 6-12 line	Return	5	1.0

One hiccup in the system was realised when a loader that was being used to install a quickseal at 60 c/t shut down in the return with a methane trip. The problem arose when the crew installing the quickseal dropped the sheet to the ground and the ventilation was travelling from further inbye carrying general body concentrations of 1.2 % CH₄. The ventilation officer was present in the east

mains panel at the time and steps were taken to recover the loader using a temporary ventilation change. In order to recover the loader the tailgate panel was closed off to increase the quantities to the east mains return and therefore reduce the general body concentrations in the return where the loader was located to below 1%. This change worked well although it created another problem as by the time the loader had been removed from the return and the tailgate ventilation was being restored it had given the methane (0.25%) in a stub located 5 m inbye of the ERZ/NERZ boundary sensor in the tailgate time to reach the sensor location with the lack of ventilation in the area. This tripped underground power. The fans that were being used to ventilate the mine during this shutdown were being powered from the 1050 volt tramming outlet on the side of the main fan switch room. This was connected to underground power and therefore tripped off the fans. The ventilation in the mine was only lost for twenty minutes but the delay was then in re-powering the mine. The whole loader recovery, that had only interrupted the tailgate ventilation for a period of 15 minutes, caused nearly two hours delay to the power in the panels. The upgraded mine fans were re-started as detailed in the procedure below and identified in Figure 3.

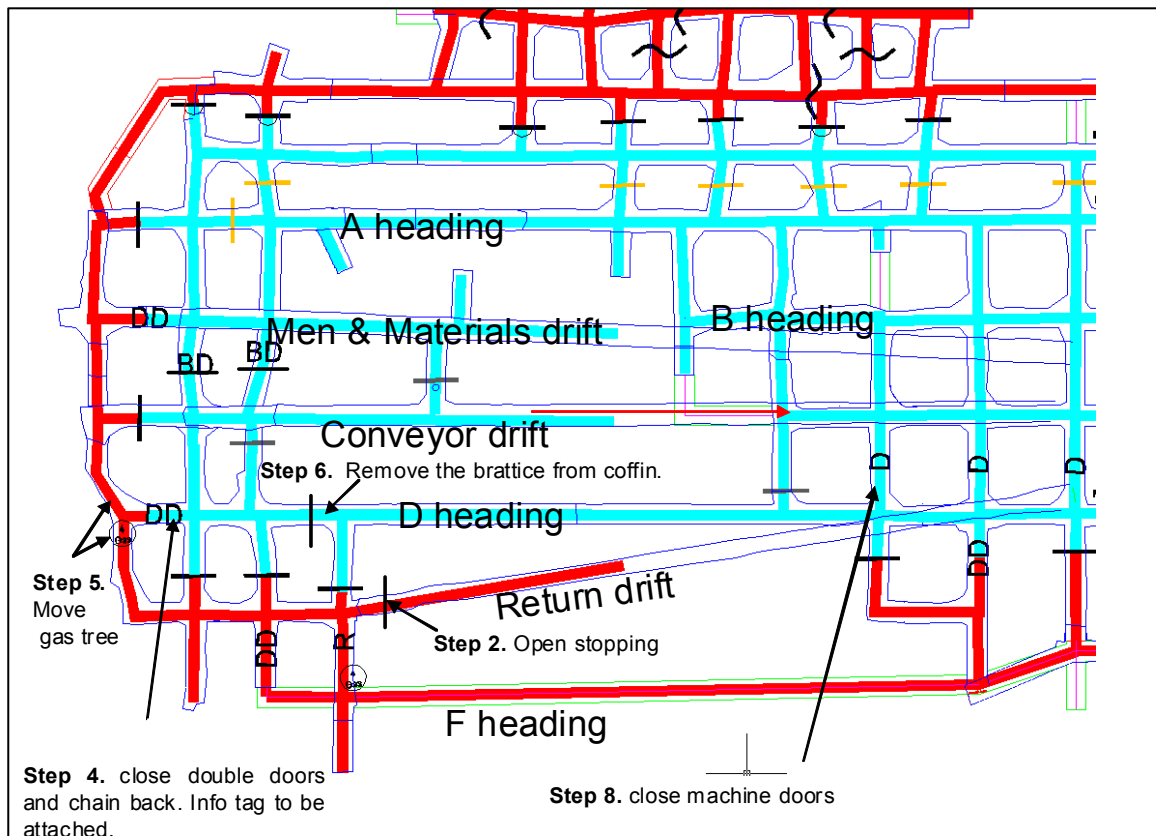


Figure 3 - Plan showing changes for recommissioning of the upgraded fans

RE-ESTABLISHING MINE VENTILATION

The main fan upgrade and ducting change out was actually completed four hours before the schedule and the ventilation circuit was then returned to its original state. Steps in Ventilation change to re-establish the normal mine ventilation were:

1. Ensure that the air lock doors, explosion doors and the damper doors on the main fan installation are closed
2. Enter the mine via the ventilation drift, dismantle the substantial brattice stopping at the bottom of the drift and store the material in 6 cut-through D-E heading to recover later.
3. Shut down the three temporary fans, open all doors and the flap on the Belt Portal and remove brattice
4. Close the machine doors at 3-4 line D heading and remove the information tags
5. Move the 4 gas tree in 3 line D heading back into the original position in the ventilation stream
6. remove the brattice from the temporary stopping in D heading 5-6 cut-through
7. Start the main fans
8. Close the machine doors at 11 cut-through C-D heading

9. Reopen the man door at 14 cut-through E-F to establish ventilation back in the E heading drift district
10. Reset the ERZ/NERZ sensor at 16 cut-through E heading
11. Change the ERZ/NERZ sensor at 16 cut-through E heading back over from environmental to ERZ/NERZ
12. Adjust the brattice stopping out-bye of the overcast in 53 cut-through B heading to allow 10-15 m³/s to ventilate the drill stub 54 cut-through B heading.
13. Measure quantities and pressures sufficient to update the mine model and record the panel quantities for the ventilation change report.
14. Update the alarm set points on Citect to reflect the relative quantities.

Subsequent to the upgrade a ventilation capacity of 245 m³/s was achieved and the mine operated with 5 working panels during October 2008:

- maingate 01
- tailgate 01
- one bord and pillar section
- mains drivage
- back drivage of the new conveyor drift.

Two of the Flaktwoods fans were used in series to ventilate the new conveyor drift from the surface with a design capacity of 28 m³/s at 1200 m through 1218 mm ducting. After the upgrade the mine collar pressure increased to 1055 Pa which increased localised leakage at the 3 and 4 cut-through location where there were a number of machine doors. Any ventilation structure which had a pressure differential of more than 500 Pa was inspected and an inspection regime implemented in line with the mines spontaneous combustion hazard management plan

CONCLUSIONS

- Mine ventilation planning needs to cater for possible increases in ventilation requirements during the life of the mine. That is catering for possible additional requirements by having contingency in the available quantities/pressure available. ie the initial modelling for Moranbah North indicated that there was a requirement for two fans with a mine requirement of 250 m³/s; three fans were installed at the fan shaft to cater for modifications to the mining plan.
- Fan in-situ tests to BS848 give a true fan curve for the fan(s) as connected to the mine and enable reliable ventilation network analysis to be conducted.
- A practical approach utilising on site equipment enabled the fan upgrade to be completed whilst maintaining mine ventilation.
- Detailed planning and dedicated supervision enabled the upgrade to be completed with minimum interference to mine operations and no hazardous accumulations of methane to occur.

REFERENCES

Queensland Coal Mining Safety and Health Regulation 2001
Queensland Coal Mining Safety and Health Act 1999

VENTILATION AND GAS EXTRACTION TECHNIQUES OF HIGH GAS SEAMS IN KAZAKHSTAN

Dzhakan Mukhamedzhanov¹, Serghzy Baimukhametov² and Aleksander Polchin³

ABSTRACT: The execution of gas drainage techniques to degas the high methane content seams, and to allow their safe working, is influenced by the low permeability of the coal seams. The extraction techniques of coal degasification employed are; mining the underlying thin seam before extracting the main thicker target seam; pre-drainage of the target seam, from surface in conjunction with in-seam drainage, both using vacuum extraction and; taking a top lift of the very thick seams. A significant proportion of the liberated gas is also extracted from the goaf area. Several special techniques are employed in managing the goaf gas, including gas extraction from surface boreholes, bleeder drivages and gas drainage from roadways constructed above the extraction seam. Methane removal levels of 60% are achieved through effective methane gas drainage management and control in coal production. These practical measures contributed to increase in methane gas volume available for utilisation to 120 m³/min.

INTRODUCTION

The Coal Division of ArcelorMittal operates eight underground mines in the Karaganda coalfield in Kazakhstan. The current production of coal is 12 Mt per annum. The working of the coal is constrained by the so called "methane barrier", which manifests itself by a high coal seam methane content at the depths more than 500 m. Most of the mines operate at depths greater than 500 m and working conditions are therefore characterised by high methane release levels at the development and production units of up to 100-150 m³/min. of pure methane equivalent.

EXTRACTION SEQUENCE

To allow safe extraction of the thick but very gassy K₁₂ coal seam, the underlying K₁₀ coal seam, which is over 4 m thick, is extracted first. The 7 m thick, K₁₂ seam can then safely be mined in two lifts, with a capacity of 10 Kt per day.

The description of this technique by undermining of the K₁₂ seam by the K₁₀ seam at Abayskaya mine is outlined in detail as follows.

Seam setting

The K₁₂ seam is some 440 m below surface. The distance between K₁₀ and K₁₂ is 55-60 m. At the distance of 30-35 m. from the K₁₀ there is a thin seam K₁₁ which is 1.0-1.5 m thick. The K₁₁ horizon is used as the drainage drive level.

The immediate roof of the seam is medium-hard mudstone, with a thickness of 5.0-8.8 m, which readily caves. Dip angles range between 10 and 26 degrees. Operating seam height of the K₁₀ is 4.45 m, with an extracting height is 3.9 m.

Natural gas content of the K₁₀ seam, as determined during the original exploration, is 25.7 m³/ton. There is methane ingress into the goaf of the K₁₀ from the overlying K₁₂ seam, giving a relative gas emission of 44.70 m³/t. 80-90% of the total K₁₀, K₁₁ and K₁₂ gas is extracted during and from the mining of the underlying K₁₀ seam.

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Gas Drainage methods

Different gas extraction methods and drivages are shown in the following figure: These consist of a combination of in-seam extraction drill holes, goaf holes from surface and an overlying sewer drainage drive.

The trials for goaf drainage, through cross measure drilling, are currently being tried out at the neighbouring mine. The longwall face ventilation is by conventional U-type system, with an intake and return gate roads. The ventilation quantity is 1,500 m³/min at the face. The methane capture by means of ventilation and degasification is shown in Table 1.

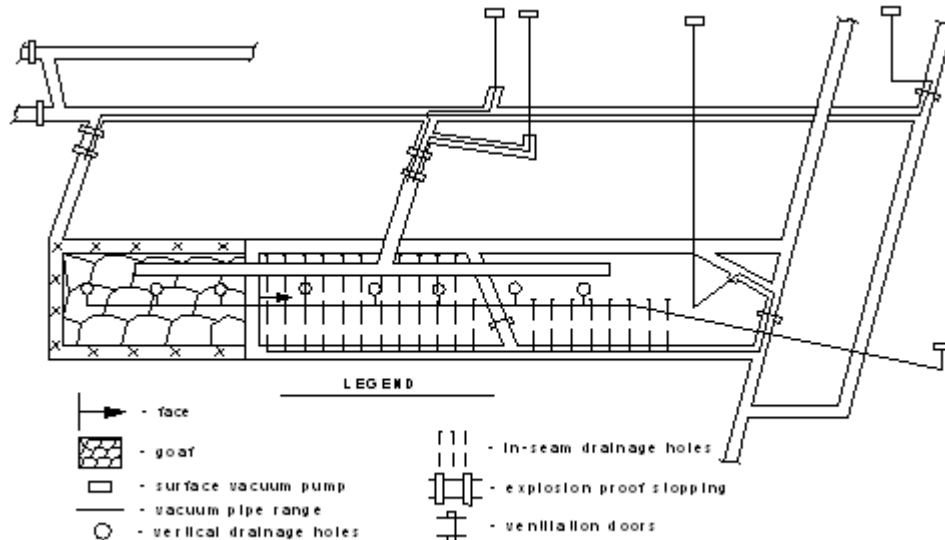


Figure 1 - The draft of methane emission control at Abayskaya mine

Longwall block pre drainage

Preliminary degassing was carried out within the working seam and included:

- preliminary seam pre drainage holes, which are drilled from the intake gate; the distance between holes – 8 m and the length – 120-140 m.; the term of degassing - 268 days;
- progressive advanced seam degassing holes, during or immediately prior to extraction, which are drilled from return gate; the distance between holes – 2 m and the length – 100-120 m.

The practice showed that the most efficient method was advanced degassing holes in the distance of 10 m from face line (the abutment zone in front of production face line); they removed 2.0-2.5 m³/min.

Goaf drainage

The main part of methane from the longwall goaf was captured by using isolating seal at the return incline K₁₀ (73.6 m³/min pure methane), i.e. more than 50% of total methane content of area. 15 vacuum pumps (HB-50) were in operation for this method of degassing to provide safe work in the production unit. Two gas pipelines (402 mm diameter) were connected from the seal to three main wells (two 325 mm diameter well and one 500 mm diameter).

The methane content in the face and production unit was constantly in the range of 0.9-1.1% under such methane management method. The gas content exceeded the allowable standard in the goaf at the distance of 5 m from face end. The efficiency of this method began to decrease after 200 m of face advance and methane content increased up to 1.5% at the face end. The situation at the production face has changed when the faceline reached the gas drainage gate. The gas drainage gate is driven in the overlying thin seam K₁₁ at a distance of 30-35 m from working seam K₁₀. The development rate was not limited by gas factor.

Table 1 - methane capture by means of ventilation and degasification

Methane recovery methods	Methane capture, m ³ /min			Effectiveness coefficient of degasification
	Methane-air mixture Q_{mix} , m ³ /min	Methane purity K, %	Methane capture in terms of 100% Q_{CH_4} , m ³ /min	
Per face output	4500 tonnes			
Ventilation			39.0	
Return	2100.0	0.9/1.1	21.0	
Isolated methane withdrawal by means of mine depression	100.0	18.0	18.0	
Methane content in upper face end		0.9/1.5		
Degassing			98.0	0.78
1. Vertical degassing wells	40.6	60.0	24.4	0.17
2. From seals of mine roadways	408.9	18.0	73.6	0.52
Total methane content of the section			137.0	

The efficiency of the degassing method of gas emission source, during coal seam mining, is dependent on accurate determination of geomechanical processes in the mining area. The efficiency of the gas drainage gate technique was dependent on its right location, taking into consideration all rock movements and faults. The major point during this degassing method designing is the determination of expected volumes of methane emission from the undermining seam (K_{12}). Methane flow rate from undermining seam is dependent on the active gas emission area of undermining seam, its natural and residual gas content.

In accordance with calculations, the actual methane release rates from undermining seam K_{12} showed that it is required to increase air-methane mixture in the gas drainage gate for isolated methane withdrawal. Therefore two gas pipelines (402 mm diameter) were applied to isolating seal. The air-methane mixture was pumped out through two main degassing wells (325 m diameter) by means of six vacuum pumps. However, the increased gas emission from the undermining seam occurred mainly in the active movement area, which was limited by the critical angles of movement and rock relief. Thus, the optimal location of gas drainage gate, providing maximum efficiency, was at the distance of 1.5-1.6 length of face from the return gate. By using this degassing method for production section 32 K_{10} -N at Abayskaya mine the calculations defined the gas drainage gate location at 30 m from the return gate and 30-35 m from the mining seam.

Gas balance

The methane capture by ventilation and degassing is shown in Table 2. Methane removal levels of 60% were achieved by using the methane management and control at the production unit through the gas drainage gate K_{11} . This allowed the increase in methane volume available for utilization (concentration 25%) to 120.5 m³/min (or 75% of total methane saturation of unit). The maximum methane concentration in the return was no more than 0.9% (per face per day output of 2000 t); the concentration of methane in the upper face end was lower 1.0%.

Table 2 - methane capture by ventilation and degassing

Methane capture methods	Methane capture, m ³ /min			Effectiveness coefficient of degasification
	methane-air mixture Q_{mix} , m ³ /min	methane purity K, %	methane capture in terms of 100% Q_{CH_4} , m ³ /min	
Per face output	5500 tons			
Ventilation			16.8	
Return	2100.0	0.8/0.9	16.8	
Methane concentration in lower face end		0.7/0.8		
Degassing			142.1	0.89
1. Vertical degassing wells	36.5	55.0	20.1	0.12
2. From seal of return incline K_{10}	225.2	8.0	18.0	0.10
3. Isolating seal of gas drainage gate K_{11}	179,3	56.0	100.4	0.63
Total methane content of the section			158.9	

OUTBURST THRESHOLD LIMITS – ARE THEY APPROPRIATE?

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ABSTRACT: The 1994 outburst threshold limits imposed on coal mines operating in the Bulli seam were lower than the conservative value proposed by Lama in 1991. Equally conservative is the DRI900 method for outburst threshold limit determination. A number of mines have encountered areas where it has been difficult, if not impossible, to reduce the seam gas to below the prescribed threshold limit prior to the arrival of roadway development machinery, despite extensive in-seam gas drainage. In such situations these mines can experience lengthy production delays or even loss of reserves. Several Bulli seam mines have completed reviews of their outburst risk management which led to increasing their threshold limits. These mines have been operating safely, without outburst, for some four years. The method of determining the outburst threshold limits applicable to non-Bulli seam coal mines also hold a high degree of conservatism which is discussed. The need for re-appraisal of the threshold limits undertaken is reported, based on the further data analysis. The process of gas desorption methodology and the optimum gas content is one particular aspect of this study as it has a clear influence on the established values of the recognised threshold limits. The study has demonstrated that there is justification to increase the operating threshold limits to values of 12 m³/t for 100% CH₄ and 8 m³/t for 100% CO₂. Research is continuing to include sample analysis from other Australian mines.

INTRODUCTION

The first recorded outburst of coal and gas occurred in the Bulli seam at the Metropolitan Colliery was on 30th September 1895. Since then there has been some 669 outburst events recorded in Australian underground coal mines, 449 in the Bulli seam of the Illawarra coal measures and more than 220 in the Bowen Basin (Lama and Bodziony, 1998). Various theories have been presented regarding the factors that contribute to the occurrence of coal and gas outbursts. A summary list of factors that have generally been accepted as having the potential to contribute to an outburst is given by Lama (1995):

1. Tensile strength of coal
2. Gas emission rate
3. Gas pressure gradient
4. Moisture level
5. Depth or stress level

Previous studies have concluded that in the Bulli seam stress does not play a significant role and it is gas which is the major contributing factor to outburst occurrence. The use of gas drainage to reduce seam gas content levels to a value considered safe for mining has been uncritically accepted by the mining industry. The factors that are considered to impact upon outburst propensity have been incorporated to provide an assessment of outburst risk condition, shown in the outburst risk matrix in Figure 1.

Virtually all of the outbursts that have occurred in the Bulli seam have been associated with geological structures and been located in areas where no substantial gas drainage has been undertaken. There have been 12 outburst related fatalities in Bulli seam mines (Harvey, 1995) all of which occurred in areas without any gas drainage and where carbon dioxide was the primary seam gas component (Lama, 1995).

Following the last outburst related fatality, which occurred at West Cliff Colliery on 25th January 1994, the NSW Department of Mineral Resources (DMR) issued a directive to all Bulli seam mine managers detailing actions to be implemented at their mines. Arguably the most significant of these actions was the stipulation of limits on seam gas content prior to mining, known as threshold limits and shown

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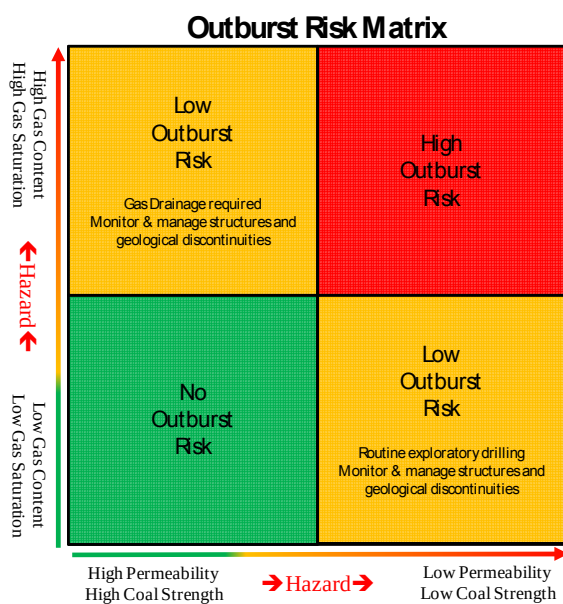


Figure 1 - Outburst risk matrix

graphically in Figure 2A. The introduction of the threshold limits resulted in a significant increase in the intensity of drilling and gas drainage in these mines for the purpose of structure identification and gas content reduction. Operators developed comprehensive outburst management plans which included standard drilling patterns and routine management controls to deal with the issue of gas content reduction. Subsequently outburst threshold limits have been adopted by all Australian mines and in the majority of cases these controls have proven successful, with the exception of two small outbursts. These two outbursts occurred in Bowen Basin mines, at Central Colliery on 20th July 2001 and at North Goonyella Mine on 22nd October 2001.

There has been a sharp decline in the research effort directed toward improved understanding of the outburst phenomenon, since the introduction of the outburst threshold limits and the virtual elimination of outburst occurrence from the Australian coal industry

With the ever increasing production capacity of mining equipment, mine operators are endeavouring to produce at much faster rates and in many cases the conventional gas drainage management techniques are struggling to achieve sufficient gas content reduction ahead of the advancing mine development. In such situations the typical response of operators has been to increase the density of boreholes through infill drilling to reduce the spacing between boreholes, however this may not be sufficient and production delays may still result. If not effectively managed it is possible that gas drainage may provide very little benefit and it is therefore important to monitor and understand the behaviour and performance of the gas drainage system to enable problems to be identified and appropriate corrective action taken where necessary (Black and Aziz, 2008). In extreme cases operators have chosen to sacrifice coal reserves in favour of redirecting mining effort to areas with more favourable drainage response. Recently both Tahmoor and West Cliff Collieries have completed formal reviews of their respective outburst threshold limits which supported increasing the threshold limits. The revised threshold limits for these two collieries are shown graphically in Figure 2B. Several other Bulli seam mines are now considering, or in the process of, reviewing threshold limits.

It is important to note that the lack of outburst incidents, although positive for the industry, has prevented the collection of outburst related experience and data necessary to improve the technical understanding of the outburst phenomenon. Therefore to a large degree the threshold limit reviews are underpinned by qualitative risk assessment and lack detailed technical assessment. The Gas Research Group (GRG) at the University of Wollongong is presently conducting a number of projects to improve the industry's understanding of gas storage, transport and drainage characteristics the results of which will support quantitative assessment of outburst risk.

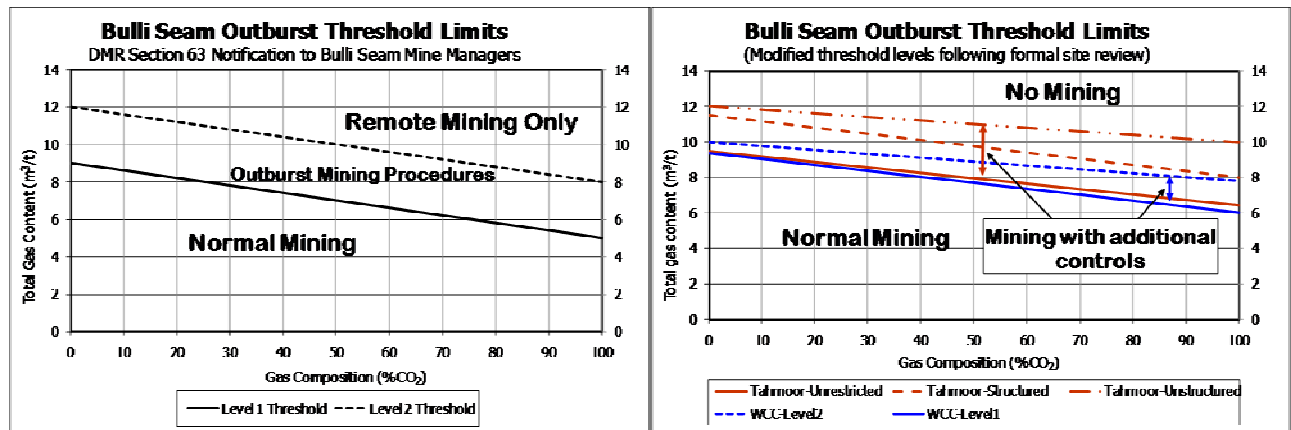


Figure 2A - DMR specified outburst threshold levels

Figure 2B - Revised thresholds – West Cliff and Tahmoor

BULLI SEAM OUTBURST THRESHOLD LEVELS

In 1995 Lama provided a description of the process that led to his 1991 recommendation of threshold levels applicable to Bulli seam mines. Lama suggested that where structures exist, within a zone of 2.5 metres from the mine workings, the 'desorbable' gas content should be less than a threshold limit of 8.0 m³/t (100% CH₄) to 4.0 m³/t (100% CO₂) and in all other areas, free of structures, the 'desorbable' gas content should be less than a threshold limit of 10.0 m³/t (100% CH₄) to 7.0 m³/t (100% CO₂). Lama acknowledged that these limit values were somewhat conservative to account for what was considered to be a high rate of development advance, up to 75 m/day.

In reviewing the methodology used by Lama it is apparent that the proposed outburst threshold limits were essentially based on previous operating experience in the Bulli seam, and the inferred gas content and composition of the seam gas present in areas where outbursts had occurred. The fact that there had been no recorded outbursts where the gas content was known to be less than the proposed threshold limits supported the proposal.

Recent slow desorption testing conducted by the GRG has demonstrated that gas will continue to desorb from coal samples in slow desorption testing for a period well beyond 12 months. Should the testing undertaken by Lama have not been afforded sufficient time to completely liberate the 'desorbable' gas content, then the gas content levels measured would be understated by several cubic metres per tonne and the actual values should be greater than those presented.

In order to determine whether the gas content within the coal seam in a particular area is below the prescribed threshold limit, coal samples, typically core samples, are collected for analysis. There is a need for mine operators to obtain gas content and composition data from coal samples as quickly as possible, to determine if an area is 'below threshold' and therefore considered safe to allow mining to continue or otherwise 'above threshold' and therefore requiring additional action to further reduce gas content.

The fast desorption method of gas content measurement, as described in AS3980, is the method accepted and employed by the Australian mining industry. The fast desorption method of gas content measurement does however determine the 'total' gas content of a coal sample, which is greater than the desorbable gas content. Lama (1995) acknowledges the need for the 1991 proposed threshold limits to be changed to reflect outburst threshold limits based on 'total' gas content. The process used by Lama to determine 'total' gas content outburst threshold limits was to determine the 'residual' gas content for both high CO₂ and high CH₄ coal seam gas conditions and simply add these measured values to the previously stated 'desorbable' gas content threshold values as (Equation 1).

$$\text{Total Gas Content } \left(\frac{\text{m}^3}{\text{t}} \right) = \text{Desorbable Gas Content } \left(\frac{\text{m}^3}{\text{t}} \right) + \text{Residual Gas Content } \left(\frac{\text{m}^3}{\text{t}} \right) \quad (1)$$

$$\text{Total Gas Content } \left(\frac{\text{m}^3}{\text{t}} \right) = \text{Desorbable Gas Content } \left(\frac{\text{m}^3}{\text{t}} \right) + \text{Residual Gas Content } \left(\frac{\text{m}^3}{\text{t}} \right)$$

In determining the value of residual gas content for both high CH₄ and high CO₂, to be added to the desorbable gas content threshold limits, Lama simply averages the mean residual gas content values determined from four separate tests. The test results reported by Lama have been reproduced and presented in Table 1. Lama acknowledges that in the case of the residual gas content determined for CO₂ in laboratory testing of dry coal samples the measured value is unacceptably high and the reported result was halved to achieve a more appropriate value for inclusion in the averaging exercise. It should also be noted that the reported residual gas content for CH₄ is greater than CO₂ for both the underground and surface borehole sampling, which is contrary to accepted gas sorption theory.

Table 1 - Residual gas content in Bulli coal samples (after Lama, 1995)

METHOD	MEAN RESIDUAL GAS CONTENT (cc/g)		
	CH ₄	CO ₂	CO ₂ *
Laboratory sorption (DRY)	2.21	6.76	3.38#
Laboratory sorption (MOIST)	1.67	3.72	3.72
U.G. Sampling	2.01	1.96	1.96
Surface borehole sampling	2.13	1.09	1.09
Other independent labs. (UG sampling from KCC operations)	2.00	2.90	2.90
COMBINED MEAN VALUE	2.00	3.29	2.61

* Test results have been modified

Value half of the result obtained from testing

From this method Lama reports the residual gas content values to be added to the previously proposed 'desorbable' gas content limits of 2.01 m³/t (100% CH₄) and 2.4 m³/t (100% CO₂).

Therefore the outburst threshold values, representing 'total' gas content are as follows:

1. Within a zone 2.5 m either side of a structure the 'total' gas content should be less than a threshold limit of 10.0 m³/t (100% CH₄) to 6.4 m³/t (100% CO₂); and
2. In all other areas, absent of structures, the 'total' gas content should be less than a threshold limit of 12.0 m³/t (100% CH₄) to 9.4 m³/t (100% CO₂).

Recent testing of Bulli seam coal samples by the GRG has determined that, in the case of sorption testing at normal temperature and pressure (NTP) conditions, the residual gas content was in the order of 0.87 m³/t for 100% CH₄ and 1.98 m³/t for 100% CO₂, and in the case of slow desorption testing, the residual gas content ranged between 0.63 m³/t and 1.8 m³/t. These results support Lama's acknowledgement that insufficient desorption time was allowed prior to residual gas content testing and the values presented in Table are likely to be somewhat overstated.

Based on the Section 63 directive from the DMR it appears that an additional 'factor of safety' was applied to the gas content threshold values as the limits imposed on Bulli seam mines was less than the limit values proposed by Lama. It also appears that allowance was not made for the introduction of intensive in-seam gas drainage drilling and the impact on structure and therefore outburst risk identification.

As shown in Figure 2B both West Cliff and Tahmoor Collieries have completed formal reviews of their respective outburst management process which resulted in their receiving approval to increase outburst threshold limits. Both mines have been operating with the increased threshold limits in place for some four years whilst remaining free of outburst.

NON-BULLI SEAM OUTBURST THRESHOLD LEVELS

In 1995 Williams and Weissman presented the concept of using gas desorption rate as a means to determine applicable outburst threshold limit values for coal mines operating in coal seams other than the Bulli seam. Underpinning this desorption rate proposal was an apparent relationship with the Bulli seam threshold limit values previously proposed by Lama, shown in Figure 3. The test involves measuring the volume of gas emitted from a 200 gram sub-sample of coal core sample after crushing for 30 seconds and relating the result to the total gas content of the full core sample. As shown, the data presented, which represents samples with gas composition >90% CH₄ and >90% CO₂, indicates

that at the proposed threshold values of 9 m³/t (100% CH₄) and 6 m³/t (100% CO₂) a common desorbed gas volume of 900 ml is liberated. It was therefore concluded that the total gas content which corresponds to a gas desorption of 900 ml represents the outburst threshold limit applicable to that coal mine. This method, known as DRI900, has been uncritically accepted by the mining industry for determining outburst threshold limit values applicable to non-Bulli seam mines.

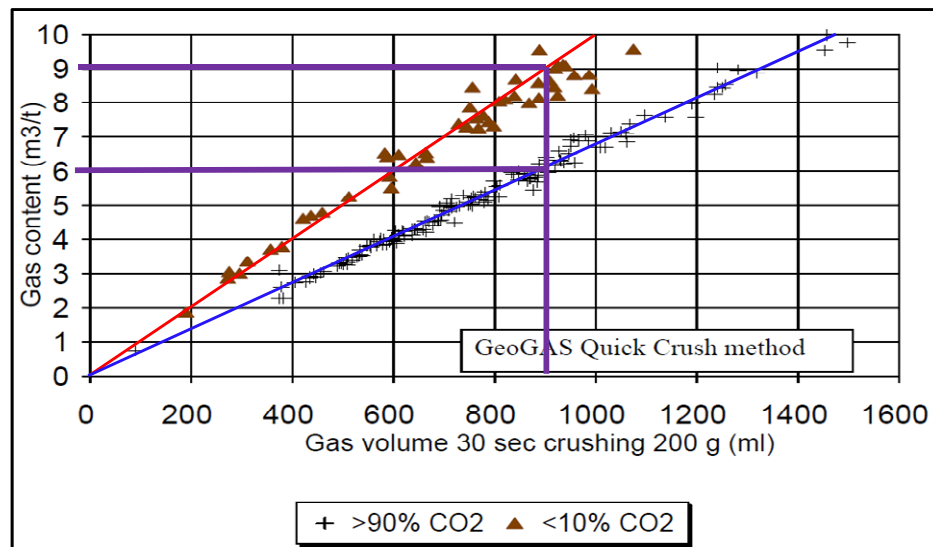


Figure 3 - GeoGAS desorption rate (DRI900) relative to Lama's outburst threshold limit values

Given the potential for Lama's proposed threshold levels to be somewhat conservative it is possible that the DRI900 value may be somewhat conservative and therefore understate the appropriate outburst threshold limit in non-Bulli seam mines. This is further supported by the fact that two Bulli seam mines have been successfully operating at threshold limits greater than those upon which the concept was originally based.

Consider a situation where state of the art drilling and data collection technology is employed at a Colliery as part of routine in-seam gas drainage drilling and that this technology is capable of identifying geological structures and other anomalies as well as draining seam gas. In such a Colliery, operating in the Bulli seam, it is considered reasonable, given the previous work of Lama and the recent experience at Tahmoor and West Cliff, that a threshold limit of 12 m³/t (100% CH₄) and 8 m³/t (100% CO₂) is not unreasonable in areas free of structures. Applying this threshold limit to the gas desorption dataset presented by Williams and Weissman, a DRI of 1200 is indicated (Figure 4).

Given the potential for the DRI900 concept to be understating outburst threshold limits in non-Bulli seam coal mines further investigation was undertaken to validate the Gas Desorption / Gas Content relationship used by Williams and Weissman (1995). Core sample gas content and composition data was obtained from two Bulli seam Collieries and analysed to enable direct comparison to the GeoGAS data. The results from this analysis show that in the case of Mine A the average gas desorption / gas content relationship is independent of gas composition and both the >90% CH₄ and >90% CO₂ trend lines have a similar gradient, which are also similar to the GeoGAS >90% CO₂ trend line. The data from both Mine A and B, within the gas content and gas desorption ranges presented by Williams and Weissman, is shown in Figure 5. The gas data from Mine B shows the trend line for >90% CH₄ is also similar to the CH₄ and CO₂ results from Mine A and the CO₂ results from GeoGAS. The Mine B >90% CO₂ trend line however has a higher gradient, which is the result of increased early stage desorption from samples with higher total gas content. The complete data set from both Mine A and Mine B, incorporating the GeoGAS datasets is shown in Figure 6. The Mine B data indicates that for >90% CO₂, below approximately 7.5 m³/t (total gas content), the average gas desorption / gas content trend line is approximately equal to the >90% CH₄ trend line.

It can be concluded from the analysis of 930 core samples representing a broad range of gas content and composition conditions within two Bulli seam mines, that the gas desorption / gas content is, to a large degree, independent of gas composition.

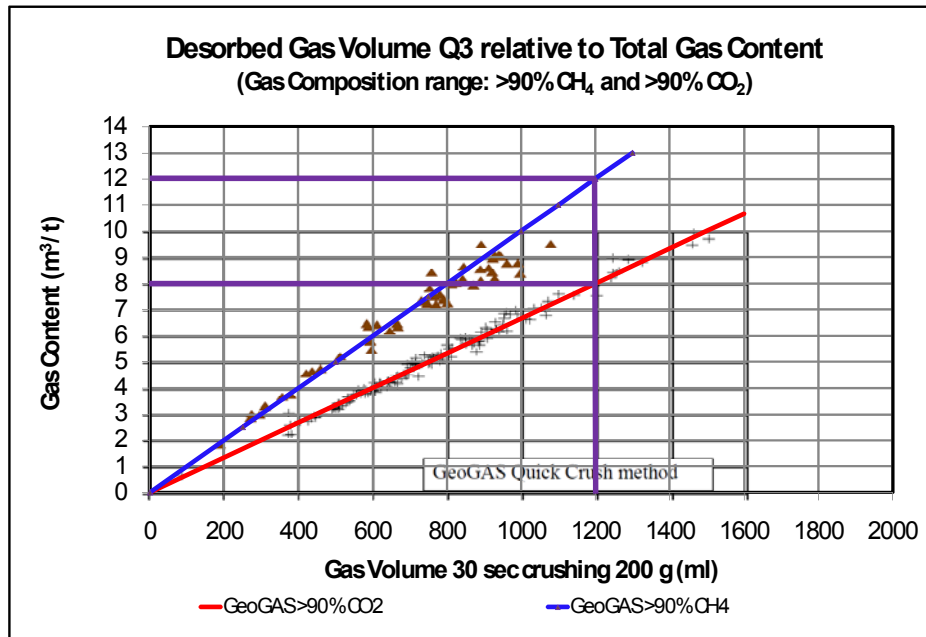


Figure 4 - DRI1200 indicated for potential Bulli seam outburst threshold limits in non-structured areas

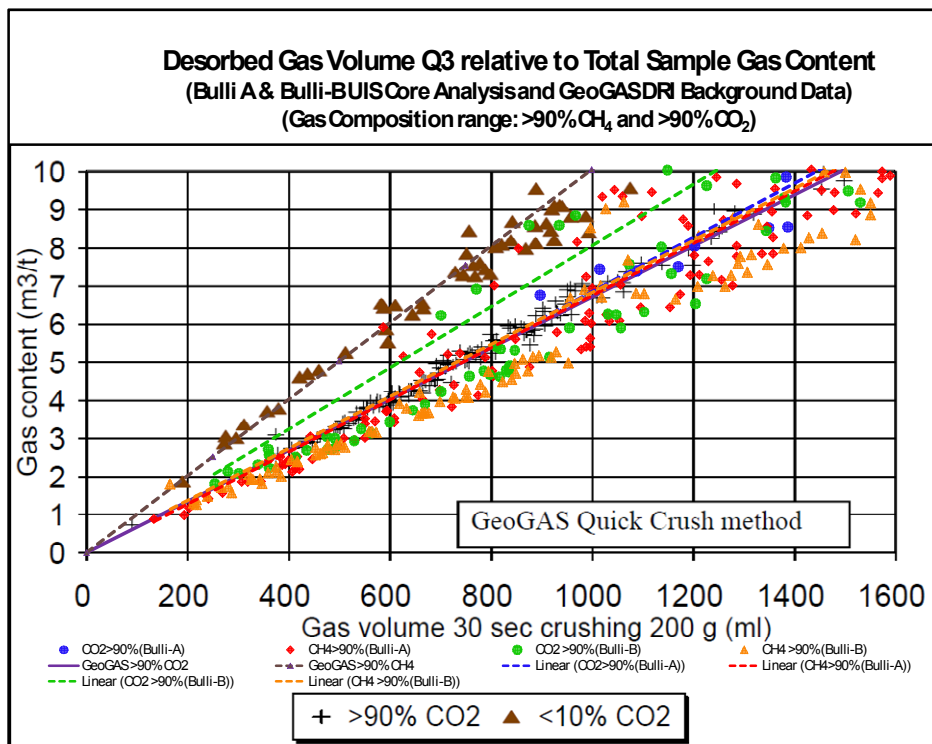


Figure 5 - Mine A and Mine B gas desorption / gas content data relative to Williams & Weissman (1995) data

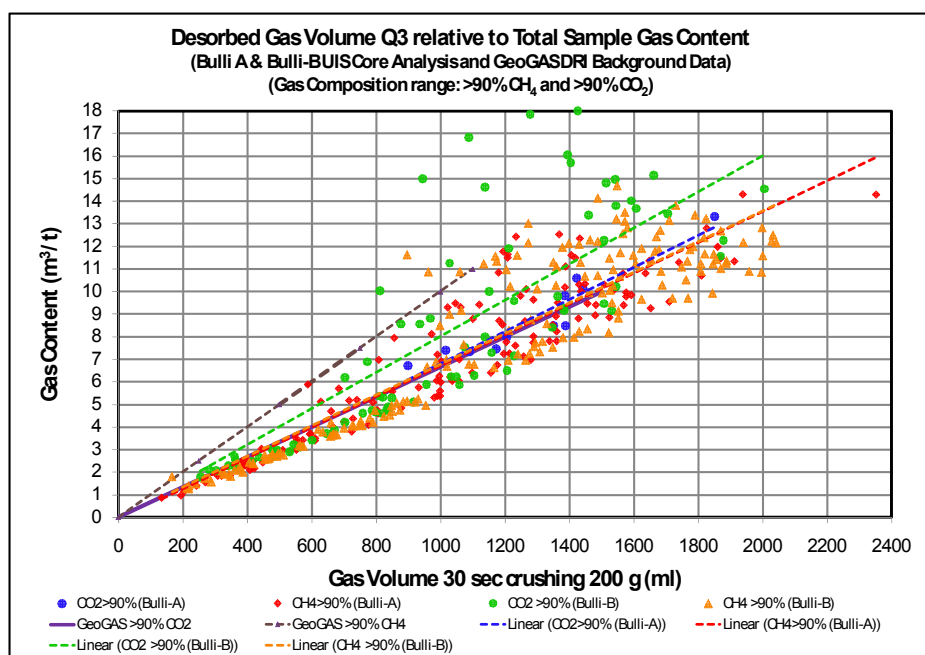


Figure 6 - Complete Mine A and B gas desorption / gas content data (930 points) relative to Williams & Weissman (1995) data

Considering the data presented in both Figures 5 and 6, as the basis for determining the desorption rate, which is applicable to the Bulli seam for given outburst threshold limits, it can be concluded that particularly in the case of CH₄, the desorbed gas volume will be somewhat higher than a DRI of 900 and will likely be somewhere in the range of 1400 to 1800, depending on the actual gas content threshold limit.

Additional data is now being sought from other Bulli and Non-Bulli seam coal mines to further investigate and analyse the extent of the gas desorption relationships which exist both within and between coal seams.

CONCLUSIONS

This analysis provides an interpretation of the process which led to the specification of outburst threshold limits applicable to mines operating in the Bulli seam of New South Wales. Given the work reported by Lama, it is evident that these threshold limits were potentially very conservative and incorporated quite high factors of safety. Given the loss of life resulting from outburst at the time and the general lack of understanding of the outburst phenomenon implementing conservative thresholds was assured of preventing further outburst related fatalities. This conservative approach to outburst threshold determination has also been applied to non-Bulli seam mines through the use of the GeoGAS DRI900 methodology.

In the fourteen years following the specification of outburst threshold limits there have been no reported outbursts in mines operating in the Bulli seam, where the gas content has been reduced to below the prescribed threshold limit. Two Bulli seam collieries have completed formal reviews of their respective outburst risk which resulted in increasing their threshold limits. Both Collieries have been operating safely, without outburst, for some four years under the increased threshold limits.

Gas is accepted as the primary risk factor associated with outburst and it is for this reason that gas drainage will for the foreseeable future be an integral part of outburst risk control and management. However unless properly controlled and managed it is possible for gas drainage to be quite ineffective.

Therefore the effective and efficient drilling and removal of gas from coal seams ahead of mining not only supports increased outburst threshold levels but also offers benefits such as reduced production delays, increased utilization of available coal reserves, reduced gas loading of mine ventilation air and, if suitable reticulation and utilisation facilities exist, reduced greenhouse gas emissions.

More research is required to improve the industry's understanding of the mechanisms that control gas storage, transport and drainage from coal, not only to better understand and manage the outburst risk, but for further improvement of mining related gas emissions reduction both into the ventilation network and into the environment.

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GAS CONTENT ESTIMATION USING INITIAL DESORPTION RATE

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ABSTRACT: The measurement of gas content plays an important role in mine safety and mine planning for coal and gas recovery. A number of methods exist to determine gas content; direct and indirect methods. The direct method of fast desorption test is the preferred method of gas content measurement. The indirect methods are based on either empirical correlations or laboratory derived sorption isotherms. Recent research has identified two new, semi-direct methods of estimating total gas content using early stage gas desorption rate measurement. Both techniques, if adopted, can provide operators with an indication of gas content and particularly whether the content is above or below the outburst threshold limit. A total of 930 samples, were analysed from two local mines, with known gas drainage problems and high degree of variability in both the insitu gas content and composition. Two specific aspects of the analysis included; the relationship of the three gas content components, Q1, Q2 and Q3, and the initial gas desorption rate relative to total gas content. Based on the relationship between desorption rate and total gas content, it was possible for minesite technical staff to provide operational personnel with an estimate of maximum expected total gas content from a particular core sample, based on the initial desorption rate value determined from Q2_{field} measurement data collected by the drillers or site geologists.

INTRODUCTION

The measurement of gas content plays an important role in mine safety and mine planning as well as coalbed methane resource assessment and recovery operations. A number of methods are available to determine gas content, direct methods, which measure the volume of gas released from a coal sample sealed in a desorption canister, and indirect methods, which are based on either empirical correlations or laboratory derived sorption isotherms. In Australia the direct method, fast gas desorption test, is the preferred method of gas content measurement used to support the underground mining industry. The benefit of this method is the relatively short time required from core recovery to the reporting of gas content and composition. This is particularly important from the point of view of outburst risk management and control as it minimises the risk of production delays whilst awaiting confirmation that a particular mining area is 'below threshold' and therefore able to be authorised to resume mining. There are however a number of shortcomings with this technique, the most significant being the accurate estimation of the gas lost during the core recovery process.

Recent research involving the analysis of results from rapid desorption gas content measurement of coal samples recovered from underground drilling in two Bulli seam coal mines has identified two new, semi-direct methods of estimating total gas content from early stage gas desorption rate measurement. The use of these techniques to estimate total gas content, although not definitive, provide operators with an indication of gas content and particularly whether the content is (a) above threshold limit, in which case additional gas content reduction action must be taken; or (b) below threshold limit, in which case planning can commence for the continuation of production, pending the receipt of validated gas content measurement from the laboratory rapid desorption testing.

It is widely accepted that the direct method of gas content determination is the preferred method and provides a more accurate result when compared to indirect methods. The use of indirect methods such as sorption isotherms possess many inherent potential errors which include:

- available isotherm data not being representative of conditions at sample location due to change in coal characteristics and gas composition;
- laboratory procedures used to determine isotherm data not being representative of insitu conditions;

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- inaccuracy in measuring gas pressure at sample location; and
- inability to determine the degree of saturation.

Although having the potential to produce a more accurate gas content measurement the direct methods also possess several areas where error may arise which include:

- inaccuracy in the estimation of the gas lost from the samples during sample recovery, prior to sealing of the sample in an air tight canister;
- inaccuracy in the field measurement of initial gas desorption rate;
- leakage of gas from the sealed canister;
- loss of gas due to dissolution when in contact with water;
- loss of gas whilst transferring sample from sealed canister for Q3 testing; and
- lack of control and inaccurate measurement of environmental conditions throughout the test.

Since the development of the first method in France by Bertard, et al. (1970), a variety of direct gas content measurement techniques now exist:

- a) Bertard's method;
- b) US Bureau of Mines method;
- c) US Bureau of Mines modified method;
- d) Smith and Williams method;
- e) Decline curve methods;
- f) Gas Research Institute (GRI) method; and
- g) Australian Standards (AS3980-1999) method

The current direct method used in Australia was initially introduced in 1991 as a standard method of testing. This was revised and a new standard published in 1999. This standard (AS3980-1999) provides a guideline for both fast and slow desorption techniques. The fast desorption method is the most common method employed because of the significantly reduced time required to produce a result. Besides the test direction, there is a significant difference in the relative percentage of the Q1, Q2 and Q3 components of the total gas content of a particular coal sample.

Testing and analysis undertaken on a number of Bulli seam coal samples have shown the component percentages for fast desorption (Equation 1) and slow desorption (Equation 2).

$$\text{Total Gas Content (Fast Desorption Method)} = (5 - 10\%) Q1 + (12 - 17\%) Q2 + (73 - 83\%) Q3 \quad (1)$$

$$\text{Total Gas Content (Slow Desorption Method)} = (5 - 10\%) Q1 + (75 - 90\%) Q2 + (5 - 15\%) Q3 * \quad (2)$$

Q3* in the slow desorption equation represents a true residual gas content which ranges between 0.7 and 1.0 m³/t (CH₄) and 1.5-1.9 m³/t (CO₂).

The results of gas content testing on coal core samples obtained from underground to in-seam drilling were provided by two collieries operating in the Bulli seam. The large number of samples, 516 from Mine A and 414 from Mine B, were considered to be highly representative of conditions present in the mine given the core sample locations covered a large area both across and along multiple longwall blocks. Throughout both of these large areas there is a high degree of variability in both the in-situ gas content and gas composition. During the analysis of these two separate datasets a number of relationships and similarities became evident. Two specific aspects of the analysis presented include the relationship to total gas content of both the three gas content components, and the initial gas desorption rate. The impact of seam gas composition is also discussed.

GAS CONTENT DETERMINED FROM Q1 MEASUREMENT

Analysis gas content test results was undertaken with particular emphasis on determining whether a relationship existed between the three components which make up total gas content. Using the samples from both Mine A and B, the analysis identified that each of the three gas content components, Q1, Q2 and Q3, represent a relatively consistent percentage of the total gas content, particularly below a total gas content of 6-8 m³/t. It was observed that above 8-10 m³/t there was an increase in the data scatter away from the mean with a general increase in the percentage of gas

liberated during the early stages of desorption (i.e. Q_1+Q_2), increasing relative to the decreasing Q_3 . Figure 1 shows the volume of gas released for each of Q_1 , Q_2 and Q_3 relative to the total content of each sample analysed for both Mines. The increased scatter above $10 \text{ m}^3/\text{t}$ is clearly evident in both datasets. In order to reduce the impact of the high degree of scatter, the data with total gas content less than $10 \text{ m}^3/\text{t}$ have been averaged and trend lines generated.

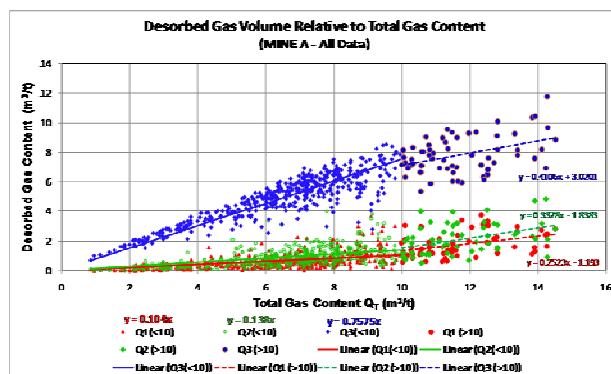


Figure 1a - Gas content component volumes measured determined during fast desorption testing (Mine A)

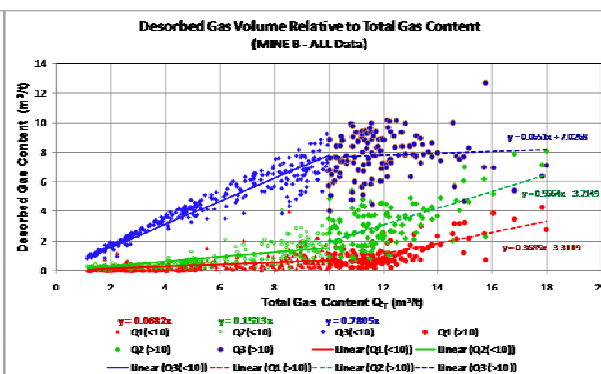


Figure 1b - Gas content component volumes measured determined during fast desorption testing (Mine B)

In the case of Mine A, Q_1 represents 10.4% of Q_T , Q_2 is 13.8% Q_T and Q_3 is 75.8% Q_T . In the case of Mine B, Q_1 represents 6.8% of Q_T , Q_2 is 15.1% Q_T and Q_3 is 79.0% Q_T . In both cases, and with Q_T above $10 \text{ m}^3/\text{t}$, there is a significant increase in the rate of early stage desorption with high gas volumes liberated during Q_1 and Q_2 with a corresponding reduction in Q_3 emission. The relative component percentages of total gas content are summarised in Table 1

Table 1 - Relative gas content component percentages of total gas content

Average Component Percentage of Total Gas Content	MINE A	MINE B	Diff (% Q_T)
Q1 (% of Q_T)	10.40	6.82	3.58
Q2 (% of Q_T)	13.80	15.13	1.33
Q3 (% of Q_T)	75.80	78.05	2.25

Using the observed relationship, it is possible for minesite technical staff to estimate the expected average total gas content for a particular core sample, based on the Q_1 component value determined from $Q_{2\text{field}}$ measurement data collected by the drillers or a site geologist.

Further investigation was undertaken to determine the extent of any impact which gas composition may have on the observed relationship. The datasets from each mine were divided into subsets comprising all samples with gas compositions greater than 80% CH_4 and greater than 80% CO_2 . A total of 305 samples were analysed in the case of Mine A and 297 in the case of Mine B.

A comparison of the component relationships for each mine, shown in Figure 2, indicates that gas composition has little impact on the relative percentage of the gas component volumes released during gas content measurement using the fast desorption method. In the case of Mine A (Figure A), although the relative percentage of total gas content of each of the three components is virtually independent of gas composition the average result for the two gas composition datasets indicates that the desorbed Q_1 and Q_2 components are marginally higher for the high CH_4 samples and lower Q_3 component than the high CO_2 samples. Although quite small, the relative difference in component percentage of total gas content between the two gas composition datasets for Mine B (Figure B) does indicate a greater difference than was observed for Mine A. In the case of Mine B the desorbed Q_1 and Q_2 components are greater for the high CO_2 samples with the Q_3 component being less for the high CO_2 samples than for the high CH_4 samples.

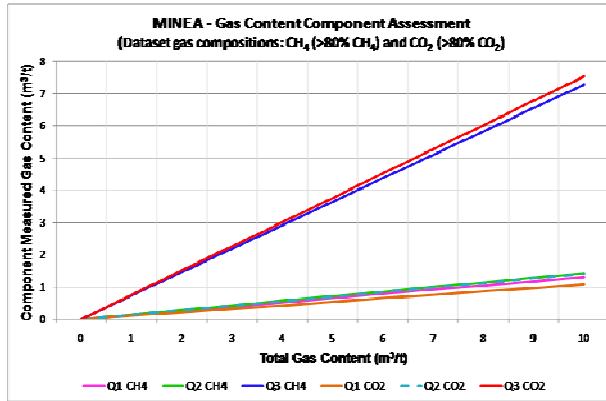


Figure 2a - Impact of gas composition on gas content component volumes (Mine A)

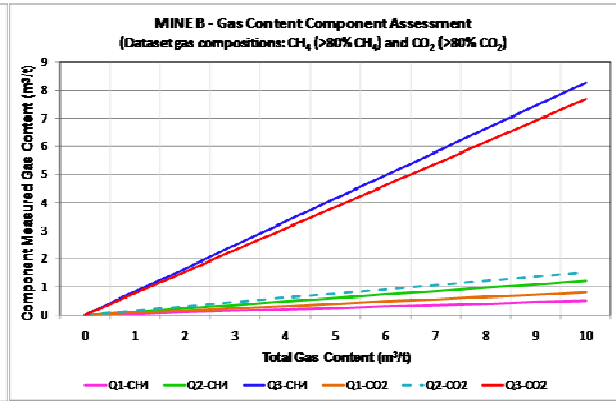


Figure 2b - Impact of gas composition on gas content component volumes (Mine B)

GAS CONTENT DETERMINED FROM INITIAL DESORPTION RATE

The analysis of the data from the two mines was extended to include the initial desorption rate with particular assessment of the desorption rate relative to the total gas content of the samples from each mine. The data obtained from each mine is illustrated in Figure 3. The data indicates quite a high degree of scatter which can occur for various reasons including:

- Where a sample is highly fractured an increased gas emission is likely to occur early in the desorption process resulting in increased desorption rate relative to total gas content;
- Where a core sample remains intact a reduced gas emission is likely to occur early in the desorption process resulting in decreased desorption relative to total gas content; and
- Where leakage of gas from the desorption canister has occurred between sealing in the field and laboratory testing the total measured gas content will be low relative to the initial desorption rate.

With the exception of leakage, scatter among the results is expected given the lack of control on the condition of the recovered core. The impact of leakage can be minimised through equipment design, maintenance and operator training.

Interestingly, all data points lie below an observed maximum envelope. A projected log relationship through the identified maximum total gas content values for each of the two data set, as shown on the two graphs, clearly indicate that the envelope line represents the maximum gas content value expected for a given desorption rate value.

Figure shows the maximum gas content / desorption rate curves for both mines. It can be seen that the two curves are very similar in shape with the Mine B curve indicating a slightly higher total gas content than Mine A for a given desorption rate value.

Using this observed relationship between desorption rate and total gas content minesite technical staff are able to estimate the maximum expected total gas content for a particular core sample using the initial desorption rate value, determined from $Q2_{\text{field}}$ measurement data collected by the drillers or a site geologist.

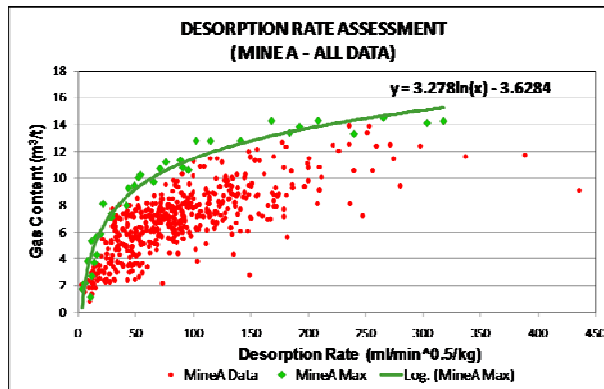


Figure 3a - Initial desorption rate measurement relative to total core sample gas content (Mine A)

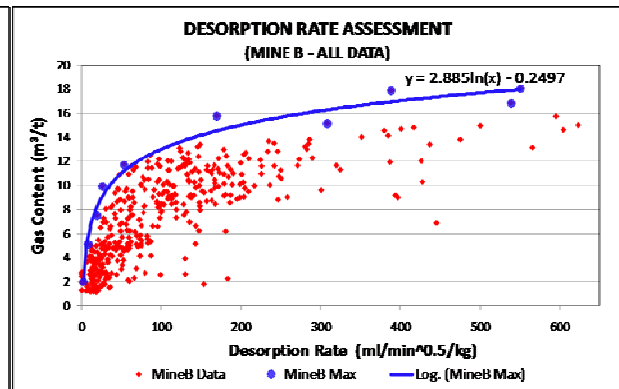


Figure 3b - Initial desorption rate measurement relative to total core sample gas content (Mine B)

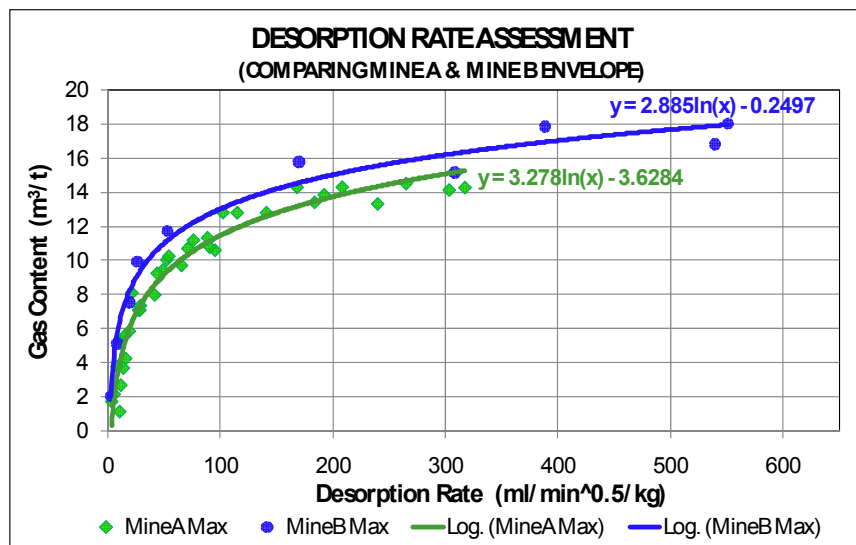


Figure 3 - Determination of maximum total gas content based on initial desorption rate measurement

CONCLUSIONS

Using the fast desorption method of determining the gas content of coal based on AS3980-1999, a number of important relationships have been identified, which have the potential to assist mine operators in the rapid estimation of the gas content of a given sample literally while the sample is in transit to the laboratory for testing.

A relationship was identified which demonstrated, for gas content values below 10 m³/t, that each of Q1, Q2 and Q3 maintained a consistent percentage relative to the total gas content. The composition of the gas within the sample had very little impact on this relationship. Thus it can be concluded that during fast desorption testing, the percentage of the desorbed gas components relative to total gas content will remain reasonably consistent and this relationship is maintained independent of gas composition.

There was increased scatter of the desorbed gas component values above 10 m³/t, with the scatter being more pronounced in the case of Mine B. This scatter indicates that a greater percentage of total gas content is released during early stage desorption in those samples with high total gas content. This trend was unexpected and should be further through extending the analysis to include additional Bulli seam and non-Bulli seam mines.

Further analysis of the initial gas desorption rate relative to the total gas content identified a maximum total gas content envelope which represents the maximum gas content for a given sample desorption rate.

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COAL RESERVOIR PARAMETERS REGULATING GAS EMISSIONS INTO AND FROM COAL MINES

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ABSTRACT: A few number of gas reservoir parameters regulate the intensity and the extent of gas emissions during and following the mining of coal. Gas content is one of the most important of these parameters. Depending on the purpose of its quantification its accurate determination could be vital to the mining activities. For instance, if this parameter is used to evaluate the outburst and its value falls near the threshold limit it needs to be accurately measured. Similarly when seam gas emissions from coal mines, is to be calculated, an accurate measurement of this parameter in a carbon constraint economy has a very important economic impact. The other challenge associated with gas content is the lower limit of measurability of the standard systems. For instance for low to very low gas content ($<0.1 \text{ m}^3/\text{t}$) encountered in 'non-gassy' underground and open cuts, the standard method is unable to deliver accurate values. A different methodology is then required to evaluate the gas content in these conditions.

Another parameter, important in evaluation of the intensity of the emissions and its time dependency nature is the gas diffusivity parameter. While the saturation indicates the onset of gas desorption, the diffusivity parameter controls the rapidity of gas movement from the micro storage sites into the larger fractures and voids. Diffusivity is, therefore, the primary rate limiting factor in the intensity of gas emissions. The diffusivity is not often measured directly and a diffusion time constant, called Tau, is often used to indicate the speed of diffusion flow in coal. This parameter is also used in numerical simulators which use a simplified model of gas diffusion, namely pseudo state diffusion models.

This paper discusses the current and new methodologies to determine the main parameters of coal reservoirs including gas content and gas diffusivity and potential errors associated with current measurement methods.

INTRODUCTION

Coal is a porous rock and can contain large volumes of gas, hence it is considered a gas reservoir. The major components of coal seam gas are methane (CH_4) and carbon dioxide (CO_2). Subsidiary volumes of ethane (C_2H_6) and higher hydrocarbons (C_{2+}), and nitrogen (N_2) can also be present in some coal seams (C_{2+} at high depths).

Gas currently present in a coal seam can be of primary or secondary origin. Primary gas has been generated as a by product of the coalification process during which large volumes of CH_4 and CO_2 are produced (thermogenic gas). Some volumes of gas generated would be adsorbed by coal but most would escape the site.

Methane gas can be also generated within the coal seam, as a result of microbial activities. In this case coal seams have to act as a permeable aquifer, allowing the movement and storage of the methanogenous micro organism and nutrients. Thus, most of the coal seam methane at fairly shallow depths is of biogenic origin. Igneous activities over geological time have also resulted in the injection of CO_2 into coal seams replacing methane in some places.

Coal seam gas is stored in pore volume and surface spaces in free and adsorbed phases. The greatest portion of the stored gas, at shallow to medium depths, is held on the pore surfaces in adsorbed phase. The largest part of the pore surfaces are located in the micro pores system (size $<2 \text{ nm}$) which are only a few times larger than the coal seam gas molecular sizes. The adsorbed phase storage in the micro pore system follows a pore filling mechanism and therefore reaches its maximum value (adsorption capacity) when the pore system is fully filled. The stored gas is then retained due to a combination of adsorption and capillary forces.

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Emissions from coal are intensified during mining due to generation of multitude of fractures and fissures within the coal seam and in the strata above and below the coal mined seams. Total volume of gas liberated depends basically on gas content of coal. The rate of gas liberation, however, depends on both gas content and gas diffusivity of coal.

The methodologies used and the accuracy of measurement of these two parameters, i.e., gas content and diffusivity, would influence the results and the estimation of emissions. In particular in case of coals of low gas content ($<0.1 \text{ m}^3/\text{t}$), the relative error of the measurement can be very high because the lower limit of measuring system is attained. While measurement of low gas content may not be of any importance to safety issues in underground mining it is of quite importance for greenhouse gas emissions inventory which would be required in carbon retrained economy of the near future.

This paper describes the general mechanism of diffusion in coal and methodologies and accuracy issues inherent to the current methods of measuring the desorption and gas content of coal and the effect of gas diffusivity.

GAS DIFFUSION IN COAL

Gas is diffused in coal under the forces of gas concentration gradient. The desorption of gas is limited by the diffusivity property of coal. A higher diffusivity allows faster desorption of gas from coal. The diffusivity affects the evaluation of gas content particularly the estimated value of the lost gas during drilling and at the surface. Diffusivity is also an important input to gas reservoir models which is directly used or indirectly in terms of a diffusion time constant τ (Kolesar et Ertekin, 1986; King et al, 1986; King, 1993).

Gas diffusion in coal can be mathematically expressed by Fick's law and by assuming that the change in gas concentration per unit of time and in a unit volume of the medium (coal) is equal to the difference between the volumes of gas diffused into and out of the elemental volume of the medium. In its general form it is written as,

$$\frac{\partial c}{\partial t} = \text{div}(D \cdot \nabla c) \quad (1)$$

where c is the gas concentration (gas content) and D is the diffusion coefficient (or diffusivity of gas in coal). Solution of Eq(1) is not straightforward and often numerical methods should be employed in particular for cases where D changes in time and space or the medium is of complex layout (for exact solutions see Crank, 1975).

Barrer and Brook (1953) investigated the molecular diffusion and adsorption of gases in powdered zeolites. They found that irrespective of shape of the powder (cube, parallelepiped, sphere or cylinder) the diffusion in early stages of gas flow can be explained by,

$$\frac{Q}{Q_m} = \frac{2A}{V} \sqrt{\frac{Dt}{\pi}} \quad (2)$$

where Q is the volume of gas desorbed since the start of diffusion, Q_m is the total gas initially contained in coal and t is the time elapsed. A and V are the surface area and volume of powders.

Some researchers have used this equation for gas desorption from coal at early stage of desorption. Gunther (1965) used this equation for gas desorption up to release of 20% of the total gas in coal. Others extend the use of the equation for up to 50% of total gas initially in coal (Smith and Williams, 1984).

For spherical grains, this equation can be simplified to,

$$\frac{Q}{Q_m} = \frac{6}{\sqrt{\pi}} \sqrt{\frac{Dt}{a^2}} \quad (3)$$

where a is the radius of spheres approximating coal grains (half of the average cleat spacing). The diffusion coefficient is expressed in unit of square m per second (m^2/s) and radius is in meter (m).

A full and exact solution for the unsteady diffusion of gas in coal can be obtained provided the assumption of uniformity of the diffusion coefficient in space and time and that coal can be seen as an

assembly of spherical grains. These assumptions are justified by the fact that coal is highly and uniformly fractured (cleats) so that the matrix can be thought of spheres delineated by cleats. The solutions have been given various authors (Carslaw and Jaeger, 1959; Crank, 1975) and have been used by coal workers (Walker and Mahajan, 1978; Smith and Williams, 1984). The cumulative volume of gas desorbed from coal, using these solutions, is,

$$\frac{Q}{Q_m} = 1 - \frac{6}{\pi^2} \sum_{n=1}^{\infty} \frac{1}{n^2} \exp(-n^2 \pi^2 t / \tau) \quad (4)$$

where τ is the diffusion time constant, defined as,

$$\tau = \frac{a^2}{D} \quad (5)$$

The parameter τ (Tau) is an important property of gas and coal and often is used to replace the diffusion coefficient when the direct measurement of D is not possible. Intuitively Tau can be thought as the time required for the diffusion flow to advance a distance 'a' in the porous medium. This parameter is used in many of gas flow simulators where a pseudo steady state diffusion flows approximates the true unsteady diffusion flow considered. In a full unsteady diffusion flow model the diffusivity D can be directly used to estimate the free gas volumes released from coal matrix into fractures.

MEASUREMENT OF GAS CONTENT OF COAL

Gas content is measured using either a slow desorption or a fast desorption method. Both methods have been used in various forms over the years (Bertard et al, 1970; Kissell et al, 1973; Williams et al., 1992; Diamond et Schatzel, 1998; Saghafi et al, 1998; Australian Standard, 1999). Though both methods consist of similar steps to determine the gas content coal, the length of the procedure is significantly longer in slow desorption method. In fast desorption method the time of testing is significantly reduced by accelerating the desorption rate (diffusivity). Coal is crushed and all gas is released in space of an hour or two long before if it were naturally to desorb its gas.

The slow desorption and current Australian fast desorption methods are both based on measurement or estimation of volume of gas desorbed from coal in several stages. For fast desorption there are three stages which delivers the three components of the 'measured' gas content. In slow desorption the last stage may not exist depending on whether a residual gas content testing is required or not.

The three components of gas content in the slow desorption method correspond to three regimes of gas desorption, i.e., 1- loss gas (initial desorption), 2- desorbed gas and 3- residual gas. In the fast desorption method, however, these stages are basically three steps in gas content testing and are not related to gas desorption kinetics of coal. The three components of gas content measured are commonly represented by Q_1 , Q_2 and Q_3 terms. The 'measured gas content', Q_m , is the sum of the 3 components (Australian Standard, 1999),

$$Q_m = Q_1 + Q_2 + Q_3 \quad (6)$$

The Q_1 or the lost gas is the volume of gas desorbed from coal during the drilling and prior to its seal in gas tight canisters. This stage is identical for the two methods. The Q_2 is the gas desorbed during transport and in the lab. It is called desorbed gas and is the main component of the gas content in slow desorption method. For this method this stage is allowed to continue until no further measurable gas desorption is observed. In fast desorption method Q_2 step is generally short as coal may be crushed any time depending on the availability of measuring system and proper conditions. The last component of gas content is Q_3 which is the gas desorbed from crushed coal. Q_3 measurement is the most important stage of gas content testing for the fast desorption method. Coal gives away most of its gas in this stage. For slow desorption this stage is often of no importance as Q_3 is expected to be low (very low residual gas content).

The measurement of the volume of gas released in the three stages is usually done by using a measuring cylinder. The released gas is admitted into a water filled inverse cylinder. The displacement of water provides the measure of the volume. This system has worked well over the years and is used

routinely in Australia. There are, however, some problems with this way of measuring the volume including gas partial pressure effect and dissolution of gas in water. These have been addressed over the years and improvements have been suggested and applied (Saghafi and Williams, 1998; Saghafi et al, 1998; Danell et al, 2003).

The volume of gas released from a coal sample can be low either because the low gas content or because of the small sample size. The latter may happen when the cost of coring is high (deep coal) and/or a large part of the core section is used for other more urgent testings including quality and geotechnical measurements. In these cases the method of measurement of volume by using water displacement could not be applied.

To illustrate the limit of measurability using the standard method it should be noted that the smallest volume that can be confidently measured by using the water displacement in measuring cylinder, is about 2-5 cm³. Therefore for a sample of 100 g size the method can measure low gas content of about 0.02 - 0.05 m³/t. These are the very real limits of measurement and though they have no impact on gas content testing for safety purposes they have important impact on coal mining economics whence the mitigation of greenhouse and carbon tax are introduced into the mining economy.

The other issue is the estimation of Q_1 which is based on initial desorption measurement in the field. Theoretically the measurement should start as soon as the core is retrieved from the exploration gas hole and visually logged by field geologist. However, this is not always the case and the kinetics of desorption may change if the delay is large. In addition there are debates on the effect of measurement temperature and whether the in-situ temperature should be used.

In the next section these questions are discussed and methodologies for overcoming the issues are suggested.

Measurement of low gas content

For very low gas content coals ($Q_m < 0.1$ m³/t) or when the coal mass available for crushing is very small the water displacement method of measuring the volume is not adequate. In such cases normally there would be no measureable Q_1 and often no measurable. Coal is crushed as soon as it reaches the gas lab and residual gas (Q_3) is determined. This is the case for most of non-gassy underground mines and surface mining.

For measurement of low gas content (non-gassy coals) the best practice is to seal the fresh sample in purposely gas tight canister in the field, and then dispatch it to the lab for crushing. Ideally coal should be sealed in a canister which can be directly mounted on the crusher so that there would be no need to open the coal canister before crushing. The total desorbed gas can then be indirectly evaluated using a gas composition testing method.

The indirect method of determining gas volume by measuring gas composition had been used over the years in some old coal mining countries in Europe. The method was used to obtain the 'total gas content' which is theoretically larger than the 'measured gas content' and includes post Q_3 component of gas content. The method consists of keeping the crushed coal in the crusher container for sometime after the completion of crushing. Then gas composition in crusher canister is measured. The volume of desorbed gas is determined from the knowledge of void volume in the crusher canister and gas composition values.

This method was used by CSIRO in the course of a number of ACARP projects to deliver the total gas content. This new component of gas content is called Q_3' (Q_3 prime)

For routine measurement of low gas content of coal we suggest a similar approach. The set up is conceptually illustrated in Figure 1. The crushing canister is initially flushed with nitrogen and then coal is placed in the crusher and crushed. After the completion of the crushing and allowing time for temperature equilibrium, the canister is opened to a closed circuit with an in-line pump. A gas sample is collected from the system after a sufficient period of time and gas composition is measured using gas chromatography. Knowing the volume of the total void space in the system the gas content is determined.

The lower limit of gas content which can be determined using this method can be evaluated from the knowledge of void volume in the system (crusher and piping) and the lowest or optimal lower limit of gas chromatography in use. For instance if the volume of void is about 500 cm³ (typical void in the CSIRO quick crush canister for a 100g coal sample), and a GC which can measure accurately a concentration of 100 ppm of methane (many GC's can measure concentration values below 10 ppm of methane) is used, then the gas content of about 0.0005 m³/t can be determined. This method, therefore, can measure gas content values of at least 100 times smaller than the standard method.

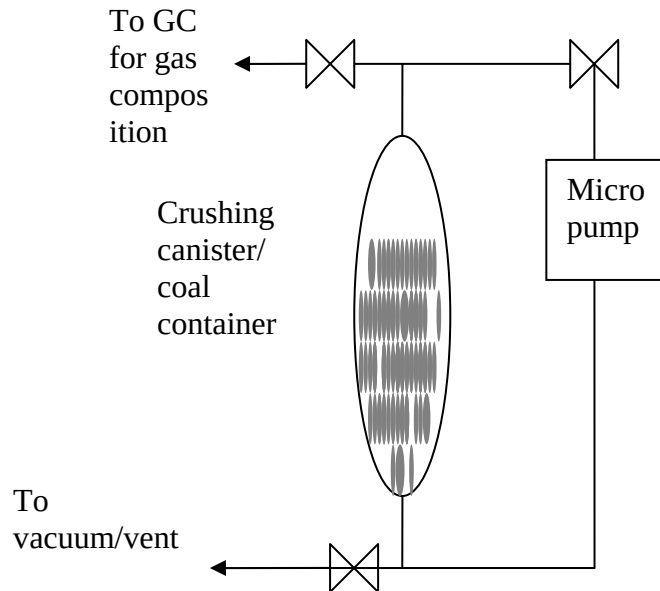


Figure 1 - Schematic diagram of measurement of residual gas content for very low gas content coal

Estimation of lost gas, Q_1

The Q_1 component of gas content is determined by extrapolating back the gas desorption curve to the time when the drill bit hit the coal. Measurement of initial desorption is undertaken in the field as soon as the core is available after its retrieval from the borehole.

Based on the discussion in previous sections in the early stages of gas desorption the cumulated volume of gas follows a linear equation of square root of time. As discussed this linearity is analytically demonstrated from the solution of the equation of gas diffusion for broken spherical pieces of coal, i. e.,

$$\frac{Q}{Q_m} = \frac{6}{\sqrt{\pi}} \sqrt{\frac{t}{\tau}} \quad (7)$$

where τ is the diffusion time constant or Tau. If the cumulative volume of desorbed gas is plotted against the square root of time, generally the desorption curve has the following mathematical expression,

$$q(t) = k\sqrt{t} - Q_1 \quad (8)$$

In this equation $q(t)$ is the volume of gas desorbed since the start of measurements but t is the time since the start of desorption in the borehole (in practice time zero is the mid time between the time the drilling hit the coal and the end of the coring run). From Eq (8) at time t_0 , $q(t_0) = 0$. Therefore the lost gas Q_1 which obtained from the intercept of the regression line; $Q_1 = -q(0)$. The lost gas can also be obtained from the slope of the regression line,

$$Q_1 = k\sqrt{t_0} \quad (9)$$

t_0 is the lost time or the time elapsed since the drill bit had hit the sample in the borehole until the sample is sealed in the canister at the surface for desorption measurement. The released gas during the field measurement is added to the Q_2 component of gas content.

In Figure 2 a typical field measurement of the initial gas desorption rate for estimation of lost gas is shown. In this case some 20-25 minutes (t_0) had passed before the sample could be sealed and measured.

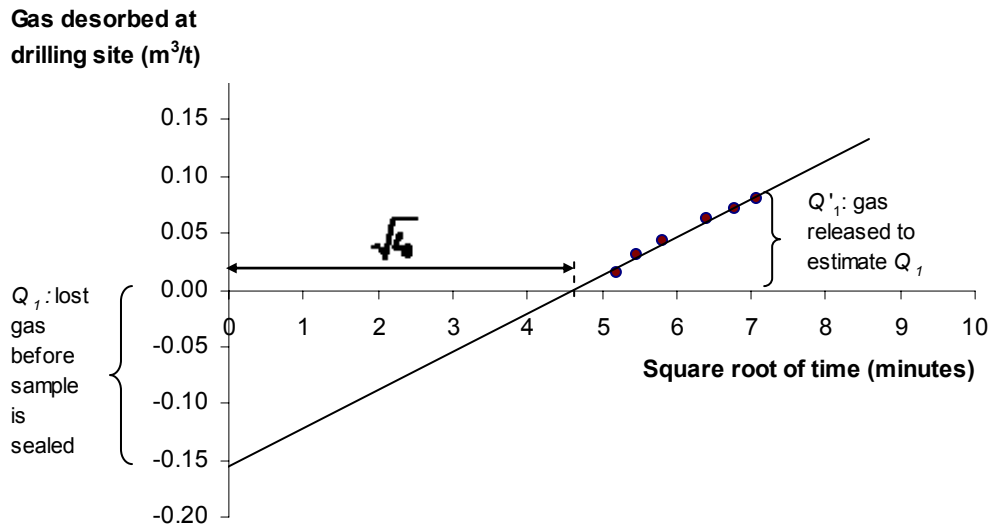


Figure 2 - Measurement of initial desorption rate for estimation of lost gas (Q_1)

Note that initial desorption rate can be used to estimate the diffusion time constant τ . This is possible after all three component of gas content are evaluated,

$$\tau = \frac{36}{\pi} \left(\frac{Q_m}{k} \right)^2 \quad (10)$$

In the above equation Q_m is the 'measured gas content' ($Q_m = Q_1 + Q_2 + Q_3$)

Accuracy of estimating Q_1 from the initial desorption rate data

Two of the concerns that have been raised in relation to the accuracy of estimating the lost gas are - the maximum length of the time that can be tolerated before starting the measurement (t_0 time) and - at what temperature the field measurement of desorption rate should be carried out.

Effect of the length of the lost time, t_0

The estimation of Q_1 is based on the assumption that desorbed gas volume is a linear function of the square root of time and therefore the lost gas can be estimated by extrapolating back the regression line. However, Eqs. (7) and (8) which are used for the extrapolation are only valid for short values of t_0 or more accurately for small values of t_0/τ . Therefore, the length of the lost time (t_0) would directly affect the magnitude of the error. The error of estimation is greater for larger t_0 . The acceptable values of t_0 depend primarily on the diffusion time constant (τ). This constant in turn depends on diffusion coefficient and fracture/cleat spacing. Imposing a condition of validity for use of these equations such as $Q/Q_m < r_0$ then the maximum value of t_0 for each individual case can be assessed.

Effect of the temperature of measurement

The temperature of desorption measurement at the drilling site is believed to affect the results. Some authors (Mavor and Pratt, 1996) recommend measuring the gas desorption at in-situ coal temperature. This, however, can be a source of error by itself. This is because gas would not desorb fully until coal is pulled out from water. Secondly as soon as gas starts desorbing the temperature rapidly falls the core is being pulled out from the borehole. Hence, the measurement of desorption rate at in-situ

temperature can falsely increase the value of the lost gas content. The effect of temperature on desorption rate is due principally to effect of temperature on the diffusivity of gas in coal. It can be shown that the relative error in Q_1 estimate is equal to half of the relative variation in diffusivity, i.e.

$$\frac{\delta Q_1}{Q_1} = 0.5 \frac{\delta D}{D} \quad (11)$$

The potential error of determination of Q_1 due to using a different temperature can be estimated by studying the effect of temperature on gas diffusivity.

DIRECT MEASUREMENT OF GAS DIFFUSIVITY IN SOLID COAL AND CALCULATION OF DIFFUSION TIME CONSTANT (TAU)

Gas diffusion coefficient can be indirectly estimated either from desorption curves generated in slow desorption method of gas content testing or from desorption isotherm data. In both cases numerous assumptions are required. For pulverized coal used in sorption tests the indirect results of diffusivity may differ considerably from gas diffusivity for solid coal. The diffusivity, however, can be directly measured. Recently new methods are presented (Saghafi et al, 2007) allowing direct measurement of diffusivity of coal. In these methods gas is flown through solid coal by maintaining a gas concentration gradient across a coal disk of small thickness (<5mm). The small thickness allows the diffusion test to take place in reasonable time (about a week). The diffusivity obtained from this method can be used in evaluating the diffusion time constant tau. For full unsteady state models the diffusion coefficient can be used in the models.

Estimation of Tau from gas content data

Tau (τ) is physical parameter related to diffusion rate and in the absence of a direct measurement of diffusion coefficient can be used in simplified diffusion flow models to simulate the flow of gas from microspores into fractures. It also gives a feel for the speed of diffusion of gases in coal. Tau can be derived from initial desorption curve which is established in previous section Eq (10). The value of Tau (τ) obtained in this way can be used both in unsteady and steady models. This is also a economic way of measuring Q_1 . However if the gas desorption can not be measured at site because of logistics or very low gas content of coal the method can not be applied. A direct diffusivity test should then be undertaken in the laboratory.

The numerical models originated from petroleum and conventional gas industries often use a pseudo steady state diffusion mechanism presented initially by Warren and Root (1963). This is to simulate gas desorption from primary porosity (coal matrix) into fractures.

According to the pseudo-steady state diffusion model, the desorption/diffusion rate out of coal is proportional to the difference between the average gas concentration in the coal matrix and the gas concentration in the fractures, i.e.,

$$\phi = \frac{-D}{a^2} (c - c_f) \quad (12)$$

where ϕ is the gas desorption rate from a unit volume of matrix, c is the matrix average gas concentration and c_f is the gas concentration in fractures. D is the diffusion coefficient and a is the radius of coal grain (half of cleat spacing). Gas desorption reduces the gas content of matrix and the change in unit time should be equal to the flow rate of desorbed gas out of matrix, i.e.,

$$\frac{dc}{dt} = - \frac{c - c_f}{\tau} \quad (13)$$

Note that a^2/D in Eq (12) is replaced by its other representation, namely parameter τ . The solution of equation (13) yields the variation of gas concentration in the coal matrix as a function of time,

$$\frac{C - C_f}{C_m - C_f} = e^{-t/\tau} \quad (14)$$

C_m is the initial gas content of coal which should all fall to gas concentration in fractures if sufficient time is allowed.

In terms of the volume of gas released to total volume of gas initially in coal the above relation can be rewritten as,

$$r = \frac{Q}{Q_m} = 1 - e^{-t/\tau} \quad (15)$$

Based on Eq(15) when desorption time equals τ , the value of $(\tau)r$ would be ~0.63 which means that coal would release more than 63% of its initial gas after a time τ passed start of diffusion.

If gas desorption from coal follows a pseudo steady state mechanism then it is legitimate to evaluate the diffusion time constant τ from gas content testing data. In this case the time required for coal to release 63% of its total gas would be obtained from various gas content desorption curves data and an average value of Tau (τ) is determined. This method is, however, costly because the slow desorption measurement should be carried out on its totality which may take weeks or sometime months.

Some gas workers had presented empirical relations similar to the Eq (15). For example Airey (1968) had suggested an equation for emission from broken coals as follows,

$$r = \frac{Q}{Q_m} = 1 - e^{(-t/t_0)^n} \quad (16)$$

Based on the measurement of gas desorption for different rank coals these workers suggest n values varying from 1/3 to 1/2. Note that t_0 in Eq (16) is similar to τ in Eq (15).

CONCLUSIONS

Gas content is the most important parameter to be evaluated for any study of coal seam gas irrespective of the end use. For low gas content conditions the current methodologies are not accurate. While from a mine safety viewpoint the accuracy of measurement of low gas content is not an issue, it is of vital for calculation of the greenhouse gas emissions. Accurate measurement of gas content is required to calculate the emissions from 'non- gassy' or class B mines (underground) and open cut mines. Because of the large volume of coal mined any small error at measurement point can be magnified significantly in the final results of emission calculation. In the context of a carbon constraint mining economy, a new methodology of gas content measurement is required. A method of measurement for low gas content coals is presented. The method is based on the measurement of gas concentration rather than gas volume in the standard methods. The analysis of the method indicates that gas content of a hundred times smaller can be measured using the new method.

The importance of diffusion flow in gas content measurement and in particular in measurement of loss gas (Q_1) was discussed. The desorption rate is a function of diffusivity which is sensitive to temperature. However, measuring the initial gas desorption at in-situ temperature can be a source of larger errors and therefore overestimating the gas content.

The desorption time constant (Tau τ) was analysed and its relation to the diffusion coefficient (diffusivity) was discussed. It is possible to estimate this parameter from the initial rate of desorption which is required for estimation of Q_1 . A less accurate, and more costly, is to measure gas content using the slow desorption method and obtain a desorption curve. In this method Tau (τ) is the time

that 63% of total gas is desorbed. It was shown that Tau can also be determined from direct measurement of diffusivity of gas in solid coal.

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SORPTION CHARACTERISTIC OF COAL, PARTICLE SIZE, GAS TYPE AND TIME

Raul Florentin¹, Naj Aziz¹, Dennis Black¹ and Long Nghiem¹

ABSTRACT: Sorption characteristics of Bulli seam coal samples were examined with respect to coal particle size, gas type and time. Sorption tests were carried out by the indirect gravimetric method for determining gas content in coal. Various coal particle sizes were tested in addition to 54 mm diameter coal core. The other coal sizes were ± 15 mm (-5/8+0.530 mesh) cubical blocks and coal fragments of ± 1.18 mm (-0.256+16 mesh), ± 6.70 mm (-5/16+0.256 mesh), and ± 8.00 mm (-5/8+5/16 mesh). The samples were maintained in specially designed pressure vessels, at constant temperature of 24 °C and subjected to gas pressures up to maximum of 4000 kPa with incremental increasing steps of 500 kPa. All samples were tested with CH₄, and CO₂. The first group of ± 1.18 mm size coal fragments achieved the highest gas adsorption, and the lowest was 54 mm size. Adsorption of carbon dioxide was typically the highest and that of methane was lowest. Furthermore, the tests also showed that the longer it takes to reach the pressure equilibrium the higher the gas adsorption. This study suggests that the gas content in coal depends strongly on gas type, sorption time, and particle size. Sorption time however, appears to be independent of particle size.

INTRODUCTION

The phenomenon of gas sorption has been studied over a long period of time spanning over two centuries and as early as 1773 by C.W. Scheele. Later on in 1777, A.F. Fontana described the adsorption of gases, mostly in charcoal. Brunauer (1945) reported on the work of De Saussure in 1811 and his systematic measurement of gas adsorption on several adsorbents. One particular reference is rather significant from the sorption point of view, in that De Saussure refers to easily condensable gases being absorbed in largest quantities by the adsorbent. This agrees with the concept that the adsorption is due to the condensation of gases on the surface of the adsorbent while desorption is the evaporation of gases.

More recently gas sorption has been the subject of intense study Seidle and Huitt (1995), Moffat and Weale (1955), Jolly et al. (1968), and Harpalani and Chen (1995), Lama (1988), and many more. Singh (1968) observed that coal could hold 1.4 to 2 times more carbon dioxide than methane at about 345 kPa (50 psi) pressure and with nitrogen achieving only 0.4 of methane under the same conditions. However, Gunther (1965) reported that at a higher pressure these figures could be less.

While there appears to be enough information about the behaviour of gas adsorption mainly in pulverised coal samples, most of these studies were made on particles sizes up to a few hundred of microns and only few tests were made in sizes larger than a millimetre. Therefore, the aim of this study is to examine gas sorption in coal of particle sizes larger than one millimetre and to evaluate isotherms for a better understanding of gas sorption in coal. Accordingly, various particles sizes were prepared and tested.

PARTICLE SIZE, GAS TYPE AND DURATION

There are different opinions regarding test sorption duration. Seidle and Huitt (1995) used coal matrix strain equilibrium for gas saturation measurement. The test was carried out using coal matrix shrinkage method with the coal samples (core samples of unknown size) being subjected to pressures up to 13,790 kPa (2,000 psi). They reported that when coal adsorption was carried out in methane, the coal matrix strain took about three months to stabilise. However, when in desorption it took about ten days. The process of desorption from the sample was carried out in steps of 1,379 kPa (200 psi). In helium however, the equilibrium time at each step was about three days and in carbon dioxide it took around four days. As a result Seidle and Huitt concluded that longer equilibrium times would be required for matrix shrinkage test.

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Harpalani and Chen (1995) swelling/shrinkage test found that methane took nearly four month to reach strain equilibrium at increasing pressure, but it took about a month in decreasing pressure. They suggested using smaller samples sizes instead of 89 mm diameter cores to reduce the duration of the experiment. Harpalani (2005) reported adsorption measurements of methane and carbon dioxide using pulverised samples of coal 0.420-0.149 mm (40-100 mesh) size from the Illinois basin. Harpalani reported very long desorption times for most Illinois coal, suggesting low diffusion rates.

Moffat and Weale (1955) using only methane gas, found that both powdered coal and lumps of about 12.5 mm ($\frac{1}{2}$ in.) size requires the same time to reach the equilibrium pressure. At 100 MPa (1000 atm.) gas pressure, the coal sample took less than one hour to reach equilibrium state. Moffat and Weale also observed that the same sample, previously saturated, when the methane was released in a given amount took almost two hours to reach the new equilibrium pressure state. All the coal particles tested in gas adsorption were 0.211 mm (72 B.S. mesh) size.

Recently, Van Bergen et al. (2009) carried out swelling test in unconfined coal samples of 1.0-1.5 mm³ exposed to each of CO₂, CH₄ and Ar. The test was run for between 17 and 24 hours for each coal samples.

In the present study carried out on Bulli coal seam, it was found that in adsorption, it took a minimum of three days for carbon dioxide and four days for methane to reach the equilibrium pressure level, as long as the suggested concept of pressure duration was applied, which is defined as the minimum period of gas sorption in coal in reaching the pressure equilibrium. However, it took more than 30 days to reach the equilibrium state at decreasing pressures (testing induced desorption characteristics), for both in carbon dioxide and methane gas environment.

EXPERIMENTAL

The indirect gravimetric method was used to determine the volume of gas adsorbed in coal. Using different coal particle sizes the volume of gas adsorbed in coal was determined at different gas pressures up to 4000 kPa. Gases used in the test were carbon dioxide and methane. The apparatus used for the test is described by Lama and Bartosiewicz (1982) and later by N.I. Aziz and W. Ming-Li (1999). This apparatus has since been modified adding a pressure transducer to each pressure vessel, commonly known as bomb. A total of 17 bombs were used in the test. Seven of these bombs were run with carbon dioxide and another seven with methane. The remaining bombs were used to evaluate the volumetric strain changes in 54 mm diameter coal core samples.

Coal Samples

The adsorption characterisation study was centred on coal samples collected from West Cliff Colliery and longwall panel 520-B3. Details of the coal samples are shown in Tables 1 and 2. Particle fragment sizes of ± 1.18 mm ($-0.256+16$ mesh), ± 6.70 mm ($-5/16+0.256$ mesh), ± 8.00 mm ($-5/8+5/16$ mesh), and ± 15.00 mm cubical blocks ($-5/8+0.530$ mesh) were tested in addition to 54 mm diameter coal core samples as shown in Figures 1 to 3.

Table 3 shows the details of fourteen bombs with the corresponding particle size and gas type. The samples were dried at 105 °C for 18 hours and were then maintained at 24 °C room temperature and atmospheric pressure for about 18-24 hours to achieve moisture equilibrium prior to sorption testing. Each bomb was filled with coal to approximately 85 % of its capacity. Prior to gas pressurisation, all bombs were vacuumed as much as practically possible to remove any residual air or gas from coal. The vacuuming was carried out for one hour, in three 20 minutes steps, down to about -51 kPa. The bombs were maintained at constant temperature in water bath, and the room temperature was acclimatised to a desirable temperature of 24 °C. The room temperature was monitored continuously through thermocouples, located on each sorption apparatus. A data logger was used to record room temperature, water bath temperature, atmospheric pressure, bombs equilibrium pressure and weight on a regular basis. The bomb weight measurements were made both at the initial and final pressure levels and at each incremental increasing step of 500 kPa. Any gas pressure leakage could easily be detected by differences in bomb weight.



Figure 1- 54 mm diameter core size

Figure 2- ± 15.00 mm cubical blocks

Table 1 - Coal density and proximate analysis

Sample code	Depth (m)	Moisture (%)	Volatile matter (%)	Fixed carbon (%)	Ash Yield (%)	Volatile Matter (% daf)	Coal density
520-B3	450	1.3	21.7	71.4	5.6	23.3	1.43

(Saghafi & Roberts, 2008)

Table 2 – Petrography

Sample code	Vitrinite Reflectance (%)	Maceral (%)			Maceral (% mineral free)			
		Vitrinite	Liptinite	Inertinite	Mineral	Vitrinite	Liptinite	Inertinite
520-B3	1.28	41.6	0.1	55.3	3.0	42.9	0.1	57.0

(Saghafi & Roberts, 2008)

Table 3 - Particle size and gas type

Bomb	Particle size	Mesh size	Gas type
B1G	1.18 mm	-0.256+16	CH ₄
BB	1.18 mm	-0.256+16	CH ₄
B4	6.70 mm	-5/16+0.256	CH ₄
BC	6.70 mm	-5/16+0.256	CH ₄
BL	15.00 mm	-5/8+0.530	CH ₄
BH	15.00 mm	-5/8+0.530	CH ₄
BD	54 mm core	54 mm	CH ₄
B2	1.18 mm	-0.256+16	CO ₂
B5	1.18 mm	-0.256+16	CO ₂
BA	8.00 mm	-5/8+5/16	CO ₂
BF	8.00 mm	-5/8+5/16	CO ₂
B6	15.00 mm	-5/8+0.530	CO ₂
BK	15.00 mm	-5/8+0.530	CO ₂
B3	54 mm core	54 mm	CO ₂

Figure 3- ± 1.18 mm size

Results and analysis

Figures 4a and 4b show the rate of gas pressure drop due to adsorption of either CH₄ or CO₂. The rate of gas pressure drop is a clear indication of how the sample is being saturated. In the initial 360 minutes the pressures drop in CO₂ gas were about 70-80% in fragments of 1.18 mm, 70% in

fragments of 8.00 mm, 50-80% in cubical blocks of 15 mm, and 40% in 54 mm diameter coal core. In summary, in fragments between 1.18 mm and 15.00 mm, the pressure drop was on average 70% while in 54 mm, it was about 40%.

In CH₄, the pressure drop was about 30-40% in fragments of 1.18 mm, 20-30% in fragments of 6.70 mm, 30-40% in cubical blocks of 15.00 mm, and 30% in 54 mm diameter coal core. In summary, in fragments between 1.18 mm and 15.00 mm, the pressure drop was on average 30% and it was 30% in 54 mm coal core. 100% represents the total pressure drop to reach equilibrium pressure for a given gas. Note, the initial confining pressures were different for different particle sizes tested. Post analysis of the data clearly indicated the need to start the test under similar confining pressure.

The concept of sorption duration on vacuumed coal sample was defined according to the data collected over a period of 12 months sorption tests using different gases and in different particle sizes carried out at the University of Wollongong. The concept of duration was studied with respect to the minimum test sorption duration and pressure rate. The minimum sorption duration is defined as the minimum sorption time for coal to reach the equilibrium pressure. This minimum duration was found to be three days for CO₂, and four days for CH₄, irrespective of particle size.

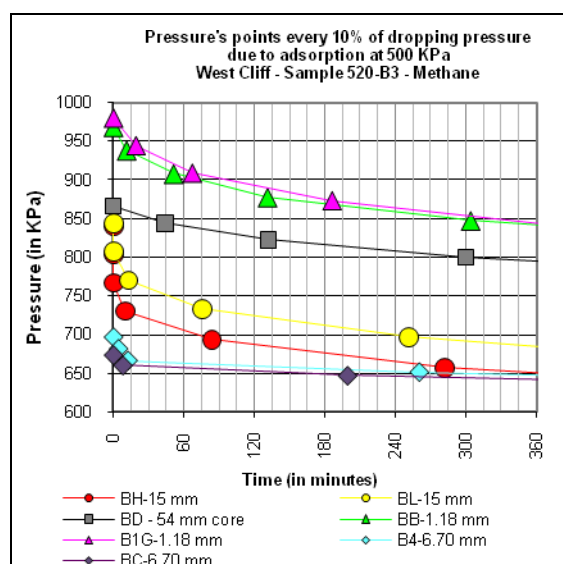


Figure 4a – Dropping of saturation pressure at 500 kPa in adsorption - Methane

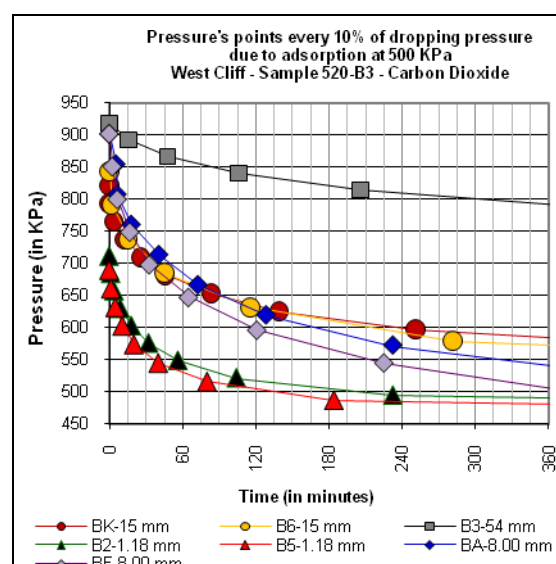


Figure 4b – Dropping of saturation pressure at 500 kPa in adsorption – Carbon dioxide

These findings were in agreement with Moffat and Weale (1955) regarding the particle size and with Seidle and Huitt (1995) with regard to sorption in different gas environments (that is, adsorption in carbon dioxide occurred faster than in methane). Further to the minimum pressurisation time and the rate of pressure change (which is the pressure change over a given period of time against the total pressure change), the study examined the last 18 hrs of monitoring before the equilibrium pressure was reached. The last observed pressure was considered as the equilibrium pressure, once the rate of pressure fluctuation was not greater than 5%. These two requirements (minimum duration and rate of pressure change) guarantee that the change of the gas pressure was stabilised over the time as indicated by the asymptotic profile of the graph to the time axis as depicted in both Figures 5a and 5b.

Figures 5a and 5b show gas adsorption equilibrium paths at 500 kPa pressure for two different gas types. The CO₂ curves show a sudden and steep decrease in pressure (Figure 5a), much faster than the CH₄ curves (Figure 5b), which are characterised by relatively slow and gradual tapering graphs to reach the equilibrium state.

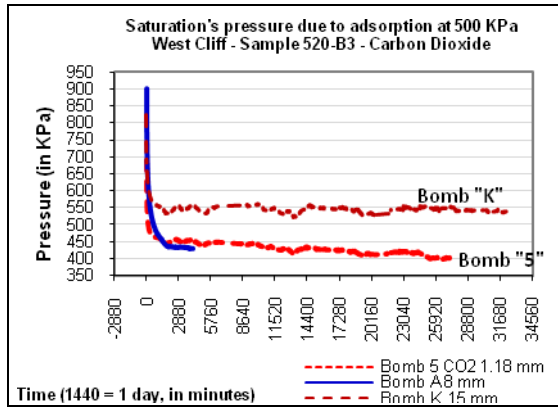


Figure 5a – Saturation pressure due to adsorption – CO₂

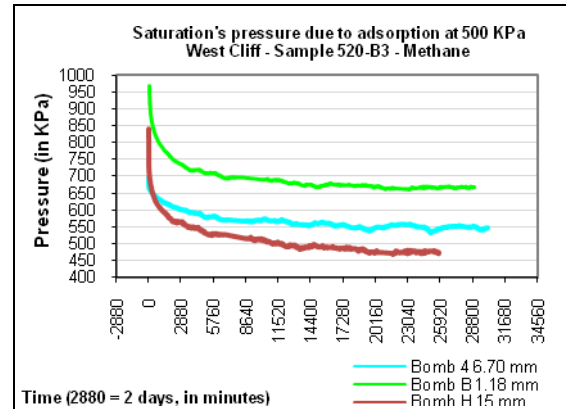


Figure 5b – Saturation pressure due to adsorption – CH₄

An interesting example of the adsorption duration is the graph of bomb K, shown in Figure 5a. The equilibrium pressure (or saturation pressure) once reached did not change even though the sample was maintained under adsorption for longer period (33120 minutes or about 23 days). That is due likely to low diffusion rates and/or large cleat spacing, particularly at low pressure (in this case 500 kPa). This may even change slower in samples with much smaller (fine) pores. It should be noted that, over the same period however, the pressure in Bomb 5 decreased by 50 kPa (about 11% of the final equilibrium pressure). It is clear that all the three bombs containing particles sizes ranging between 1.18 and 15 mm took around three days to reach equilibrium pressure (Figures 5c and 5d).

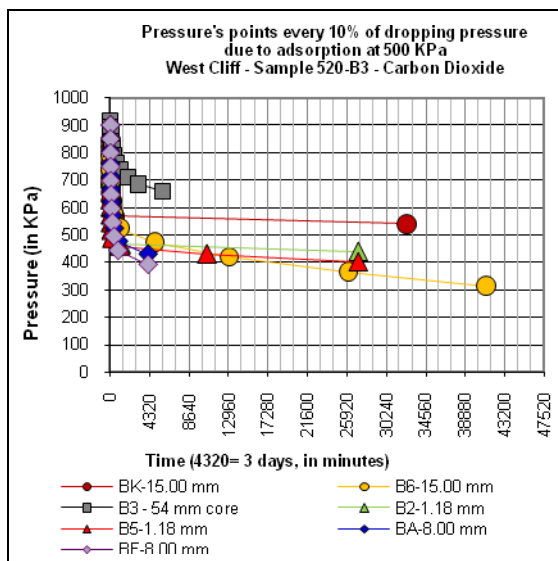


Figure 5c – Saturation pressure due to adsorption – CO₂

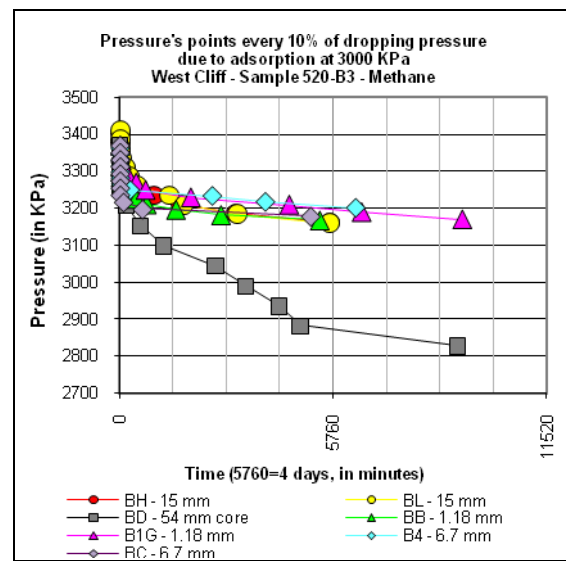


Figure 5d – Saturation pressure due to adsorption – CH₄

Figures 6a and 6b show how long the gas takes to reach the equilibrium pressure at low confining pressures and with repeated charges spanning over a period of around two days. The behaviour is similar for both gases, methane and carbon dioxide, and depend more on the physical characteristic of the coal and the surface attraction and repulsion forces. The same phenomenon explains why at low gas pressures the coal adsorbs much more gas than at high pressure.

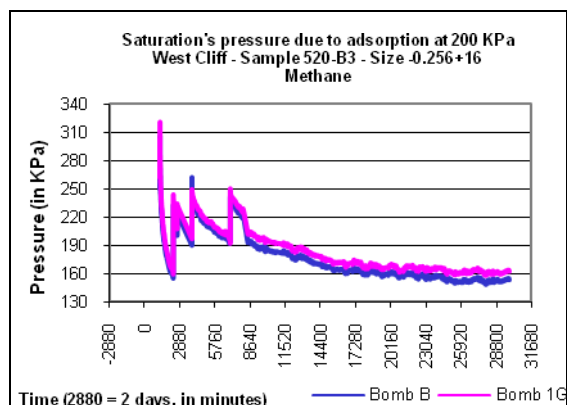


Figure 6a – Saturation pressure due to adsorption – CO₂

Figures 7a and 7b show the equilibrium pressure due to induced desorption (in reverse process) in three different coal sizes in both methane and carbon dioxide (e.g. at 3,500 kPa). The changes in pressure levels along the time line in all the samples behave similarly with respect to pressure fluctuation and

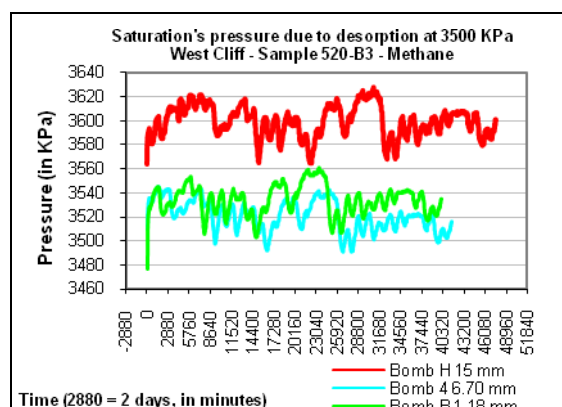


Figure 7a – Saturation pressure due to desorption – CH₄

This gradual drop in bomb pressure as indicated in Figure 7b is a clear indication that gas is continually being re-adsorbed in coal, and that the process of desorption occurs for very short times and intermittently. In CH₄, the desorption process seems to be predominant with the pressure being gradually increased. In other words, the sorption behaviour and interpretation of the graphs in Figures 7a and 7b is that coal is more prone to adsorb carbon dioxide than methane. These findings are in agreement with the findings of other researchers such as Deitz, Carpenter, and Arnold (1964), Sereshki (2005), and others. St. George and Barakat (2001) stated that the adsorption of carbon dioxide in coal is easier than methane. This differential behaviour is commonly known as the “affinity” of carbon dioxide to the coal. The preferential sorption of CO₂ in coal as compared to CH₄ and coal affinity to carbon dioxide is explained in terms of gas molecular weight and gas thermodynamics. The higher the molecular weight of the gas the lower the rate of vaporization, which means the lower rate of desorption. Thus, the evaporation rate of CO₂ will be lower than of CH₄. The same conclusion can be made by considering the amount of the heat of vaporization. The lower the heat of vaporization the easier the desorption process. Also Harpalani (2005) reported the same affinity in three different coal seams from Illinois basin.

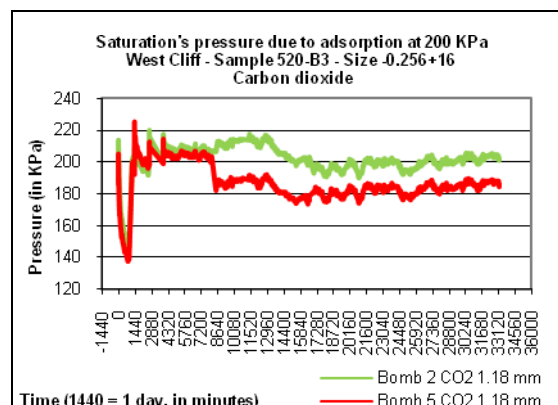


Figure 6b – Saturation pressure due to adsorption – CH₄

sorption process. Methane desorbs easily but re-adsorbs slowly, which is different to carbon dioxide sorption behaviour. In the case of CO₂, the adsorption is the dominant process, rather than desorption as one expects.

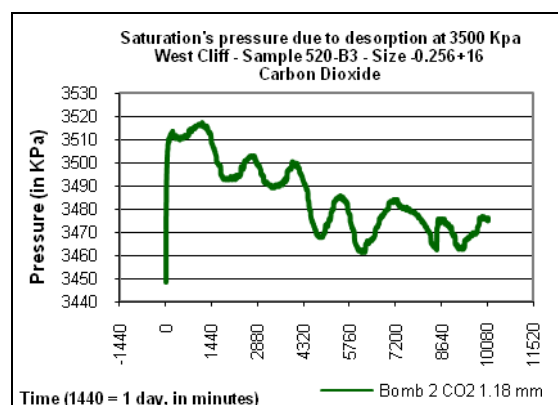


Figure 7b – Saturation pressure due to desorption – CO

The influence of the adsorption duration on isotherms calculation can be seen in both Figures 8a and 8b. The graph in Figure 8a shows three isotherms for saturated and less than saturated samples. They are: (a) Bomb 1G isotherm profile of full saturation and in accordance with the minimum sorption duration time, (b) an under saturated isotherm profile for 48 hr (2 days) test duration (obtained from data of Bomb 1G), and (c) a simulated profile, termed Sim T profile, calculated based on the changes in pressures of about 50 kPa less than full saturation isotherm profile (a) and for comparison with (b).

The graphs in Figure 8b include two isotherm profiles consisting of (a) full saturation test result isotherm profile of Bomb 1G, containing 250 grams of coal crushed to about -0.256+16 mesh and measured at 24 °C, from Westcliff panel 520-B3 (the result is produced for coal 'as received' and no correction for moisture or ash content was made), and (b) the isotherm (A) measured by Saghaei and Roberts (2008) from 300 gms of West Cliff Colliery coal (panel 520-B3), crushed to 150 µm particle size. The test was carried out at 27 °C and the results were produced for coal 'as received', and no corrections were made for moisture or ash content. Saghaei (2009) reported that, every points of the isotherm (A) reached equilibrium pressure after 4-6 hours, though generally after 2 hours the equilibrium was virtually reached.

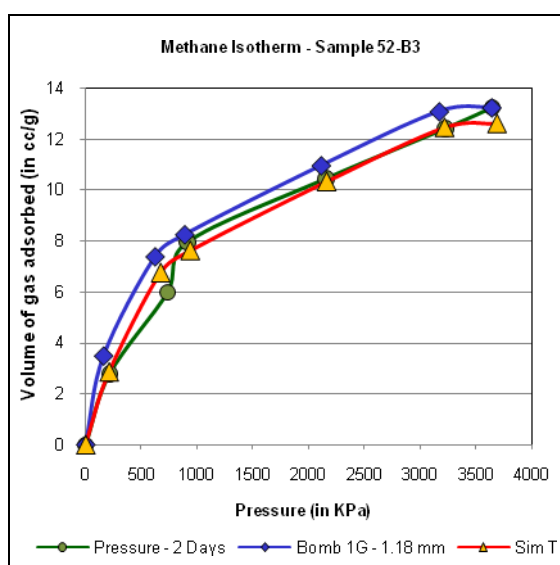


Figure 8a – Saturation at 48 hours

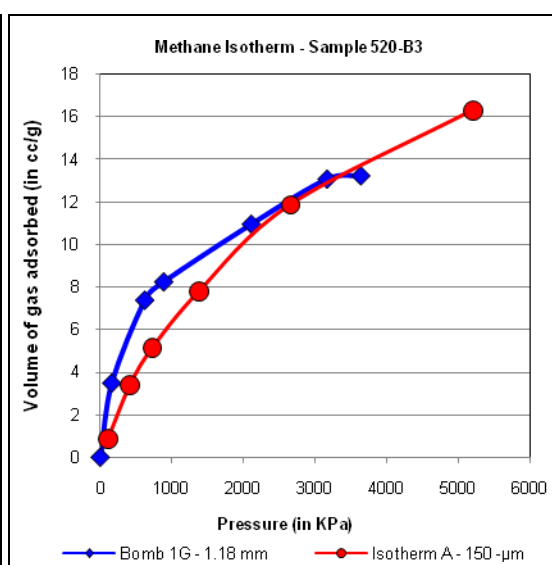


Figure 8b – Saturation at minimum duration

It was found that (Figure 8a), in 1.18 mm size coal, and at under saturation adsorption condition (sorption duration below that the minimum suggested sorption time), there was 18% less gas adsorbed in coal at exposed pressures below 500 kPa. However, at pressures up to 4000 kPa the change was 5%. The adsorption duration for 48 hours was also reported by Day, Fry and Sakurovs (2008).

In comparing the isotherms shown in Figure 8b and named as 1G and A, it can be seen that at low pressures, the gas content of the vessel (1G) is higher than the A graph, but at high pressure (at about 4000 kPa), it was quite similar. Clearly under saturation test has its draw backs which, if not addressed properly may lead to errors in the determination of the gas content in coal, thus influencing the production potential of a mine.

CONCLUSIONS

The following conclusions were drawn from this study:

- Coal charged with CO₂ to a confining pressures of up to 4000 kPa;
 - It takes at least three days to reach the equilibrium pressure in coal fragments and four days in 54 mm coal core samples;
 - In coal fragments, around 70% of the equilibrium pressure was reached during the first 360 minutes, while in coal core sample was reached 30% of the equilibrium pressure in the same period.

- The equilibrium pressure behaviour due to the CH₄ was similar to CO₂ but takes longer;
 - It takes at least four days to reach the equilibrium pressure in coal fragments and six days in 54 mm coal core samples.
 - It takes 360 minutes to achieve in average 30% of the equilibrium pressure in coal fragments. In coal core it reached 30% of the equilibrium pressure in the same period of time.
- In general, the higher the initial charged confining pressure at any step the faster is the adsorption for the same particle sizes. However, at lower initial charged confining pressure, the adsorption process takes longer, and the rate of adsorption appears constant, which may be due to the low process of diffusion to the coal matrix.
- Contrary to general belief, the adsorption duration for a given gas appears not strongly affected by the coal particle size (especially among fragments of coal). Further tests are underway to confirm this finding on different coal samples from different coal seams.
- Irrespective of sorption pressurisation, the level of pressure does not play a role in improving the minimum sorption duration.
- In general, the physical characteristic of coal plays an important role at any pressure, as it drives the sorption process and its duration (for a given gas pressure and temperature).
- The preferential sorption of CO₂ in coal as compared to CH₄ and coal affinity to carbon dioxide is explained in terms of gas molecular weight and gas thermodynamics.

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REDUCING COAL MINE GHG EMISSIONS THROUGH EFFECTIVE GAS DRAINAGE AND UTILISATION

Dennis Black¹ and Naj Aziz¹

ABSTRACT: Gas emission from Australian coal mining is estimated to account for 4-5% of the nation's 559 million tonnes of CO₂ equivalent (MtCO_{2-e}) Greenhouse Gas (GHG) emissions. With the intense focus on global GHG management and reduction, to slow the rate of climate change, significant community and political pressure exists to reduce the rate of gas emission. In December 2007 Australia committed to join the Kyoto Protocol, which in part requires annual GHG emissions not to exceed 108% of 1990 levels by the end of the 2012 commitment period. The current Australian Federal Government is presently developing the Australian carbon pollution reduction scheme, which is due to be implemented by 2010. This scheme is expected to place a value on GHG emissions and thereby introduce a financial penalty/incentive on organisations to manage and reduce their GHG footprint. In the case of the Australian coal industry, with an estimated annual GHG contribution of 22.5 Mt CO_{2-e}, the introduction of the emissions reduction scheme will add in the order of half a billion dollars to the cost of operations (based on a carbon unit cost of \$20/t CO_{2-e}). In light of such a significant additional cost it can be expected that gas capture and emissions reduction will receive an unprecedented increase in attention and corporate support. This paper discusses the various sources of gas emission from underground coal mines and describes methods to improve both the capture and utilisation of this gas to reduce GHG emissions.

INTRODUCTION

Whether coal seam gas is considered a nuisance or threat, in the case of coal mine operators, or an opportunity, in the case of coalbed methane gas producers, it is essential that operators and planners understand the principles of gas generation, storage and its ability to be drained from the seam.

Gas is generated during the coalification process and the amount of gas present within a particular coal seam, known as gas content, is dependent upon a range of factors, which include; seam thickness, depth of burial, bounding strata type, coal geology, coal structure, coal strength, igneous activity and/or igneous sources, secondary biogenic activity and the ground stress regime.

The flow of gas in coal seams involves migration, through fractures and cleat, and diffusion through the coal matrix. Gas molecules diffuse through the coal matrix in response to concentration gradients and upon reaching the cleat system migrate in response to pressure gradients, obeying Darcy's Law. However as greater than 90% of the gas in coal is stored in micropores, diffusivity is the rate limiting factor for gas flow in most low permeability coals. Given the large number of factors that impact gas generation, storage and movement it should be no surprise that there is such a high degree of variability in gas content and composition as well as the ability to drain gas from coal seams throughout Australia.

Where the seam gas content is considered high, greater than 6-8 m³/t, gas drainage is employed to reduce the naturally occurring gas content within a coal seam to a level where the risk of initiating an outburst is significantly reduced and the volume of gas liberated from the coal during mining is able to be diluted by the mine ventilation air to a level which complies with mine safety regulations. Among the mines that employ gas drainage the complexity and effectiveness of the drainage systems varies significantly, ranging from boreholes that discharge into the mine return airways which in turn discharge to atmosphere via the mines ventilation fans, through to mines whose drainage boreholes are connected to surface drainage plant via complex reticulation network with subsequent downstream utilisation of the drainage gas.

In those mines considered to be less gassy and therefore not requiring gas drainage for operational issues the gas emitted from the coal during operations is cleared from the working areas by the mine

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ventilation system where it is removed from the mine and discharged to atmosphere via the mines ventilation fans.

There are many sources of gas emission throughout an operating underground coal mine, shown on Figure 1, which include:

- Rib emission into both intake and return airways
- Emission from coal cutting – both development and longwall production
- Emission into longwall goaf from adjacent gas bearing coal seams and strata
- Emission from longwall goaf into connecting airways
- Emission from coal being removed from the mine via the coal clearance system

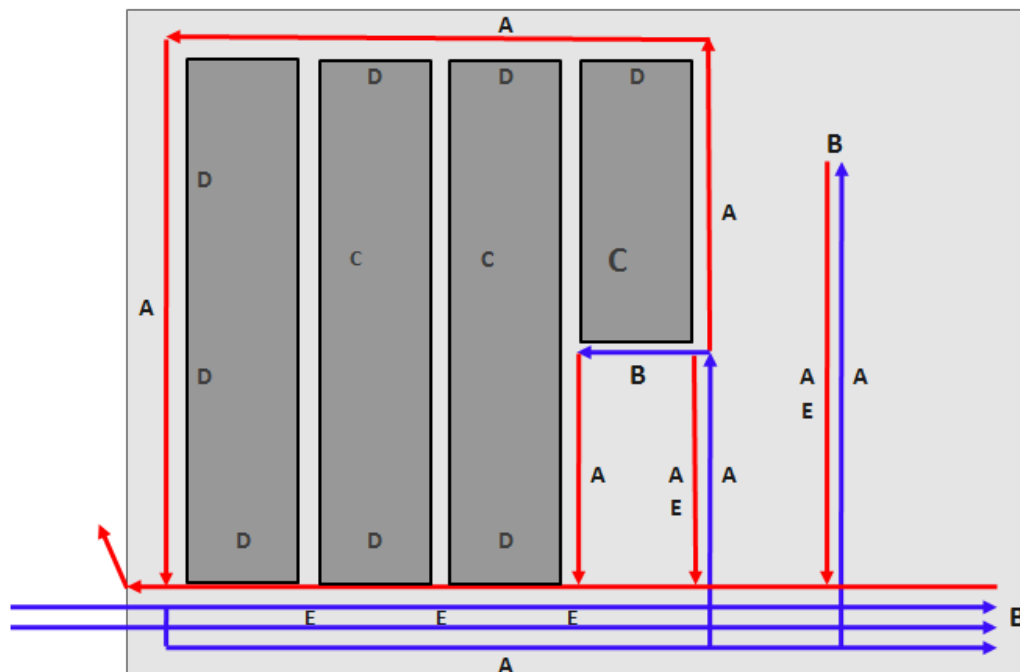


Figure 1 - Conceptual mine layout illustrating the location of potential gas emission sources

Given the potential for high seam gas content and coal production capacity, coal mining is widely considered by community and government to be a major emitter of greenhouse gases. The scale of emissions will however vary between mines and is primarily controlled by the gas content (m^3/t) of the coal seam or specific gas emission (m^3/t) from all gas sources impacted by mining and the rate at which coal is produced (tonnes). Table 1 illustrates the scale of annual greenhouse gas emission, in tonnes of CO_2 equivalent, for a range of gas content and coal production capacities.

Should there be a value placed on carbon emissions and corresponding financial penalty imposed on mining companies based on net emissions it can be expected that strong corporate support will be provided to implement emission reduction measures. Should the cost of GHG emission be \$20.00/t CO_{2-e} , the impact on a mine with an SGE of $15 \text{ m}^3/\text{t}$ producing 4.0 Mtpa, would be an additional \$17.0 million per annum (\$4.25/ROMtonne) in emissions penalties. For higher producing mines and/or those with greater specific gas emissions the cost of the penalties will be greater and will be further impacted should the unit cost of carbon emission increase.

In order to reduce the net overall cost of minesite emissions it is expected that many operations will implement measures to capture and utilise coal seam gas thereby reducing emissions.

GAS DRAINAGE – PRE-DRAINAGE

The use of in-seam drilling ahead of mining for gas drainage was first introduced in Australia in 1980 to reduce the coal seam gas concentrations to levels sufficient to be managed by the mine ventilation

system during both the roadway development and longwall coal extraction processes. Since 1980 Underground to Inseam (UIS) drilling has evolved from simple rotary drilling rigs with limited directional control and depth capability to the current technically advanced units incorporating down-hole motors capable of achieving depths in excess of 1,600 metres. The use of UIS drilling has expanded throughout the Australian coal mining industry to become the method of choice for underground gas drainage drilling, particularly in mining regions such as the Illawarra which operate at depths in the order of 450-500m and have substantial surface access constraints which restrict access for surface based methods.

Table 1 - Annual GHG emission based on gas content and annual coal production rate

		Annual Coal Mine Gas Emission (tCO _{2-e})							
		35	497,595	995,190	1,492,785	1,990,380	2,487,975	2,985,570	3,483,165
Specific Gas Emission (m ³ /t)	Gas Content	30	426,510	853,020	1,279,530	1,706,040	2,132,550	2,559,060	2,985,570
	25	355,425	710,850	1,066,275	1,421,700	1,777,125	2,132,550	2,487,975	
	20	284,340	568,680	853,020	1,137,360	1,421,700	1,706,040	1,990,380	
	15	213,255	426,510	639,765	853,020	1,066,275	1,279,530	1,492,785	
	10	142,170	284,340	426,510	568,680	710,850	853,020	995,190	
	5	71,085	142,170	213,255	284,340	355,425	426,510	497,595	
		1,000,000	2,000,000	3,000,000	4,000,000	5,000,000	6,000,000	7,000,000	
		Annual Coal Production (tonnes)							
Note: The global warming potential (GWP) of methane is 21 times greater than carbon dioxide.									

In gassy mines, such as those operating in the Bulli seam, it is common for substantial UIS drilling to be completed ahead of mine development, with in excess of 100,000 metres being drilled annually. The cost of such an intensive drilling program, along with the associated infrastructure, is in the order of \$4-6 million per annum. A variety of drilling patterns and treatments are available, illustrated in Figure 2, with the most common pattern presently in use being the Fan pattern.

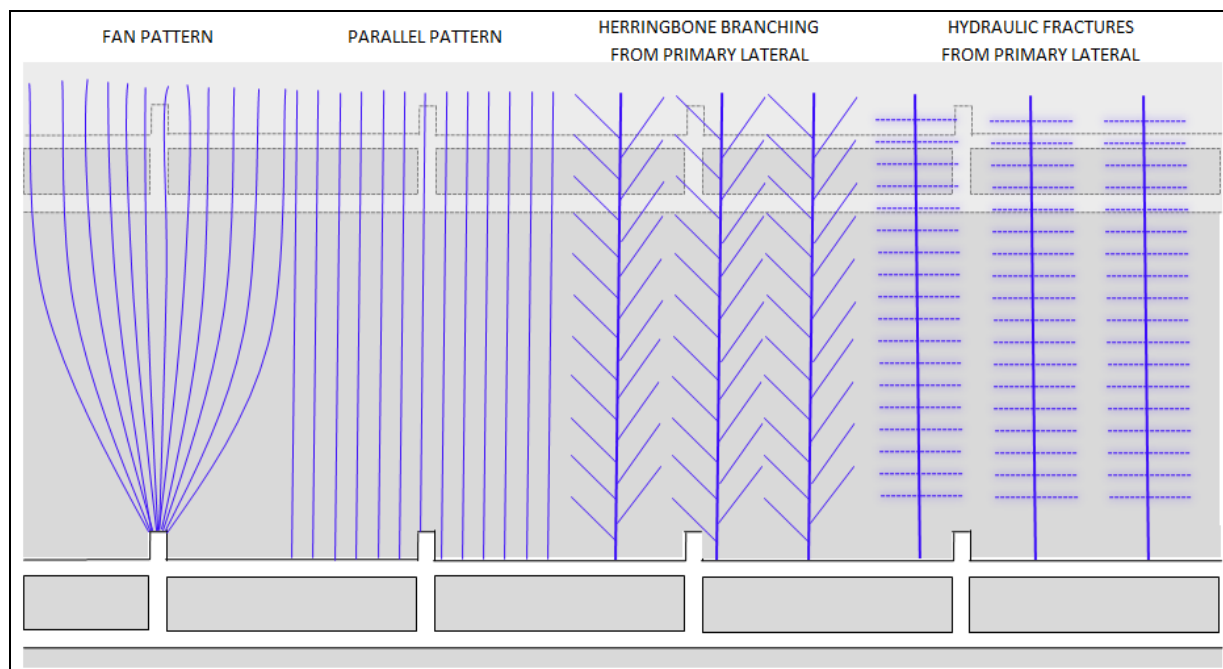


Figure 2 - UIS drilling patterns and gas drainage enhancement options

Recent studies have been undertaken to evaluate the effectiveness of the intensive UIS gas drainage programs (Black and Aziz, 2008) and it was found that some 50% of the drilling effort delivered little to no benefit to gas content reduction. In such cases where the gas drainage system was not achieving

optimum performance it is not uncommon for the mine to address the problem by drilling many more holes in the area, which essentially amounts to throwing good money after bad.

The reasons identified for the failure and poor performance of such a significant percentage of the boreholes in the drainage program include:

- 1) Insufficient drainage time prior to intersection by development gateroads;
- 2) Insufficient monitoring and management of borehole performance resulting in low to no flow due to accumulation of water and/or coal fines within the borehole;
- 3) Insufficient monitoring and management of the gas reticulation pipe network due to blockages or significantly restricted flow capacity due to the accumulation of water and/or fines in sections of the range;
- 4) Poor standard of sealing holes following intersection by development resulting in air in the pipe range and reduced suction pressure;
- 5) Insufficient standpipe length and sealing (grouting) standard resulting in air dilution in the pipe range and reduced suction pressure;
- 6) Boreholes drilled down-dip and not in the optimum orientation for maximum drainage performance; and
- 7) Absence of in-hole dewatering where boreholes have been drilled down-dip resulting in in-hole water accumulation restricting gas desorption.

A further inherent problem with the UIS method of gas drainage is the reliance on mine development to be completed in order to provide access to areas where drilling can be undertaken. Given the objective of most mining operations to achieve rapid development to form longwall blocks that can be extracted quickly to achieve high annual production, the amount of time available for drilling and draining the next gateroad in the development sequence is reducing. In areas with higher gas content and lower permeability there have been many examples where the seam gas content has not been reduced sufficiently resulting in production delays. During development production delays the longwall typically continues to operate which erodes development lead placing even greater pressure on development and further reduces the available drainage lead time. In the extreme cases operations have chosen to cut longwall panels short and therefore sacrifice valuable reserves rather than incur potentially significant production delays while waiting for sufficient gas to be drained.

It is therefore extremely important that mine operators clearly understand both the drainage characteristics of the future mining areas, particularly those areas expected to be slow draining, and the expected drainage time available, based on the mine production and drilling schedule. Where areas are identified that drainage time is expected to be insufficient it will be necessary to employ additional drainage methods and possibly stimulation treatments to avoid production delays or loss of reserves.

A method that offers a significant increase in drainage time is Surface to Inseam (SIS) drilling. Originally vertical wells were drilled from the surface to intersect the various gas bearing seams however these wells achieve very low surface contact with the respective seams and, in the absence of high permeability and favourable drainage characteristics, the resulting gas drainage flow rates were quite low. Methods were developed to stimulate the gas production performance of these wells, which included under-reaming, cavity completion, and hydraulic fracturing.

Further drilling technology development led to the introduction of deviated well drilling, also known as radius drilling. This method involves initially commencing the drilling with a vertical, or near vertical, section and then bending the drill string through an acceptable radius, which is governed by the capability of the drill string, to intersect the coal seam, or target drilling horizon, horizontally and then continuing to drill and extend the borehole at the desired horizon to the planned borehole length. A range of radius drilling designs are presented by Logan et. al. (1987) and illustrated in Figure 3. The total length of the inseam section of such boreholes is capable of exceeding 2,000 metres, however the length is principally dictated by the capacity of the drill rig and the drilling fluids used.

Following the introduction and development of the SIS drilling technology in Australia the use of Medium-Radius Drilling (MRD), employing a typical bend radius of 250-350m, has seen widespread application, particularly in the Queensland Coalbed Methane (CBM) industry. MRD is now becoming a favoured method in many Queensland coal mine pre-drainage programs with increasing application in the Hunter Valley and consideration is being given to trials in the Illawarra.

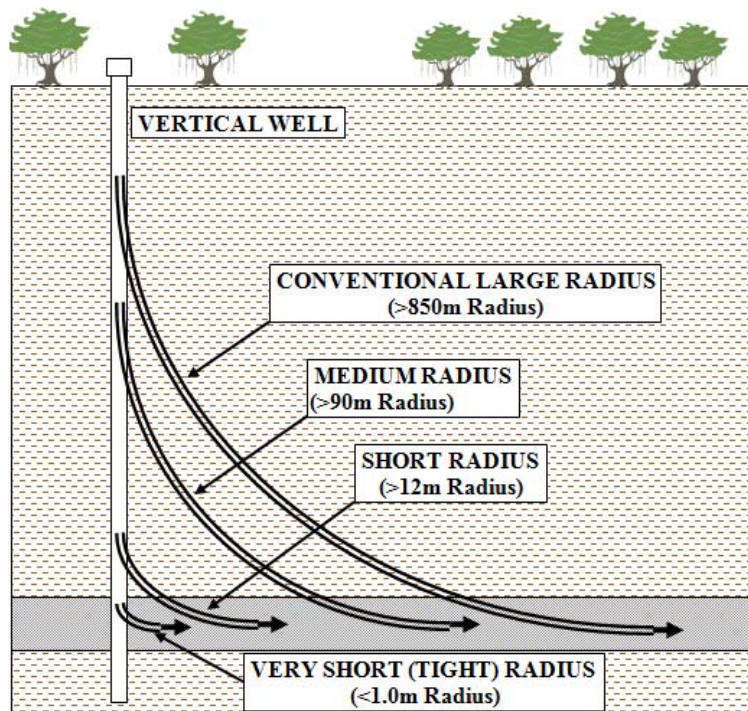


Figure 3 - Surface to in-seam horizontal drainage drilling technologies, after Logan et. al. (1987)

GAS DRAINAGE – POST-MINING (GOAF) DRAINAGE

The gas released during and subsequent to the longwall mining process represents the major source of coal mine gas emission, particularly in situations where additional gas bearing coal seams and strata, located in close proximity to the seam being extracted releases its stored gases. In the case of mines operating in the Bulli seam the combined impact of gas liberated from all effected sources during longwall extraction is in the order of 35-45 m³/tonne. In cases where high gas emission occur the use of effective gas drainage techniques is essential to minimise gas related production delays and maintain the safety of the mine and its workforce. There have been many methods used by mines to drain gas from both the active and sealed goaf, these underground based methods include:

- a) Cross-measure boreholes – boreholes drilled above and/or below the working seam located along the length of the longwall panel;
- b) Back-of-block drainage – boreholes drilled above the working section to connect into the goaf to remove accumulated high purity gas;
- c) Goaf seal drainage – removal of gas from sealed goaf via pipes passing through seals; and
- d) Horizontal directional drilling – long boreholes drilled above and/or below the working seam and oriented parallel to the longwall panel which connect to the forming goaf to drain the accumulating gas.

Although the underground gas drainage methods are capable of removing very high volumes of gas (>>2,500 lps), there are many examples where the rate of gas emission has exceeded the capacity of the drainage system resulting in gas-related production delays. For mines in such situations the use of additional surface based goaf drainage techniques may be appropriate. One such technique is the use of vertical boreholes, located toward the tailgate side of the longwall panel and drilled ahead of the retreating longwall face. The bottom of the hole is typically located a distance of 10-35 metres above the roof of the working section. Following the passing of the longwall face and goaf formation, suction is applied to the goaf drainage borehole and the gas accumulating in the goaf is drawn to the surface, typically through the use of vacuum plants.

Figure 4 illustrates the method of vertical well goaf drainage typically employed.

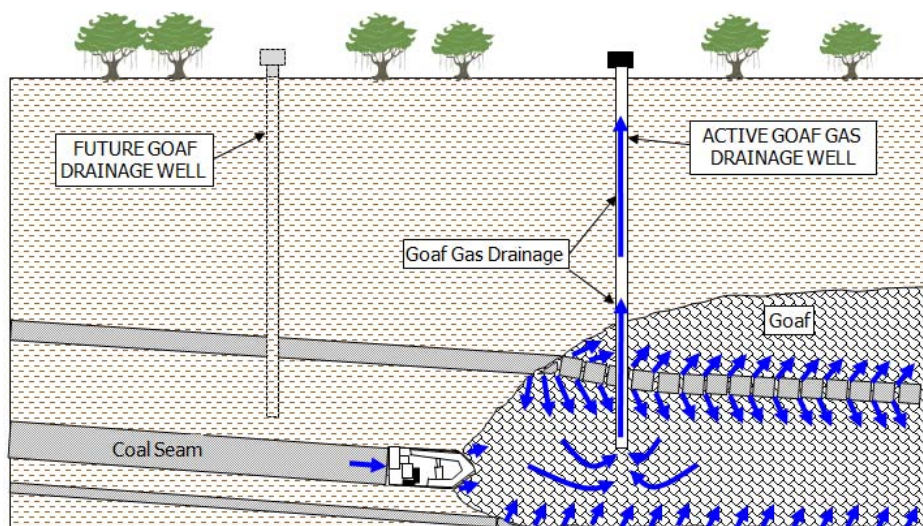


Figure 4 - Vertical well surface-based goaf gas drainage

With the ever increasing pressure being applied to mine operations through urban development and environmental sensitivity the use of vertical goaf drainage wells, typically spaced no greater than 300-400 metres apart, represents a high impact, particularly give the needs for ancillary plant such as drainage plant, emissions reduction plant (e.g. flare units) and/or gas reticulation pipelines to service the wells. In situations where significant surface access restrictions exist, mines may be prevented from employing vertical well surface goaf drainage which may result in restricted production through inability to manage total gas emissions. In such cases alternative gas drainage methods must be developed and utilised. One such alternative method, proposed by the first author, is the use of radius drilling to form boreholes parallel to the longwall block, positioned on the tailgate side of the longwall face, approximately 30-50 metres above the roof of the working section, drilled ahead of the retreating longwall face. As the longwall face passes the end of the borehole and connection to the goaf occurs suction is applied to the goaf drainage borehole to remove the accumulating gas. Due to the nature of goaf formation relative to the longwall face the position of the open end of the horizontal drainage borehole can be expected to remain relatively constant throughout the operating life of the well, resulting in a stable and overall greater gas production capacity than that which is achievable through the use of vertical goaf drainage wells. Figure 5 provides an illustration of one particular horizontal goaf drainage well design.

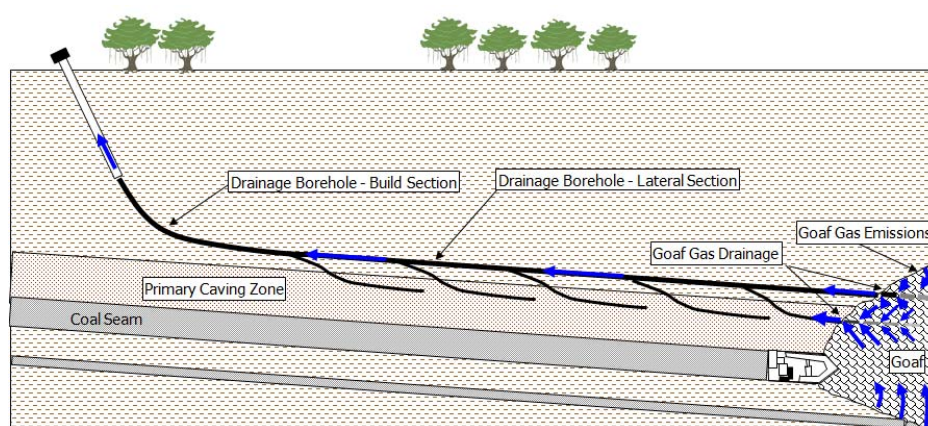


Figure 5 - Horizontal well surface-based goaf gas drainage

A further advantage of the use of radius drilling for the formation of horizontal goaf drainage wells is the ability to drill multiple laterals to form multiple connections to the goaf which improves both redundancy and overall gas production capability. A production and financial comparison between the use of single and twin lateral horizontal well has been provided in Figure 6.

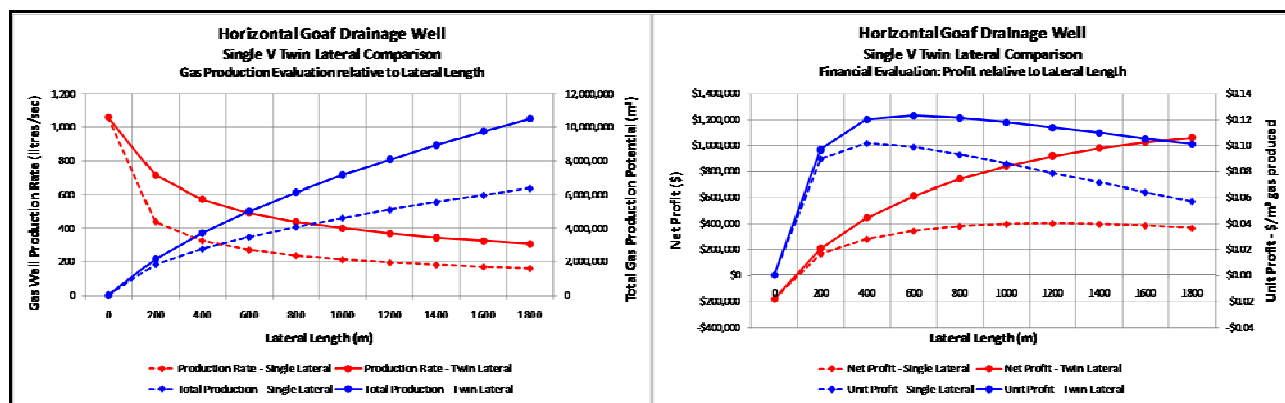


Figure 6 - Comparison between single and twin lateral horizontal goaf drainage well production and economic performance

GAS UTILISATION

Prior to the introduction of government schemes and incentives for the utilisation of coal mine methane only three Australia mines actively utilised gas for power generation, being Appin, West Cliff and Tower collieries. The majority of gas emission from other mines was vented to atmosphere with few exceptions that employed flaring. Following the introduction of schemes such as the NSW Greenhouse Gas Abatement Scheme (GGAS) and the federal government's Greenhouse Friendly program, a number of utilisation projects have commenced.

Flaring is the simplest form of emissions reduction and simply involves the burning of methane gas to produce carbon dioxide and water. Where flares are to be located close to developed areas it may be necessary to minimise the visual impact of the project. In such cases enclosed flare units have been developed to limit the height of the flame so as not to be seen by the local community.

The utilisation of coal mine methane in the generation of power has the potential to increase the financial benefits from abating a given volume of gas. In the case of power generation the financial benefits are derived from the sale of carbon credits, and electricity.

Turbines were first used to generate electricity from coal mine methane. The two Australian gas turbine installations, both rated at 15MW, were located at Appin and West Cliff collieries and operated between the years 1986 to 1995 and 1984 to 1999 respectively. The increasing maintenance costs and inefficiencies associated with variable drainage gas concentration led to the decommissioning of these units. These units were replaced by internal combustion engine technology that utilised methane gas as the primary fuel. The most common internal combustion engine utilising methane gas for minesite power generation are the 1.0MW units (e.g. Caterpillar 3516 and GE Jenbacher 320) although both larger and smaller units are available. There are now eight coal mine methane gas power generation projects operating at Australian coal mines, and these include:

- Appin (54MW)
- Tower (40MW)
- Moranbah North (40MW)
- Grasstree (32MW)
- Oaky Creek (12-20MW)
- Glennies Creek (10MW)
- Tahmoor (7MW)
- Teralba (6-8MW)

The largest source of coal mine methane (CMM) is the dilute methane emitted from mine ventilation shafts. Known as Ventilation Air Methane (VAM), is difficult to capture and use because it has a low methane concentration. VAM emissions are typically characterised by large airflows and low concentrations, ranging from 0.1-1.5%, but more typically 0.3 to 0.5%. Further adding to the complexity of mitigating VAM is the large airflow volumes associated with mine ventilation systems, typically ranging from 150 to 350 m³/s. It has been estimated that greater than 55% of all CMM emissions originate from mine ventilation shafts, thus VAM offers both the greatest emission reduction and energy production potential.

Technical applications for VAM use include direct use as a principal energy source in oxidation units, lean-burn turbines, and kilns, where it is mixed with coal fines or other combustible materials. In addition to direct greenhouse gas abatement it is also possible to recover and transfer the heat produced from this oxidation to generate electricity. Table 2 provides a summary of a variety of known VAM utilisation technologies that exist or are being developed.

Table 2 - Summary of VAM utilisation technology development

Vendor / System	Description	Country	Development Status
MEGTEC/ Vocsidizer	Thermal flow-reversal reactor (oxidiser). Heat energy used to superheat steam to power a steam turbine.	United Kingdom Australia USA	8,000m ³ /hr unit installed at British Coal (1994). 6,000m ³ /hr unit installed at Appin Colliery (2002). 250,000m ³ /hr unit installed at West Cliff Colliery (2007) powering a conventional 6MW steam turbine. 50,000m ³ /hr unit installed at CONSOL's Windsor Mine (2007) .
BIOHERMICA/ Vamox	Thermal flow-reversal reactor (oxidiser)	USA Canada	50,000m ³ /hr unit being installed at Jm Walter Resources No.4 Mine (Blue Creek Coal) (2009). 8,500m ³ /hr unit being installed at Quinsam Mine, British Columbia (2009).
CANMET/ CH4MIN	Catalytic flow-reversal reactor (oxidiser)	Canada	500mm pilot plant constructed to demonstrate technology. Seeking to commercialise the technology and undertake minesite demonstration project.
BEESTECH/ HOGT	Waste coal and VAM co-fired in rotary kiln. Compressed air heated in heat exchanger powers a gas turbine.	Australia	CSIRO designed 1.0MW prototype demonstration unit successfully trialled. Seeking minesite demonstration opportunities.
CSIRO/ VAMCAT	Lean-fuelled gas turbine with catalytic combustor (1.0% VAM)	Australia	Demonstration unit (25kW) installed at Panyi Mine, Huainan, China.
FlexEnergy / Lean-fuelled catalytic microturbine	Lean-fuelled Capstone microturbine (1.3%VAM)	USA	Multiple 30kW units operating at abandoned Akabira Mine, Japan
Ingersoll-Rand / Lean-fuelled recuperated microturbine	Lean-fuelled IRPower Works microturbine (1.0% VAM)	USA	1x70kW unit installed at CONSOL's Bailey Mine utilising mine drainage gas (2007). 2x250kW units installed on wellhead at PetroChina's Changqing oil field (2008).
EDL / Carbureted gas turbine (CGT)	Lean-fuelled Solar gas turbine with patented combustor (1.6% VAM)	Australia	2.7MW SOLAR Centaur gas turbine tested at EDL's Appin power station.
EDL / Ancillary VAM use	VAM used to supplement combustion air in Caterpillar 1.0MW engines	Australia	VAM successfully used to supplement combustion air intake to CAT 1MW gas engines at Appin power station.

CONCLUDING REMARKS

With the imminent introduction of the Australian government's Carbon Reduction Scheme there will be potentially significant financial incentive for coal mines to implement effective gas drainage and utilisation strategies to reduce the volume of methane gas emitted to the atmosphere. A number of methods available to drain and capture coal mine methane have been presented along with a range of commonly encountered problem that exist within coal mine gas drainage systems that prevent optimum drainage system performance and effectiveness from being achieved. A variety of methods for utilising the drained gas are also presented.

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APPLICATION OF ENHANCED GAS RECOVERY TO COAL MINE GAS DRAINAGE SYSTEMS

Russell Packham¹, Yildiray Cinar², Roy Moreby.³

ABSTRACT: Over the past 30 years rapid development of the coalbed methane industry in the USA and Australia has stimulated research into the mechanisms that control gas migration in coal seams. A technique for enhancing gas recovery from coal was trialed in the San Juan Basin, USA in 1998. The results showed a sustained 500% increase in gas production rates. The technique involves using an injectant gas to stimulate coalbed gas diffusion and increase seam permeability. This paper describes the technique, the potential applications for coal mining and presents a conceptual field trial for an Australian coal mine to demonstrate the effectiveness in partially drained coal mine workings.

INTRODUCTION

In 1990 Puri and Yee published a paper describing a coal bed methane reservoir as analogous to an absorbent bed. Adsorbent bed regeneration techniques are described as follows (Puri and Yee 1990):

- Pressure depletion – equating to “drawdown” of a coalbed methane gas well, or drilling of an underground gas drainage hole at atmospheric pressure.
- Thermal desorption – reducing the capacity of coal to adsorb gas by increasing the temperature of the coal (not practical for an underground coal mine).
- Displacement desorption – stimulating desorption by displacement with a more strongly adsorbing gas (CO₂)
- Inert gas stripping – stimulate desorption by flushing the absorbent bed (coal seam) by a non-adsorbing or weakly adsorbing gas nitrogen (N₂) to increase concentration gradient.

Most subsequent investigations of enhanced gas recovery have revolved around the theory of the mechanisms and economics in relation to coalbed methane gas. This paper explores the possibility of enhanced gas recovery utilizing an inert gas stripping technique in relation to coal mine gas drainage.

BACKGROUND

Gas drainage objectives in coal mines

Pre-drainage of gas in an underground coal mine is generally conducted for one or more of the following reasons:

- Management of an outburst hazard
- Management of development rib emission
- Management of frictional ignitions (both longwall and development)
- Maintenance of ventilation contaminants to acceptable levels (CH₄, CO₂, H₂S)

In general it is the residual gas content of the coal, after gas drainage operations have been conducted, which is important for coal mine operators. The residual value may be a pre-determined gas content for an outburst threshold or may be a level at which the problems described above are considered manageable (typically 2 m³/t) (Packham 2005).

Achieving specific residual gas content is not a primary objective for coal bed methane (CBM) operators. For a CBM operator, gas production cost is the primary driver, consequently any processes that incur costs, such as the use of an injectant gas to enhance gas production are cautiously examined (Stevenson, Pinczewski and Downey 1993; Reeves, Davis and Oudinot 2004).

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In the Hunter Valley, Australia, examples exist where developed longwall reserves have been sterilised as a consequence of gas drainage lead times (Robertson 2008). Likewise in the Illawarra southern coalfield, potential coal reserves may have to be sterilised due to gas drainage limitations (Black 2007). Clearly the economic considerations for enhanced gas recovery differ between the coalbed methane industry and the coal mining industry.

GAS DRAINAGE MECHANISM

In order to explain the process of enhanced gas recovery it is desirable to understand the mechanism of gas production from a coal seam. Two processes control the rate of gas recovery from a coal seam under pressure depletion as a means of drainage, Darcy's law in relation to gas and water flow through the cleat system and Fick's Law in relation to diffusion of the adsorbed gas from the coal matrix into the cleat.

Darcian Flow

Darcy's Law describes a 1-D, single phase flow through a porous medium (coal seam) in the following manner:

$$v_x = -\frac{k_x}{\mu} \left(\frac{\partial P}{\partial x} - \rho g_x \frac{\partial D}{\partial x} \right) \quad (1)$$

Volumetric flux in the x direction, v_x , is a function of seam permeability, k_x , the fluid viscosity, μ , and the incremental pressure drop. The pressure drop relates to the difference in gas drainage borehole pressure and the seam gas pressure. Seam permeability is a dominant parameter for gas production rates. In Australian longwall mining environments the coal seams are typically comparatively level, as a consequence gravitational effects on gas flow are negligible.

Gray (1987) described permeability of a coal in relation to the changes in effective stress in the coal seam, where, if water is removed from the cleat, the matrix blocks are less constrained and tend to compress the cleat. This process, referred to as cleat compression, leads to a reduction in permeability. As the fluid pressure in the cleat system falls, gas desorption occurs. The release of gas from the matrix into the cleat subsequently causes the matrix to shrink and a reduction in effective horizontal stress. In terms of permeability changes, the two processes, cleat compression and matrix shrinkage tend to cancel each other. Gray relates the matrix shrinkage to effective stress in terms of a linear change in strain for a change in sorption pressure.

Palmer and Mansoori (1998) proposed a relationship for relative changes in permeability using the cubic relationship between permeability and porosity (cleat volume)

$$\frac{k}{k_o} = \left(\frac{\Phi}{\Phi_o} \right)^3 \quad (2)$$

where k_o and Φ_o are reference permeability and reference porosity respectively. Equation 2 reads that permeability normally depends on pore throat volume, the more open the cleat the greater the permeability. Palmer and Mansoori (1998) show the relative change in porosity in response to change in reservoir pressure is given by:

$$\frac{\Phi}{\Phi_o} = 1 - \frac{c_m}{\Phi_o} (p - p_o) + \frac{\epsilon_l}{\Phi_o} \left(\frac{K}{M} - 1 \right) \left(\frac{\beta p}{1 + \beta p} - \frac{\beta p_o}{1 + \beta p_o} \right) \quad (3)$$

Where c_m is matrix compressibility, K is bulk modulus, M is unconstrained axial modulus. The three parameters are derived from the coal geo-mechanical properties of Young's modulus and Poisson's Ratio. The terms ϵ_l and β are parameters matching volumetric strain caused by matrix shrinkage resulting from gas desorption. Reservoir pressure and initial reservoir pressure are p and p_o respectively.

Change in coal bed permeability in relation to change in effective stress is described by Seidle (Seidle, Jeansonne and Erickson 1992):

$$k = k_0 e^{-\varepsilon_c (\sigma - \sigma_0)} \quad (4)$$

where the parameter, ε_c is cleat-volume compressibility. Shi and Durucan (2004) developed a relationship to enable calculation of change in horizontal effective stress resulting from changes in reservoir pressure and desorption of gas from the coal:

$$(\sigma - \sigma_0) = -\frac{\nu}{1-\nu} (p - p_0) + \frac{E}{3(1-\nu)} \varepsilon_l \left(\frac{p}{p + P_s} - \frac{p_0}{p_0 + P_s} \right) \quad (5)$$

where ε_l and P_s are matrix shrinkage constants, ν and E are the Poisson's ratio and Young's Modulus of the coal, respectively. Initial or reference horizontal stress and pore pressure are σ_0 and p_0 , respectively. The two terms on the right hand side of the equation relate to cleat compression and matrix shrinkage respectively.

Volumetric shrinkage strain is considered in both the Palmer/Mansoori and Shi/Durucan formulations to be related to the Langmuir type relationship of matrix strain at maximum adsorbed gas content and the gas content pressure at which half of the maximum strain occurs:

$$\Delta \varepsilon_s = \varepsilon_l \left(\frac{\beta p}{1 + \beta p} - \frac{\beta p_0}{1 + \beta p_0} \right) = \varepsilon_l \left(\frac{p}{p + P_s} - \frac{p_0}{p_0 + P_s} \right) \quad (6)$$

where $P_s = 1/\beta$. Shi and Durucan (2005) further developed this relationship to account for matrix swelling (as may occur where a gas adsorbs onto the coal matrix in enhanced gas drainage). Assuming the pressure of the free gas in the cleat is in equilibrium with the adsorbed gas, then:

$$\Delta \varepsilon_s = \alpha_s (V - V_0) \quad (7)$$

where α_s is the volumetric shrinkage/swelling coefficient for a specific gas (i.e. a seam gas, methane or an injectant gas such as nitrogen), V corresponds to the gas content at reservoir pressure, p ; V_0 is the gas content at initial reservoir pressure p_0 . V and V_0 can be determined using the Langmuir isotherm (Zuber 1996):

$$V = \frac{V_L \beta p}{\beta p + 1} \quad (8)$$

where β is the Langmuir constant, and V_L is the Langmuir volume defining the adsorption isotherm for a single gas in a specific coal seam. This allows the change in effective horizontal stress to be determined resulting from cleat compression and matrix shrinkage or swelling due to change in pore pressure:

$$(\sigma - \sigma_0) = -\frac{\nu}{1-\nu} (p - p_0) + \frac{E \alpha_s V_L}{3(1-\nu)} \left(\frac{\beta p}{1 + \beta p} - \frac{\beta p_0}{1 + \beta p_0} \right) \quad (9)$$

It can be seen that change in permeability of a coal seam can be related to either change in porosity (equation 3) or effective horizontal stress (equation 5) and that both formulations are dependent on change in reservoir pressure. The effect of matrix swelling, resulting from adsorption of an injectant gas into the coal may also be determined (equation 9).

In an enhanced gas drainage process using nitrogen as an injectant, the coal matrix desorbs one gas, generally methane or carbon dioxide, and adsorbs nitrogen. The net matrix shrinkage effect is thus

determined by the volumetric shrinkage coefficient, α_s and the Langmuir isotherm parameters for the desorbing and adsorbing gasses.

Diffusional Flow

Diffusion of gas from the coal matrix into the cleat system may be described by the modified Fick's law (Zuber 1996):

$$q_{gm} = \frac{8\pi DV_m}{s_f^2} (C_m - C(p)) \quad (10)$$

where gas production rate q_{gm} , is a function of matrix volume, V_m , and the difference between the matrix gas concentration, C_m , less the equilibrium concentration at the matrix cleat boundary $C(p)$. The diffusion coefficient, D , and fracture spacing s_f , are normally resolved by the use of desorption time, τ , which is derived from gas content testing.

$$\tau = \frac{s_f^2}{8\pi D} \quad (11)$$

It is significant that the gas production rate is a function of difference in the gas concentration rather than the gas pressure. In a primary production (pressure depletion), the gas composition in the matrix is the same as the gas composition in the cleat, then difference between C_m and $C(p)$ is proportional to the difference between the cleat pressure and adsorbed gas pressure. If however the cleat system is flooded with an inert gas, such as nitrogen, then difference in concentration of the seam gas between the matrix and cleat is significantly increased.

Relative Permeability

A third property which regulates the gas flow through the cleat system is the relative permeability of gas and water within the cleat at varying water saturation levels. In simple terms, when the cleat system is saturated with only water no gas will flow; as the water saturation decreases the effective permeability of gas slowly increases and gas may begin migrating through the cleat (Figure 1). This is the reason why coal seams must be dewatered for successful gas drainage. In relation to enhanced gas recovery, where an injectant is introduced into a cleat system the coal matrix may be compressed and the cleat volume increased.

Because water is only slightly compressible the water volume in the cleat system remains roughly constant but the cleat volume increases, resulting in an apparent

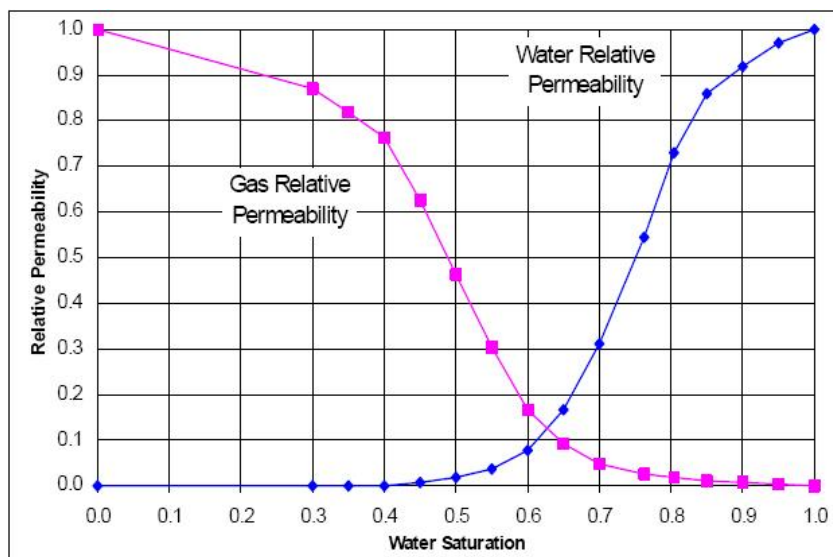


Figure 1 - Relative Permeability Curve for gas and water (permeability <1mD)
(SIMED2 handbook 2003 pp.16)

reduction in water saturation. The effect is not only to increase absolute permeability but to reduce the cleat water saturation and thus improve the gas relative permeability. (Gray 1987; Stevenson Pinczewski and Downey 1993; Mavor and Gunter 2004).

1. Increased permeability resulting from a change in effective horizontal stress or cleat porosity (Equations 3, 5 and 9)
2. Increased concentration gradient between the matrix and cleat interface and thus diffusion rate (Equation 6)
3. A reduced water phase saturation in the cleat system resulting in an improved effective permeability to the gas phase.

FIELD TRIALS

Coal Mine Field Trials

There are no documented cases of an injectant gas being used for enhanced gas recovery in coal mine gas drainage systems. Two references to the possibilities that nitrogen may provide in relation to coal mine gas drainage focus on the potential for improved drainage in low permeability environments (Thakur 2006; Brunner 2007).

Thakur (2006), suggested that gas flooding using nitrogen or carbon dioxide may be a solution to drainage in low permeability (<1mD) environments. Brunner (2007), claimed that for a 0.1mD permeability reservoir, nitrogen enhanced drainage would achieve a 50% gas content reduction in 7.2 months compared to 12 months for hydraulic fracture stimulation, and 24 months for traditional pressure depletion. Brunner does not provide site characterisation details or well/borehole geometry.

Enhanced coalbed methane (ECBM) field trials

Nitrogen has been used as an injectant in three ECBM field trials. The trials were conducted to examine the potential for CO₂ sequestration with associated enhanced methane recovery. Nitrogen injection was conducted to develop an understanding of the behaviour of the gas in coal seams.

Tiffany trial, San Juan basin, Colorado, US

The Tiffany trial was conducted in an existing CBM operation which utilized vertical wells drilled to intersect 4 seams of ~14.3m total thickness at an average depth to top of the highest seam of 926m (Reeves and Oudinot, 2004). The production wells were spaced 320 acre. CBM primary production began in 1983; ECBM utilizing nitrogen began in 1998 and concluded in 2002. During the trial the methane production rate increased by fivefold (Figure 2). Initial seam permeability was assumed to be 8 mD, and porosity of 0.2%. An anticipated feature of the trial was the nitrogen breakthrough at the production wells, and the reduced water flow rate at the production wells.

Fenn-Big Valley trial, Alberta, Canada

The Fenn-Big Valley trial occurred between 1998 and 2000. The trial involved two wells, one of which was an existing oil well which had been drilled through coal measures, the other well was purpose drilled. The oil well was re-completed to allow access to a Medicine River seam at a depth of ~1259m. Both wells were subject to CO₂ injection subsequently experiencing losses in injectivity. Injection trials using nitrogen and flue gas demonstrated increases of absolute permeability from initial conditions of 1.2mD to 13.8mD for nitrogen injection and 0.985mD to 23.7mD for the flue gas injection (Mavor, Gunter and Robinson 2004). Mavor describes '*the injection ballooned the natural fracture system and substantially increased the permeability*'.

Yubari trial, Hokaido, Japan

The Yubari trial in the Ishikari coal field, Japan, was a CO₂ sequestration trial conducted between 2003 and 2008 (Shi, Durucan and Fujioka 2008). The trial involved drilling two wells to access a coal seam at about 890m depth. The injection well (IW-1) was subject to a period of initial pressure depletion, followed by multi-well production tests involving the injection of CO₂ at IW-1 and subsequent monitoring of pressure, flow and gas composition characteristics at production well, PW-1. The results from the CO₂ injection trials indicated a significant loss in injectivity due to matrix swelling and associated permeability loss, (a similar effect had been observed in a CO₂ ECBM trial in the San Juan Basin). Permeability fell from 1mD to 0.1mD due to CO₂ injection. In an attempt to improve the CO₂

injection rate N_2 was injected at IW-1. Modelling of the results indicated that an improvement in well block permeability from 0.1mD to 40mD was achieved. This improvement in permeability enabled a temporary four-fold increase in subsequent CO_2 injection.

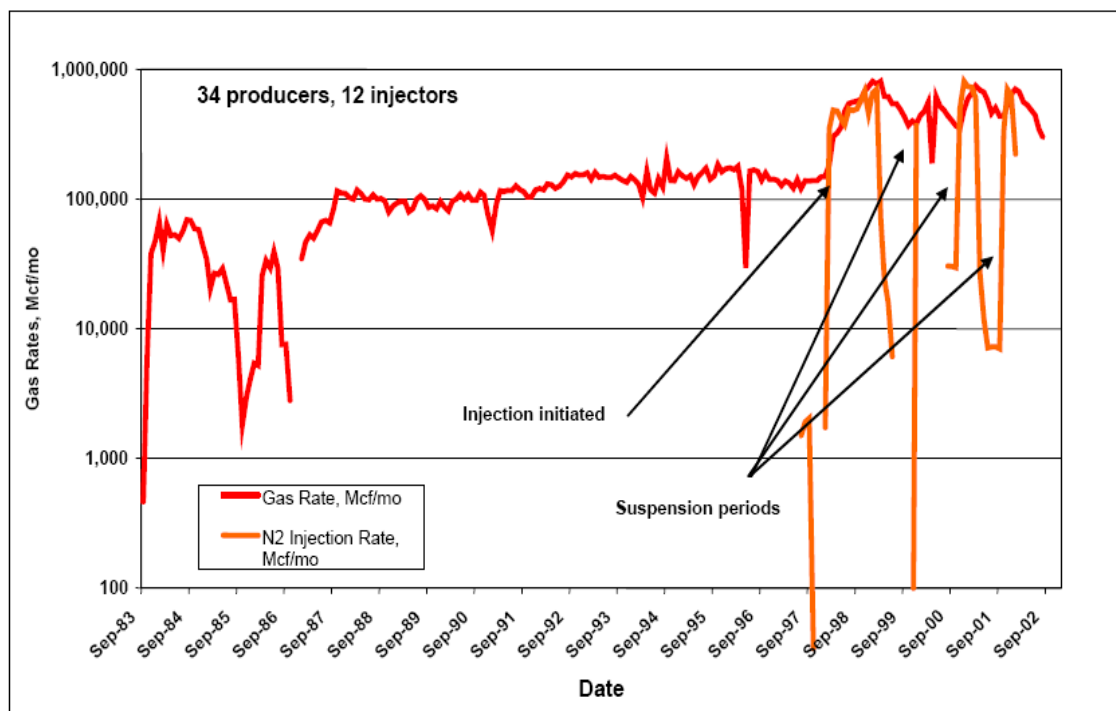


Figure 2 - Coal seam methane production and nitrogen injection gas rates for the Tiffany project (after Reeves and Oudinot 2004)

The trials all indicate improved permeability resulting from nitrogen injection, in the case of the Tiffany trial improved gas production. On the basis of the theoretical affect of nitrogen as an injectant in an enhanced gas system and from the ECBM field trial results it is reasonable to assume similar effects may be achievable in coal mine gas drainage systems.

CONCEPTUAL APPLICATION TO COAL MINE GAS DRAINAGE SYSTEMS

Coalmine gas drainage system may be considered as either surface based or underground based. Surface drainage systems in Australia predominantly utilise surface to in-seam (SIS) holes, using medium radius drilling techniques. Underground drainage systems involve drilling groups of horizontal holes typically <600m long from purpose driven stubs off development roadways.

Surface Drainage Systems

SIS holes involve the drilling of an inclined hole from the surface through overburden to enter the target seam at close to seam dip. After entering the seam, the SIS hole is drilled to intersect a vertical well typically 1-2 km down dip. A pump is installed in the vertical well to dewater the SIS hole. The use of vertical wells independent of a SIS lateral for pre-drainage is not common in the Australian coal mining industry. SIS holes are generally drilled parallel to development roadways. The ability to conduct SIS drilling from the surface provides the opportunity for drainage times to be several years.

Where parallel SIS holes are prepared for pre-drainage of a proposed development roadway a simple application of enhanced gas recovery would be to use one SIS hole as an injector and on SIS hole as a producer well (Figure 3). This geometry is likely to have effective drainage between the wells, ie the coal in which the proposed development roadway is to be driven, however may be less effective in draining seam gas in the longwall block side of the SIS holes. This arrangement may be suited to an environment where a high risk of frictional ignition is present.

The same configuration of gas drainage wells may be used in relation to specific regions identified with inadequate drainage. When gas drainage is being conducted primarily for management of an outburst hazard, development roadway drifage is prohibited unless residual gas contents are below pre-established outburst threshold values. The gas content residual value is often determined by a vertically cored borehole into the pillar in advance of the development drifage.

High residual gas contents may arise due to lack of drainage time; unusually high virgin gas content or unusually low localised permeability. Where a 'compliance' core returns a result that is not below the outburst threshold value the options available to mine management are to allow more time for gas drainage to occur, to drill further gas drainage holes from underground to accelerate the drainage, or to adopt a remote mining technique such as shotfiring. Each option has significant scheduling and cost implications for longwall operations. An alternative may be to use an enhanced gas recovery process utilizing the compliance borehole as an injection well (Figure 4).

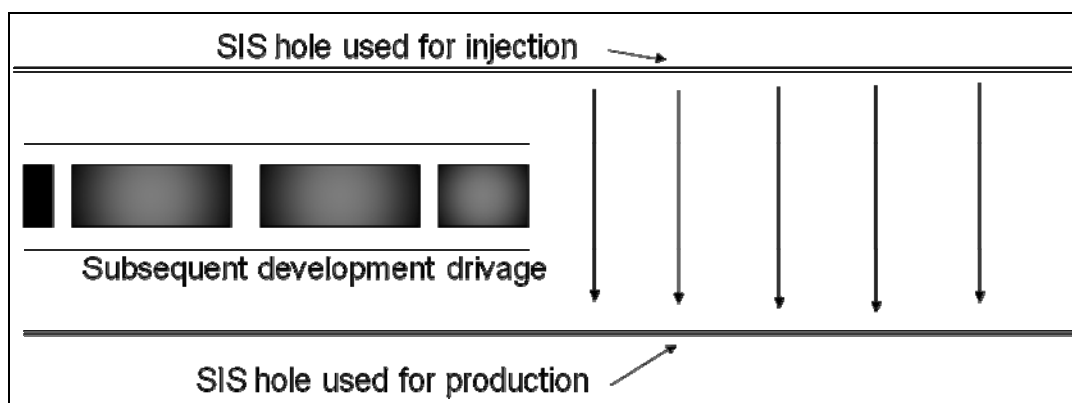


Figure 3 - Parallel SIS holes as injector and producer wells

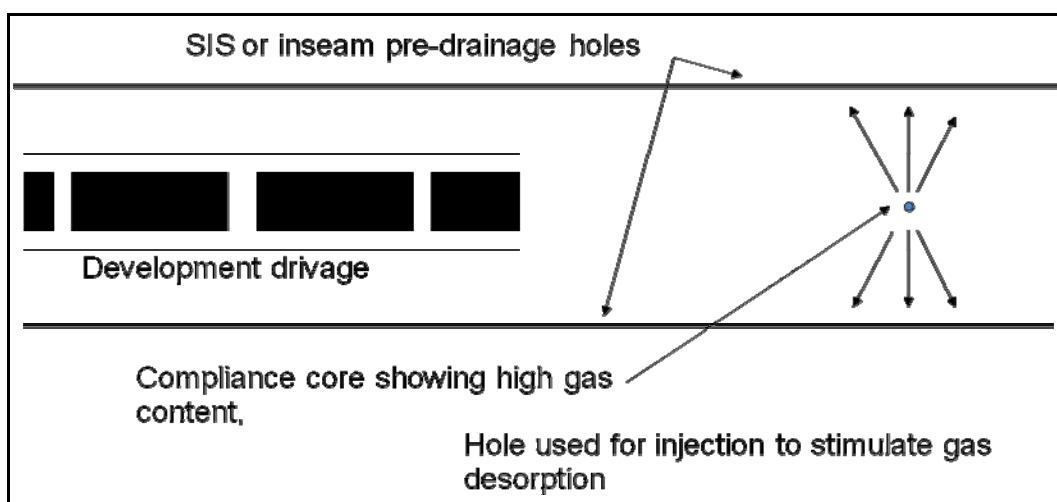


Figure 4 - Use of a compliance borehole for localised enhanced gas drainage

Underground Gas Drainage Applications

Underground drainage is often conducted where there is inadequate lead time for surface drainage; where access to surface drilling locations are impractical; or where depth of cover makes drilling cost prohibitive. Due to practical considerations underground gas drainage holes are typically 300-600m long. The restriction of the hole length has implications for the drainage time available. The reduced gas drainage time is generally offset by reducing spacing between drainage holes (typically 50-70m). In coal mining environments where U/G gas drainage is conducted to manage outbursts similar problems may arise as described above. An enhanced gas drainage system utilizing an underground

gas drainage layout may be feasible using alternate drainage holes as injectors (Figure 5). Such an approach would be subject to problems associated with highly heterogeneous seam conditions i.e. localised faulting allowing rapid breakthrough of the injectant to the producing hole. Furthermore the proximity of the hole collars may lead to rapid breakthrough and reduce flow at inbye sections of the hole.

In a gas drainage environment of very low permeability ($<0.01\text{md}$) achieving two phase drainage condition may be difficult (the permeability being so low that water flow in the cleat is minimal). In such conditions an injector/production borehole arrangement may not be effective; an alternative may be to adopt a 'huff-puff' process of cyclic injection then bleeding off of the injectant/seam gas mix. The procedure would be continued until the localised permeability had improved to allow an injector/production borehole arrangement.

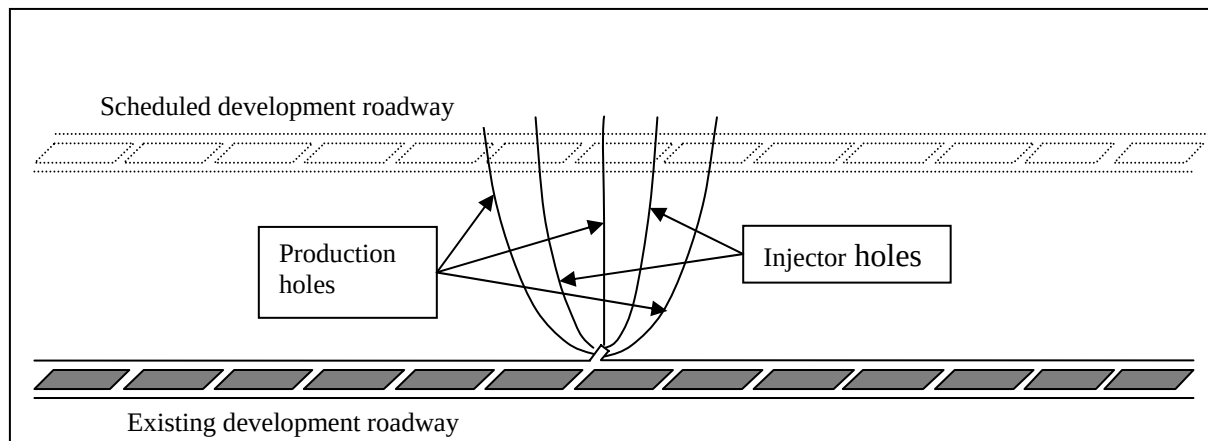


Figure 5b - Schematic layout for an underground enhanced gas drainage layout

Injector Operational Considerations

The operational pressure of an injector is typically close to hydrostatic pressure in the reservoir. In a reservoir that already undergone some pressure depletion such as an existing surface gas drainage installation, the pressure of the injectant gas would be greater than the pore pressure of the region to be subject to enhanced gas recovery.

Bowen basin coal mine surface gas drainage operations have typically 150-250m of cover. Injectant pressures of 1-2 MPa would be feasible where primary production had been undertaken. Illawarra coal mines typically operate in the Bulli seam at initial gas contents of 10-20 m³/t. Hargraves (1995) reported that insitu gas pressures of 4 MPa have been measured at Appin Colliery, Lama (1995) states pressures of up to 4.6 MPa have been measured underground in the Bulli seam.

Infrastructure

Existing underground compressed air ranges operate at ~700 kPa using victualic type couplings. Operating a compressed air victualic range to transport compressed nitrogen would be feasible up to 1.2 MPa. At the depth of workings of most Australian mines, a gas drainage system in primary production could be expected to have a pore pressure of less than 1.2 MPa.

The surface facilities of typical SIS wells use ANSI 300 fittings, with a maximum pressure rating of 4.65 MPa (675 psi). Compressors are available for operation at 3.4 MPa and 420 l/s.

Application of an injectant gas would be comparatively simple at pressure less than 1.2 MPa for underground operation and up to 3.4 MPa for a surface installation. Higher operating pressures may be possible but would require purpose designed standpipes and delivery pipework for underground applications, and wellhead arrangements and compression facilities for surface applications.

Injectant gas source

Coal mines in the Bowen Basin routinely use inert gasses to accelerate the transition of newly sealed goafs to non-explosive atmospheres. The inertisation involves injection of nitrogen or flue gas to displace or dilute oxygen in goaf regions. Facilities for the production of inert gasses (routine and emergency) include liquid nitrogen systems, membrane nitrogen systems and flue gas generators.

Liquid nitrogen systems are comparatively expensive and suffer from cryogenic (hazardous goods) transport limitations. The latter issue would be particularly significant for continuity of supply to Bowen Basin mine sites.

Existing flue gas generators are not directly suitable for enhanced gas recovery at mine sites. Flue gas contains nitrogen as well as CO₂ however requires a catalytic converter to remove residual oxygen and scrubbers to remove carbonic acid. Use of flue gas as an injectant in a coal mine enhanced gas recovery system would require careful management to avoid generating problematic CO₂ concentrations in the coal reserve.

Membrane nitrogen filtration systems are in use at mines in the Bowen Basin and Hunter valley. The "AMSA" membrane units generate nitrogen at ~97% purity and an outlet pressure of 900 KPa. Units in use in the mining industry have flow rates of 120 and 500 l/s. Use of membrane systems currently at mine sites is considered feasible as a source of injectant gas for enhanced gas recovery. The membrane units are self contained and have good reliability and require only a power supply to operate (Figure 6).



Figure 6 - "AMSA" Membrane filter at a Hunter Valley Coal Mine

SUMMARY

Enhanced gas recovery in coal mine gas drainage operations has the potential to be a step change in drainage practice. Drainage in low permeability conditions and low residual gas level objectives, which had hitherto been impractical, appears technically feasible.

Implementation of an enhanced gas recovery scheme at a minesite where primary gas production is being conducted (pressure depletion) and seam pore-pressures are below 900 kPa offers no insurmountable problems.

The next stage of this research will involve detailed site characterisation and field trial preparation with a view to undertaking a field trial mid 2009.

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COAL BUMPS IN AN EASTERN KENTUCKY USA LONGWALL COAL MINE 1989 TO 1997

John Hoelle¹

ABSTRACT: Coal mines in southern West Virginia, south-western Virginia and eastern Kentucky have experienced coal bumps at least since 1933. Most of the bumps have occurred due to high cover, strong roof and floor strata and stress concentrations due to the mining sequence. A longwall mine in eastern Kentucky first experienced coal bumps on the tailgate side of the longwall face in 1989. The bumps continued until 1996. The bumps were the result of:

- thick overburden up to 670m.
- strong roof and floor (strata strengths up to 177 MPa UCS and elasticity modulus up to 33.1 MPa)
- previous over-mining in places
- sandstone channels

Not all characteristics occurred simultaneously. The bumps produced seismic events recorded up to 4.3 (Richter scale magnitude), and damaged pillars that were up to 45 by 46 m in size. During the eight years that the bumps occurred, a large quantity of data was obtained in an effort to develop methods to predict an event, and reduce or eliminate the bumps.

- In-situ strength properties of floor, coal and roof strata
- Lab testing of floor, coal and roof samples
- Monitoring gate road pillar response with stress metres, extensometers and convergence stations
- Shield leg response
- Monitoring in an effort to determine precursors was conducted using a digital microseismic monitoring system.
- Back calculation of gate road pillar strength

A number of different remedies were trialled in an effort to eliminate or decrease the severity of the bumps.

- The gate road longwall design was varied
- Pillar size and shape
- 3 and 4 entry gate road designs
 - water infusion in longwall panels,
 - hydraulic induced face bumps,
 - disruption of the roof strata

A yield-abutment-pillar design was the most effective method in reducing the affect of bumps by moving the events onto the abutment pillars, but the bumps were never eliminated and adequate precursors and advanced warnings were never achieved.

INTRODUCTION

This mine is in a region that was characterised by deeply incised valleys and high ridges which were over 640 m above the stream valley bottoms. The strata is predominantly sandstone and there are a number of coal seams above and below drainage. Room-and-pillar mines in the immediate area of the subject mine recorded bumps as far back as 1933 (Coughlin and Rowell, 1993, Holland, 1942) and probably bumps were experienced in the years prior to 1933. The mine started in 1972 as a room-and-pillar drift mine and was developed as a longwall mine starting in 1981. Longwall reserves were exhausted in late 1997 and the mine closed in the first quarter of 1998 when economic reserves were exhausted. The coal seam was the Harlan seam and ranged in thickness from approximately

¹ Anglo Coal, Australia

2.5 to 3.7 m in economical mining thickness. The seam occurred in four benches and in some areas, the parting between the upper bench and the three lower coal benches or the parting between the lower bench and the three upper benches became too thick to mine. A typical section is shown in Figure 1. The seam dipped 2 to 3 degrees to the south. The access portals were drift portals with four additional sets of intake and return portals – all drifts were located at the north-end of the reserve. The overburden gradually increased as mining advanced to the south with overburden thicknesses up to 670 m.

Based on movement of the coal pillars along coal bench-parting contacts and information from nearby mines, there was a regional stress field to the north-east, which trended approximately N55E. Three in-situ stress tests were conducted at the mine. There were five groups of panels with the last three shown in Figure 2. Group 3 is shown with five panels (numbered 1 through 5) and Group 4 had four panels (numbered 6 through 9 in Figure 2). Group 5 is shown with 11 panels. The panel widths were 152 m for Group 3 and 4 shown in Figure 2. Panels 1 through 11 in the last group were 183 m wide.

The first bump in this mine occurred in April 1989 and the last bump occurred in May 1996. There have been a number of papers published on the bumps that have occurred at this mine (Coughlin and Rowell, 1993, Heasley and Zelanko, 1992, Mark and DeMarco, 1992, Rowell, and Lemons, 1991, and Zalenko, Rowell and Barczak, 1991, Hoelle 2008). The major bumps that occurred are summarised in Table 1. A major bump was defined when mining had to be stopped, a significant quantity of coal was displaced, equipment damaged, ventilation disruption or similar criteria. A minor bump consisted of noise, small quantities of coal displaced, ground bounce, but no major large coal moved or equipment damaged. The terms were very subjective and the definitions were not well defined.

This paper is a presentation of what occurred in this mine, what mitigating efforts were attempted, and the success of these efforts.

Table 1 - Summary of major bump events

Panel No. (Group-panel)	Date	Seismic Magnitude	Overburden thickness m (ft)
3-3	18 Apr 89	N-A	357 (1,170)
3-3	8 May 89	N-A	396 (1,300)
3-4	22 Nov 89	2.3	488 (1,600)
4-2	25 Jul 90	N-A	674 (2,210)
4-2	8 Oct 90	N-A	668 (2,190)
4-3	11 Jan 91	N-A	543 (1,780)
5-6	3 Aug 94	3.5	646 (2,120)
5-6	5 Oct 94	3.6	640 (2,100)
5-7	23 Dec 94	3.5	518 (1,700)
5-7	19 Jan 95	3.7	594 (1,950)
5-7	11 Mar 95	4.0	518 (1,700)
5-8	22 Jul 95	3.4	518 (1,700)
5-8	5 Aug 95	2.8	579 (1,900)
5-9	25 Oct 95	4.3	502 (1,650)
5-10	19 April 96	3.7	495 (1,625)
5-10	4 May 96	3.7	533 (1,750)
5-10	13 May 96	3.5	572 (1,875)
5-10	16 May 96	2.0	579 (1,900)

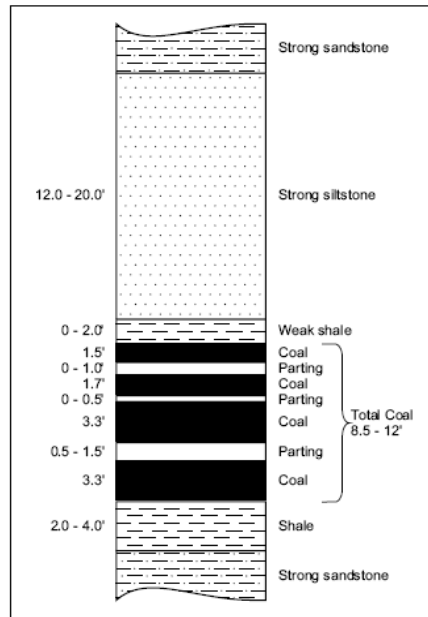


Figure No 1 - Typical stratigraphical section

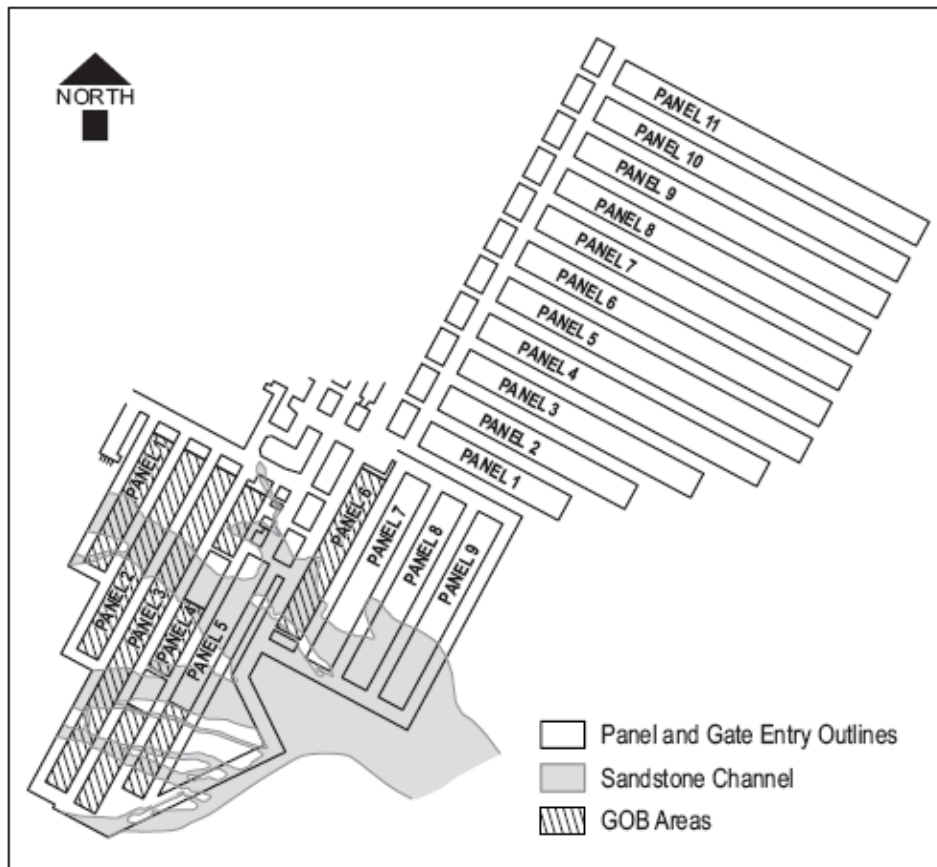


Figure No 2 - Mine plan for Groups 3, 4 and 5

HISTORY OF BUMPS

The initial two groups of longwall panels were mined without incident. The approximate maximum cover thickness in the first set of panels was 457m and 579m in the second set of panels. These panels were located north of the Group 3 panels shown in Figure 2.

The first set of 9 panels in Figure 2 was a three-entry abutment-yield configuration, although the "yield" pillar was not designed as a yield pillar. Mine personnel referred to these smaller pillars as yield pillars

or as the "small" pillars. The first two panels had pillar sizes of 2m wide by 30m long (abutment pillar) and 17 m wide by 30.5 m long for the smaller pillar. The abutment pillar was adjacent to the previously mined panel. The maximum cover was 610 m, although the subsequent bumps did not occur in the areas of highest cover. The first bump was encountered in the panel 3 of longwall panels in April 1989. The bump occurred on the edges of a sandstone channel that cut into the upper portion of the coal seam. The shearer was heavily damaged and removed from the mine for major repairs.

A second large bump occurred in May 1989 when the longwall face exited the same sandstone channel. The shearer was heavily damaged and was removed from the mine for major repairs. At that time, mine personnel believed that the bumps were the result of the sandstone channel when the sandstone comprised the immediate roof. In order to avoid this situation, the mine began to move the longwall around the known locations of sandstone channels since these appeared to be the cause of the bumps. Figure 2 shows the approximate location of the sandstone channels. A third bump occurred in the panel 4 in November 1989. The bump occurred as the longwall face was approaching but not under a sandstone channel. The shearer was heavily damaged and an in-panel move around the projected sandstone was conducted and the panel 4 finished without incident. After the first bump had occurred, the abutment pillar sizes between panels 4 and 5 were increased to 30.5 m wide by 30.5 m long with the smaller pillar width increased to 18 m. The pillar sizes for all of the groups are summarised in Table 2. Two of the abutment pillars in the headgate of Panel 4 were instrumented with an array of borehole platen flatjacks, coal pillar extensometers and convergence stations (Zalenko, Rowell and Barczak, 1991). The abutment pillars in the panel 4 headgate were increased to 33.5 m wide. Microseismic geophones were installed in the tailgate and headgate of panel 4 and shield support leg pressure response data was collected and analysed (Zalenko, Rowell and Barczak, 1991).

Table 2 - Summary of gate road configuration and pillar sizes

Group	No. of Panels in Group	Configuration	Abutment Dimension (Width by length) m(ft)	Small pillar (Width by length), m (ft)
1	4	3-entry abutment-yield	29 x 29 (95 x 95)	15 x 29 (50 x 95)
2	4	3-entry abutment-yield	29 x 29 (95 x 95)	15 x 29 (50 x 95)
3	2	3-entry abutment-yield	25 x 30.5 (85 x 100)	16.8 x 36.5 (85 x 100)
3	1	3-entry abutment-yield	30.5 x 30.5 (100 x 100)	18.3 x 30.5 (60 x 100)
3	1	3-entry abutment-yield	33.5 x 30.5 (110 x 100)	18.3 x 30.5 (60 x 100)
4	4	3-entry abutment-yield	42.7 x 36.5 (140 x 120)	15.3 x 36.6 (50 x 120)
5	2	4-entry yield-abutment-yield	47.5 x 47.5 (156x156)	12 x 47.5 (40 x 156)
5	1	4-entry yield-abutment-yield	46.4 x 47.5 (152 x 156)	12 x 47.5 (40 x 156)
5	7	4-entry yield-abutment-yield	45 x 46.3 (147 x152)	12 x 46.3 (40 x 152)

The instrumented pillars indicated that the pillars were failing prior to the bumps and that stress loading was being shifted across the gate road pillars onto the longwall panels, indicating that the

pillars were of insufficient size to control stress override. The first three bumps occurred within 12 m of the tailgate, which apparently agreed with the load shedding of the tailgate pillars. Review of the data indicated that the shield leg pressures did not indicate any significant change in pressure. The microcosmic activity, while showing an increase in the number and intensity, did not produce an obvious precursor.

Panel 5 of Group 3 was only partially mined with no bump occurrences and the longwall moved to the Group 4 set of 4 panels. There were some changes in Group 4. The gate road pillar sizes were increased to 36.5 m long by 42.7 m wide and 36.5 m long by 12 m wide. Again the "yield" pillar was not designed as a true yield pillar. The maximum approximate cover was a little over 670 m and the high cover areas flagged. This group of panels had adverse room-and-pillar mining barrier pillars and goaf (gob) areas approximately 40 m above the longwall mine and these areas were flagged.

The high cover and the projections of the sandstone channels indicated that there was still a potential for coal bumps. In an effort to reduce the potential for bumps, water infusion into the panel was initiated. Horizontal holes were drilled into the panel ahead of mining and water infused into the panel. The depths of the holes ranged from 21 to 95 m and were drilled from both the tailgate and headgate. In addition, the face of the longwall panel was hydro-fractured when the face appeared to be hard and standing straight. Due to the height of the seam and the overburden thickness, the panel face normally spalled readily. However, when the panel was taking weight and being loaded, the face stood straight with no spalling. The headgate entries of Panels 2, 3 and 4 were instrumented with convergence stations and one abutment pillar was instrumented with stress meters. Plexiglas shields were hung along the shield walk way to protect personnel from coal that may be ejected from the face. In order to try to reduce the affect of hanging roof, attempts were made to create a "pre-split" line along the tailgate and through the shields along the face. The tailgate holes could not be loaded. The holes along the shield line were drilled between the shields (on 1.5-metre centres, the shield width). The holes were shot; however, the height of the disrupted strata did not appear to be sufficient to initiate caving.

Mining started in April 1990 in Group 4 and the first panel (panel 6 in Figure 2) was mined without major incident but there were two minor bumps near the beginning of the panel. The second panel (panel 7 in Figure 2) had 2 minor bumps and then a large bump after mining a short distance into the panel, with the bump damaging the shearer. Since this bump was under the same sandstone channel feature that was affiliated with the bumps in Group 3, the decision was made to move the longwall around the sandstone channel. Seven more minor bumps occurred before two large bumps occurred on the 8th October 1990 when the face was under a high mountain peak as well as under a barrier pillar that was in the overlying adverse mine. At about the same time, an abutment pillar in the headgate bumped, totally destroying the integrity of the pillar. No additional bumps occurred during mining of the rest of the panel. The bumps that occurred in the longwall panel were within 46 m of the tailgate

The third panel (panel 8 in Figure 2) was started inbye the sandstone channel. There were no bumps until the panel was under a barrier pillar in the overlying adverse mine when a minor bump followed by a major bump approximately 46 m from the tailgate. The decision was made to conduct an in-panel move around the high cover area that caused the bumps in the second panel. This also avoided the area of the previously destroyed abutment pillar in the tailgate. There were no further bumps in the rest of panel 8 or in panel 9.

The Group 5 set of 11 panels was started in October 1991. These panels were oriented at 90° to the orientation of the first four set of panels, as shown in Figure 2. Since a large number of bumps occurred in Group 4, the size and configuration of the gate road pillars were evaluated and major changes were made. The abutment pillars were designed as a 4-entry yield-abutment-yield arrangement. In the initial two panels, the gate road pillars were designed with the length at 47.5 m and the width at 12 and 47.5 and 12 m respectively. As part of this re-design, the smaller pillars were designed as true yield pillars. The length of the pillars was shortened to 46 m in the third panel and the 4th through the 11th abutment pillars were 45 m long by 46 m wide. The yield pillars remained 12 m wide. The pillar configurations are summarised in Table 2. The memory cut capabilities of the shearer continued to be used in the Group 5 panels.

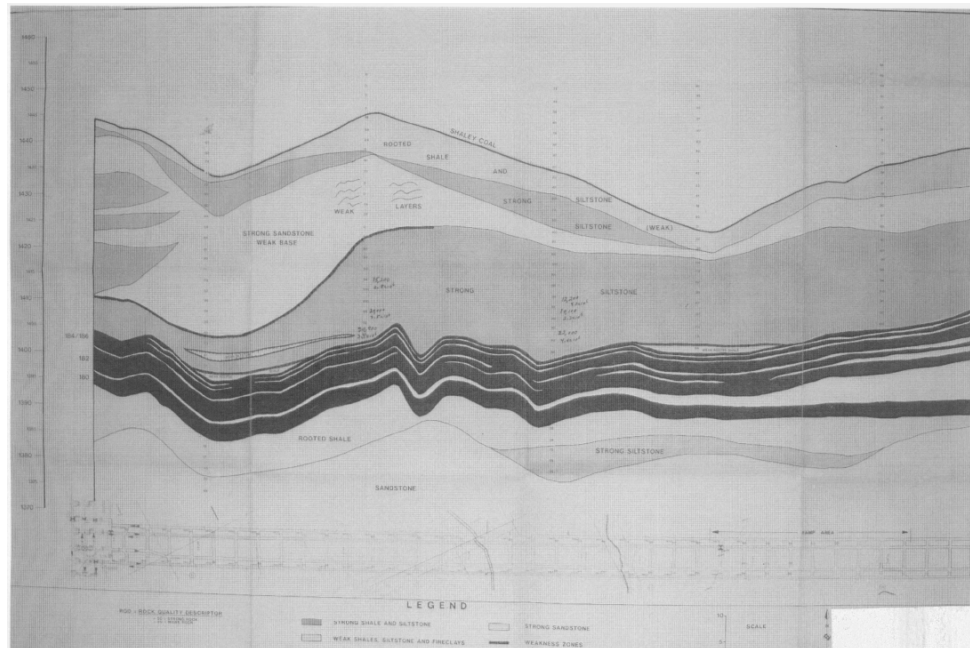


Figure 3 - Profile of gateroad

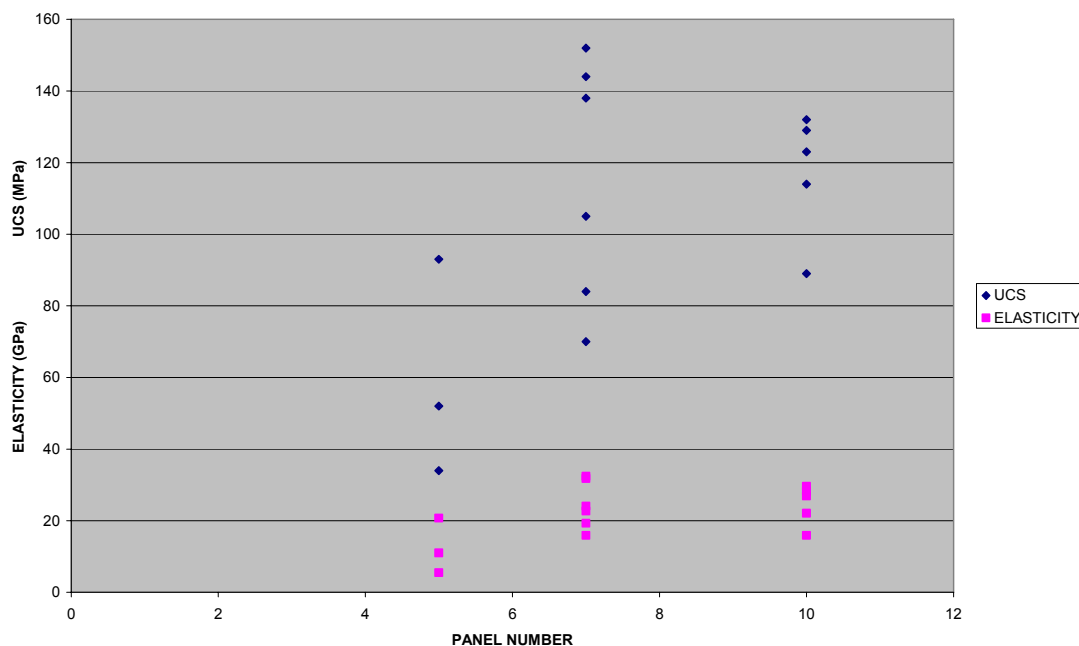


Figure 4 - Roof rock strengths

Starting with the panel 3 headgate, core holes were drilled 2.5 to 3.0 m into the floor and 12 to 15 m into the roof from the gate roads. The purpose of this was to determine if thick sandstone roof strata was present. A strong sandstone on or near the coal was not encountered; however a strong siltstone did occur within 0.6 m of the coal seam. A typical profile from the drilling is shown in Figure 3. Along with strata identification, RQD's were recorded and starting with the fifth panel, samples were tested for UCS and Young's modulus strength characteristics. The test results are illustrated in Table 3. The UCS strengths and the Young's modulus increased from south to north and from west to east. Figure 4 illustrates the increase in UCS and Young's modulus. The first five panels were bump-free even though the overburden thicknesses were up to 670 m. Because the first five panels were bump free, it was decided that the abutment pillars were properly designed to prevent bumps from occurring.

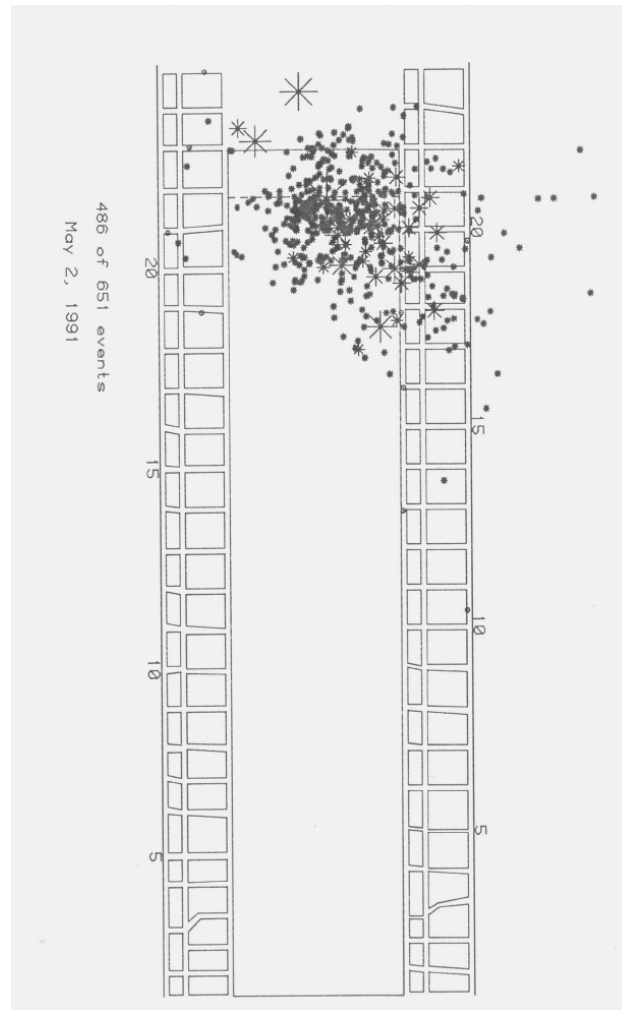


Figure 5 - Microseismic activity, Group 4, Panel 4

The first bump in the Group 5 set of panels occurred on 3rd August 1994 in an abutment pillar next to the longwall tailgate in panel 6 when the overburden reached 646 m. Four additional bumps in abutment pillars occurred in this panel with the cover over 640 m. The seismic Magnitudes ranged from 3.0 to 3.6. After the second bump, hydro-fracturing of the face was conducted every third cut, memory cutting was again initiated, and shearer initiation of the shields commence when the shearer was past the 131 m mark towards the tailgate (of a 183-metre wide face). Many of the abutment pillar failures were accompanied by short-term reversal of ventilation, high dust levels and air blasts, some of which tumbled personnel.

Panel 7 had seven significant bumps in abutment pillars adjacent to the active longwall when the cover thickness was approximately 640 metres. The Magnitude events ranged from 2.8 to 4.0. The panel was stopped approximately 520 m from the recover room due to the number of bumps on the tailgate abutment pillars. Panel 8 had two significant bumps in abutment pillars adjacent to the active longwall when the cover thickness exceeded 580 m. The Magnitudes ranged from 2.8 to 3.0. There were several shearer-initiated bumps at the face. Panel 9 had one significant bump in an abutment pillar and several face bumps near the tailgate. The abutment pillar event registered a Magnitude of 4.3. Panel 10 had three significant bumps. These bumps occurred when the cover exceeded 488 m. The bumps occurred at a lower cover thickness than previously experienced, possibly due to the increased strength

SUMMARY

During the eight years that these bump events were occurring, geological/geotechnical characteristics of the strata and of the coal pillars were obtained. A number of methods to prevent or reduce the affect of the bumps were trialled.

- In-situ strength properties of floor, coal and roof strata were obtained (Zalanko, Rowell and Barczak, 1991).
- Lab testing of floor, coal and roof samples was conducted (Zalanko, Rowell and Barczak, 1991, Newman and Hoelle, 1993).
 - o A systematic program of obtaining coal strengths for the four benches and the partings between these benches was conducted. The results of this program are discussed in reference 8.
 - o Lab strength characteristics were obtained from roof strata samples from an in-mine exploratory drill program.
- Gate road pillar response was monitored using stress metres, extensometers and convergence stations (Zalanko, Rowell and Barczak, 1991). The results were somewhat inconclusive. Either the longwall moved around the area being monitored or the signal was lost from the monitors at the time data was most important.
- Shield leg pressures were monitored (Zalanko, Rowell and Barczak, 1991). Three sets (located near the headgate, middle of panel face and near the tailgate) were fitted with data loggers which recorded the hydraulic leg pressures. A precursor of impending bumps or high stress was not noted.
- Geologic mapping of thick sandstone layers was conducted by in-mine mapping and in-mine exploratory drilling.
- Potential high stress areas of high cover and adverse conditions due to over mining were flagged.
- Extensometer data obtained in the panel 5 of Group 3 indicated a yield zone on the periphery of the abutment pillars of up to 5 m (Zalanko, Rowell and Barczak, 1991)). Auger drilling conducted during the hydrofracturing process in the face in panels in Group 4 and 5 indicated that the stressed coal core was approximately 5 m from the face.
- Starting with some of the initial bumps, the mine obtained the Magnitude for each from seismic stations located at the University of Kentucky in Lexington and VPI in Blacksburg Virginia. The Magnitude for some of the major events is listed in Table 1.
- The US Bureau of Mines installed geophones in the head and tail gates starting in Panel 4 of Group 3 with the objective of determining if precursors of events could be obtained. An example of the number of events that were recorded in one of the panels (Group 4, panel 4) is shown in Figure 5. However, a definitive precursor was never determined (Coughlin and Rowell, 1993). Microseismics were recorded from mid 1989 until mid 1993.
- The size and configuration of the gate road pillars were adjusted several times.
- Water infusion ahead of the face did not appear to have an affect; possibly due to the water infusion having to occur well in advance of the longwall. There was no noticeable increase in water content when these areas were mined through.
- Hydro fracturing of the face was affective if done before the stresses built up
- Attempts to stop hanging roof over the shields were not successful
- Use of memory cut and shearer initiation of shields reduced personnel exposure

There was not a single cause for the bumps that occurred from 1989 to 1997. Almost all of the major bumps that occurred in Groups 3 and 4 occurred on the tailgate portion of the longwall panel. The major bumps that occurred in Group 5 were on the abutment pillars. The main causes appeared to be:

- Overburden thickness
- Thick strong sandstone as immediate roof, possibly creating hanging roof over gob
- A strong siltstone or sandstone present as the main roof, with a high elastic modulus or a high UCS value
- Thickness of the main roof
- Adverse over mining configuration
- Tailgate lagging behind headgate creating a stress point
- Rapid mining into the stressed panel core

Table 3 – Lab strength test results, in-mine exploration program

Panel	Strata type	Above seam m (ft)	UCS MPa (psi)	E GPa (psi x10 ⁶)	RQD
5	siltstone	0.6- 4.9 (2 to 16)	53 (7,500)	11 (1.6)	78 -100
5	siltstone	0.6 – 4.9 (2 to 16)	35 (5,000)	6 (0.8)	78 -100
5	siltstone	0.6 – 4.9 (2 to 16)	93 (13,500)	21 (3.0)	78 -100
7	siltstone	0 – 6.1 (0 to 20)	105 (15,200)	1.9 (2.8)	76 -100
7	siltstone	0 – 6.1 (0 to 20)	138 (20,000)	24 (3.5)	76 -100
7	siltstone	0 - 6.1 (0 to 20)	144 (20,900)	22.8 (3.3)	76 -100
7	siltstone	0 -6.1 (0 to 20)	84 (12,200)	32.4 (4.7)	84 -100
7	siltstone	0 -6.1 (0 to 20)	70 (10,100)	15.9 (2.3)	84 -100
7	siltstone	0 -6.1 (0 to 20)	152 (22,000)	31.7 (4.6)	84 -100
10	sandstone	0 -6.1 (1 to 20)	89 (12,900)	15.9 (2.3)	67 -100
10	siltstone	0 – 1.5 (0 to 5)	132 (19,200)	26.9 (3.9)	62 - 85
10	sandstone	1.5 -6.1 (5 to 20)	114 (16,600)	22.0 (3.2)	50 -100
10	sandstone	1.5 -6.1 (5 to 20)	129 (18,700)	30 (4.3)	50 -100
10	sandstone	1.5 -6.1 (5 to 20)	123 (17,900)	28.3 (4.1)	50 -100
10	shale	2.1 – 5.8 (7 to 19)	177 (25,700)	31 (4.5)	52 -77
10	siltstone	0 -1.1 (0 - 4)	117 (16,900)	33 (4.8)	65

However, not all of the features listed above were required to create a bump situation. While a high cover thickness was a common feature, bumps occurred under thicknesses varying from 488 to 670 m. There were instances where a bump occurred unexpectedly since the cover height was less than what was anticipated to cause a bump. The mine was never able to totally eliminate the bumps, but a

combination of gate road design, reducing the stressed face by hydro fracturing and limiting the exposure to personnel by memory cut and shield initiation by the shearer greatly reduced the affect on mining.

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DUST CONTROL ON LONGWALL FACES BY FINE MIST (ATOMISING) SPRAYS - CAN THEY REALLY WORK?

Ian McDonell¹

ABSTRACT: The ongoing efforts to suppress dust emissions from longwall face shearing and transporting operations have met with varying degrees of success. A particular of concern has been the thicker, gas drained seams where both respirable and combustible fallout dust production has not always met with successful suppression methods using water spray system. There are current programs planning to use very fine atomising sprays that give a water droplet size approaching the respirable dust particle size, and mines are waiting to see how well these systems will work. Some peripheral science such as fluid mechanics and aerodynamics are discussed to determine whether other factors are present that may inhibit the ability of water spray systems to successfully reduce fine dust emissions to acceptable levels.

It appears from the initial investigation that water droplets cannot capture all the dust particles generated during coaling operations, and that chemical additives to the spray water may also be limited in their success. The paper also looks at possible methods to better suppress dust, and suggests that far more research and engineering may be warranted. In particular the risks from use of ultra fine water droplets for longwall dust suppression may in fact have health risks that outweigh any potential benefits.

INTRODUCTION

Since the introduction of high productivity longwalls, particularly in thick seams that have been gas drained, respirable dust suppression has been a challenge not always met with success. Many mines still suffer dust sampling results that are close to or in excess of statutory limits, despite ongoing (and expensive) attempts to improve those results.

By way of relating some scientific reasoning with observations made over many years, the author proposes that there are reasons for the ongoing challenges that are not purely mining related, and further suggests that proposals to implement fine mist (atomising) water sprays as an improved dust mitigation strategy may be flawed. While significant research has been undertaken over many years in this area, the author contends that some further work in basic science may be warranted to justify the significant expense of some of the high level engineering dust control systems that are being considered for use.

THE TRIGGER FOR SCIENTIFIC ANALYSIS

People who regularly fly out of capital cities frequently notice the surface atmospheric dust level. This is the yellow brown, fine visible dust that hangs in the air to levels sometimes exceeding three thousand meters or to the base of cloud on other occasions.

Figure 1 shows typical visible dust layer over a city shown from an aircraft.

Figure 2 shows a wing leading edge of a five year old aircraft with no visible paint damage from dust erosion. It is fair to question as why this fine dust did not remove paint work from the leading edges of an aircraft, given that it would certainly be abrasive enough to do so. A research into aerodynamics and the boundary layer effects, a well known phenomenon in aircraft design since World War 2, has provided the answer. A boundary layer is created around an object passing through air by the surface resistance of the shape to the passage of the airflow. The boundary layer can be considered to be the

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**Figure 1 – Typical visible dust layer over a city shown from an aircraft
(Airliners.net website 2008)**



**Figure 2 – Leading edge of an aeroplane wing showing no paint wear
(Airliners.net website, 2008)**

transition from still air at the surface of the object, to the area of full air flow where the surface resistance has no further effect, as shown in Figure 3 (John Glenn Research Centre of NASA, undated). Boundary layers have been proven in wind tunnel testing, and many attempts to break down the layer have been made over the years, as it is believed that reducing the layer will increase lift on an airfoil while reducing drag, leading to improved aircraft performance. Some success has been recorded but no quantum leap forward to date.



Boundary Layer

Glenn
Research
Center

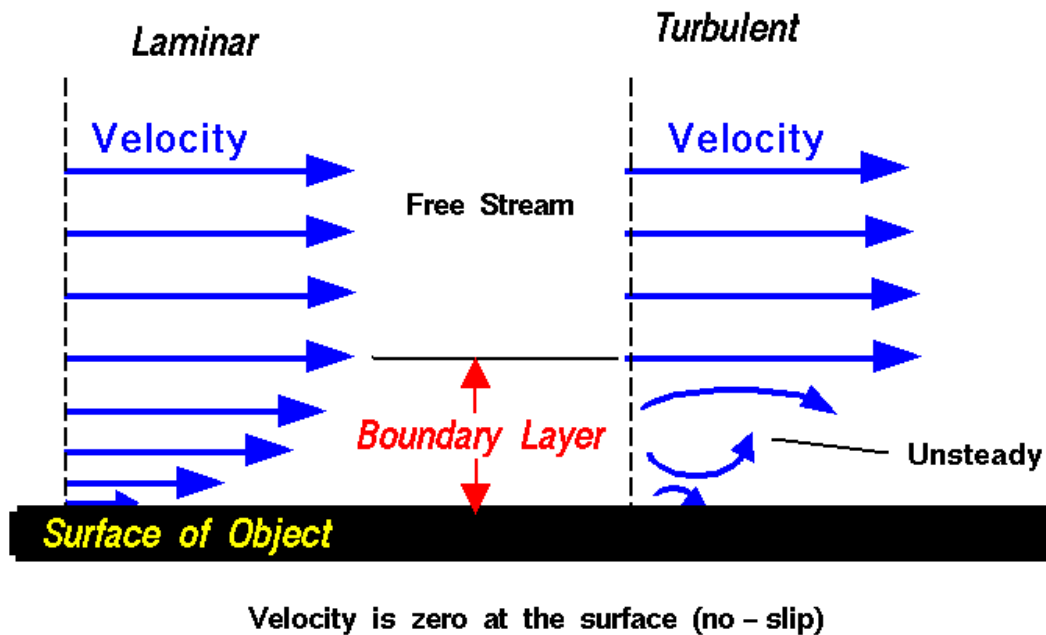


Figure 3 – The boundary layer above an object's surface in airflow (NASA)

It is the author's belief that a boundary layer exists around dust particles in the airflow along longwall faces, and the net effect of this layer is to make collisions between dust particles and water droplets, essential for suppression of dust, all but practically impossible where the air flow is laminar and not turbulent. The remainder of this paper addresses these matters. This paper does not offer practical working solutions as the base hypothesis has not been tested; however suggestions for further work are included. The effect of the layer is shown in Figure 4 (Ian McDonnell, 2008); where the smaller dust particles are carried past the water droplet with little chance of collision.

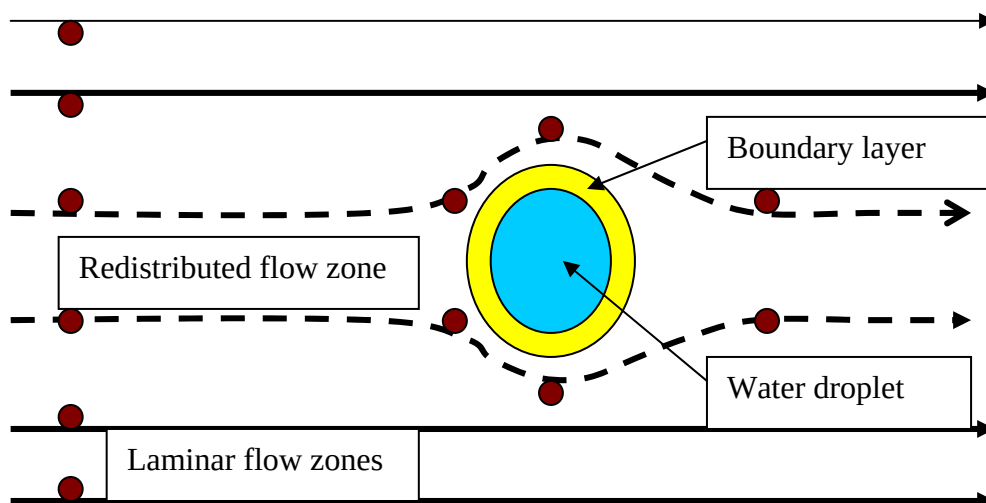


Figure 4 – The redistribution of laminar airflow around the boundary layer
(After "Beginners guide to Aeronautics", NASA website, undated)

LONGWALL DUST CONCEPTS

It is important to note the following particle size comparisons for dust and water droplets as applied to a longwall face.

- Visible dust is normally considered to be larger than 100 microns in size
- Respirable dust as defined in legislation is that fraction less than 10 microns, and is considered hazardous particularly when less than 4 microns when it can be deposited into the gas exchange regions of the lungs. It has a sub-division called "thoracic", being the particle range 4 to 10 micron, which is stated to be hazardous when deposited in the lung airways.
- Inhalable dust is defined as the fraction that is less than 100 microns, and is stated to be hazardous as it can deposit anywhere in the head airway region (from AS2985, AS3640 and "Personal Multi-fraction and Bio-aerosol sampling" by Weber Consulting Website.)
- Water mist and droplets as produced by typical longwall sprays is formed in the size range from 100 to 5000 microns (Spray data from "Engineer's Guide to Spray Technology", bulletin number 498 from Spraying Systems Co. Illinois, USA, undated)
- 1000 microns is 1 mm
- The full stop of 12 point Times New Roman print is about 250 microns.
- It should also be noted that for the common hollow cone spray used for dust suppression, and operated at 700 Pa (100 psi) at a flow of 3 litres per second (30 g.p.h.), an average droplet size of 1260 microns is produced. (Spray data from "Engineer's Guide to Spray Technology", bulletin number 498 from Spraying Systems Co. Illinois, USA, undated)
- To reduce this to an average size of 200 microns, getting near to dust particle size, an air atomising spray operating at 70 Pa (10 psi) and 0.0015 litres per second (0.02 g.p.h.) is required. (Spray data from "Engineer's Guide to Spray Technology", bulletin number 498 from Spraying Systems Co. Illinois, USA, undated)

MINING LEGISLATION REGARDING DUST

Both NSW and Queensland legislation sets limits of total dust content in the respirable range including limits for quartz. The legislation also requires management plans for control of airborne dust including regular sampling of respirable dust and quartz. Recent changes to NSW legislation also require sampling for inhalable dust. It is likely that tighter controls will be implemented over time, and particularly if mines register results around or over the statutory limits. There are two Australian Standards regarding dust:

1. AS2985-2004 "Workplace Atmospheres – Method for sampling and gravimetric determination of respirable dust"
2. AS3640 "Workplace atmospheres - Method for sampling and gravimetric determination of inhalable dust"

DUST SUPPRESSION BY WATER DROPLETS

Suppression dust by water droplets in a moving airstream requires impacts between the dust and droplets. By impacting, the dust adheres to the droplet, falling under gravity to floor level. Conventional logic suggests that a higher quantity of finer water droplets will suppress the dust better. This is not often backed up by actual data, but inconsistencies in data collection may hinder the true analysis.

Around the shearer drums, and in the outbye shearer clearer zone, turbulence is created in such magnitude so as to be a credible factor in creating these impacts. The maingate to face corner should also have a certain amount of turbulence that can be used with sprays to remove both incoming dust and dust created by the longwall from such sources as the shearer at the maingate, coal breakage on the chain and from the movement of the shields. The face to tailgate corner is similar to the maingate end in this respect.

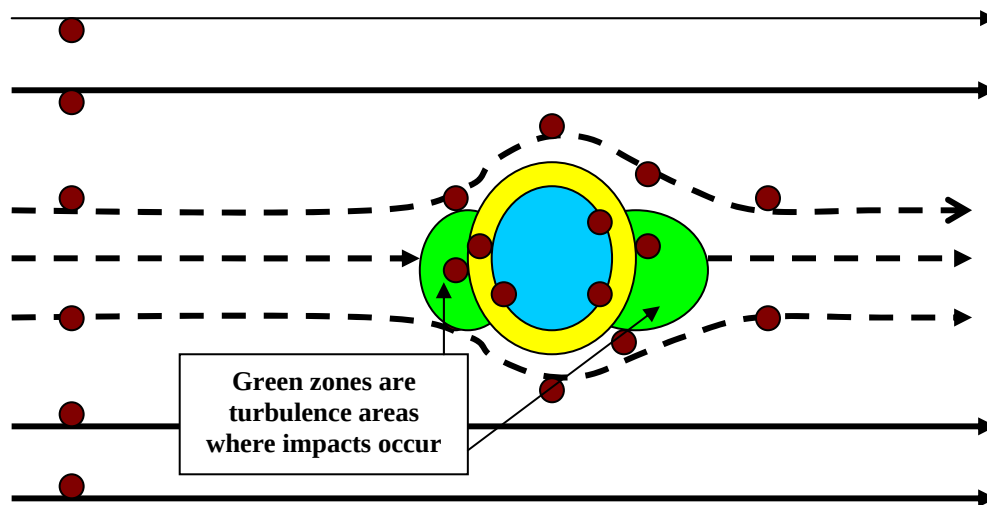


Figure 5 – The turbulent areas around a water drop where impacts may occur (After “Beginners guide to Aeronautics”, NASA website, undated)

From the previous redistribution diagram, Figure 5 shows the zones of turbulence that are known to occur when airflow passes over an airfoil. The rear turbulence zone is much stronger than the frontal one, but due to the airstream velocity effect, more impacts are likely at the front. This diagram only applies to laminar flow areas, where I believe a minimum of impacts occur. In a longwall environment, the large majority of impacts only happen in the extremely turbulent zones around the drums and clearers.

The balance of the face, that is from maingate to shearer, and from shearer to tailgate, and tailgate corner outbye are all zones of laminar airflow, and as such may rely more on luck for impacts rather than turbulence. In all probability, dust suppression in these areas could only occur if additional turbulence is introduced, for example by way of deflection curtains and spray systems.

Full mine testing has shown that a significant amount of respirable dust particles travel extensive distances out the return airways (Gillies and Wu, 2008), and only appear to fall out of the airflow when the airspeed drops considerably or it goes around sharp, higher resistance corners. This has been tested at several mines by Gillies, and Wu, (2008).

USE OF WATER SURFACE TENSION AND STATIC CHARGE MODIFIERS

Surface tension modifiers are designed to allow the droplets to take other shapes than the pure sphere, which under equilibrium conditions is the lowest energy state. In a laminar airflow, the effect will be to elongate the droplet while reducing the frontal surface area. The amount of elongation and consequent decrease in frontal surface area will depend on the surfactant properties of the additive, its strength and the velocity of the airflow. This will certainly result in much smaller turbulent zones in front and behind the droplets, making random collisions between dust and water even more remote, as shown in figure 6.

Any beneficial effects of surfactant will be felt more in the highest turbulence areas, where the airflow effect is less than in laminar flow areas.

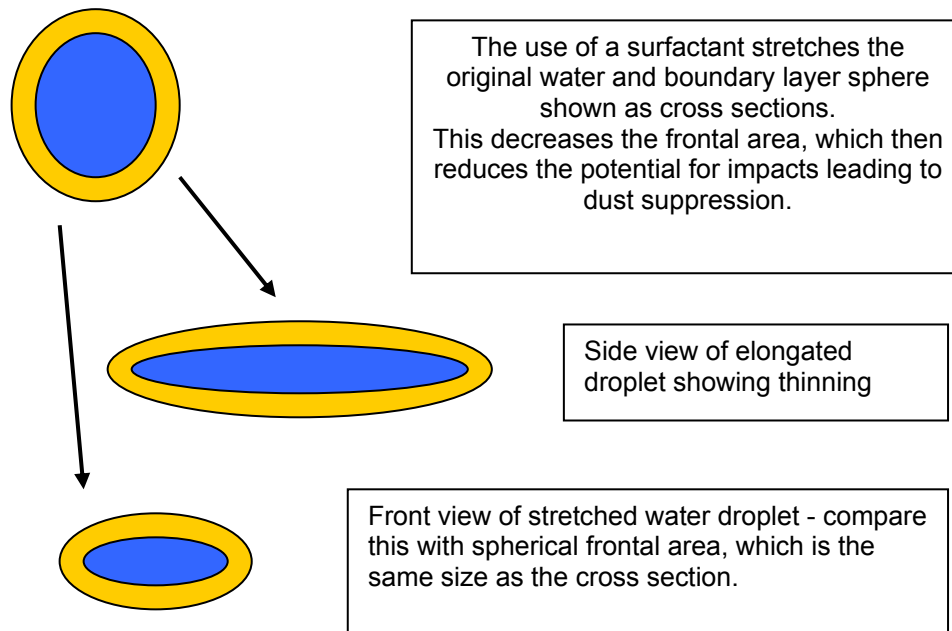


Figure 6 – The effect of surfactant on a water droplet in airflow

Water and dust particles in airflow will be affected by static electricity charge pick up. The intensity and polarity of the charge will vary depending on factors such as dust material type and size range (rubbing surface), water purity including acidity or alkalinity, water droplet size, velocity of airflow and effect of electromagnetic radiation from cables and motors. The rubbing surface is probably the biggest variable giving a likely result that the charge on small dust particles is quite low compared to the much larger water particles. The net effect is possibly not able to be accurately modelled, and so selection of a suitable static charge modifier may be difficult. It would also at this time be difficult to predict whether or not smaller water droplets would assist or hamper electrostatic attraction.

The object of a static charge modifier added to the water (be it anionic or cationic) is to impart opposite change in the water to that of the dust, causing electrostatic attraction and hence collision / capture. The author believes that this process has a long way to go before it can be demonstrated as successful.

Major fine dust producers such as coal fired power stations frequently use electrostatic dust precipitators, which use very high voltages across metallic plates to polarise and attract the dust. There is no possibility of using this technology underground.

EFFECT OF WATER AND GAS DRAINAGE FROM COAL ON DUST PROPERTIES

Gas drainage from coal whether methane or carbon dioxide is usually accompanied by large volume of water, which then allows shrinkage and drying out of the coal matrix. This directly contributes to the dust concentration when the coal is mined, but may have other outcomes dependant on the actual make up of the coal. Personal observations over many years have noted that some types of coal are very difficult to wet after the drainage process.

It has been suggested in discussion between coal chemists and geologists that this may be caused by a very fine "oily" film forming on the coal particles after drainage. While this is anecdotal only, it certainly could explain why added water sprays even around the shearers drums and clearers do not get the desired dust suppression effect.

DUST MEASUREMENT

The majority of dust sampling to date has been done with cyclone separation and collection of the sized particles for weighing, generally over the period of a full shift. This gives an accurate figure for the total dust exposure for the period sampled, but cannot be related to any actual longwall operation

such as position of shearer on face. Further, standard sampling does not always accurately reflect the source, quantity and timing of respirable dust entering the longwall from outbye. This further detracts from the value of measurements done to test control measures.

The author believes that for most operations, there is a need to use the newer real-time dust sampling tools available combined with a comprehensive test program that relates mine external as well as longwall dust sources to the longwall operations. For example respirable and inhalable dust reporting to the longwall may enter the mine airstream from the surface, from belt systems and travelling roads and from machinery.

CREATING MORE DUST TO WATER DROPLET IMPACTS AND IMPROVING COLLECTION OF DUST

Table 1 shows concepts that are considered by the author to be potential ways to enhance suppression, and are given along with potential weaknesses. The list is not exhaustive.

Table 1 – Details of methods to improve the number of dust to water droplet impacts

Concept	Method	Weaknesses
Reduced size of water droplets	Air / water atomising sprays	Vastly decreased water flow rates Much larger number of sprays for motor cooling flow
Increased size of dust particles but less fracture of the coal and rock	Modified cutting picks Modified drums, lacing, flights Slower drum speeds Slower chain speeds Modified lump breaker	Reduced output of shearer Potential blockages from larger lump sizes
Increased turbulent zones	Airflow deflectors Air blowers	Damage to equipment from face spalling and impacting with the shearer and shields
Modified turbulent zones	Use of cowls to contain zone Shrouding around shearer and clearers	Damage to cowls and shrouds
Machine mounted dust scrubbers	Use of rotary air curtain drums, with or without extraction Use of shearer mounted scrubbers	Cost versus effectiveness
Antitropal versus homotropal air flow for shearer clearer areas. Maingate clearer is Antitropal whereas tailgate is homotropal, with reduced effect	Modification of design to address.	Size and power usage Unproven exercise would be very costly.

HEALTH AND SAFETY CONSIDERATIONS

The use of ultra fine water mist sprays for dust suppressions is not without concerns in the health and safety area. The amount of additional fine water vapour may have the effect of more rapidly blocking filters used in air helmets and filter masks. This may result in more frequent changes of disposable masks – not a bad thing – or the temptation not to wear them.

Early blockage of air helmets leads to discomfort for people using them, and again tempts them to remove the PPE.

A further consideration is that by reducing the droplet size to an inhalable or respirable size will allow some of the water to enter the breathing zones of the body. The effect of this is not documented, but will need some medical considerations.

CONCLUSIONS

From my experience, observations and limited research I have made the following (highly arguable) conclusions:

1. Results of fine dust measurements as a result of changes made in an attempt to enhance dust suppression on longwall faces (particularly thicker seams) have not always shown effectiveness of the changes.
2. Reasons for this variation may be because the testing is not suitably comprehensive or relatable to the changes due to shift duration aggregate tests versus real time tests.
3. There may be many other scientific reasons for lack of effectiveness of longwall dust suppression that could be based on fluid mechanics and aerodynamic principles.
4. The use of fine atomising air / water sprays may not be effective when trialed due to these other effects.
5. There is an opportunity for specialised research into some of these matters that could lead to major improvements.
6. There is a lot of work in risk management to be done, both from the safety and health and operational points of view.

ACKNOWLEDGMENT

This paper is the work of the author, and any referred material that may be subject to copyright is acknowledged. The views expressed in this paper are those of the author, and do not purport to have any relationship to any coal mining operation, operator or mining company.

REFERENCES AND OTHERS

1. Photos are from the website cited from "Airliners.net".
2. Boundary layer drawing is from the John Glenn Research Centre of NASA
3. Spray data from "Engineer's Guide to Spray Technology", bulletin number 498 from Spraying Systems Co. Illinois, USA
4. Various references including AS2985, AS3640 and "Personal Multi-fraction and Bio-aerosol sampling" by Weber Consulting (Website).
5. Gillies-Wu Mining Technology, 1 Swann Road Taringa 4068.

TIMES TO IGNITION ANALYSIS OF NEW SOUTH WALES

Joel Sargeant¹, Basil Beamish² and Duncan Chalmers¹

ABSTRACT: The 'times to ignition' theory has had limited use in the coal industry and virtually no use within Australia. The concept was originally applied to a range of Scottish coals to assess their spontaneous combustion propensity during transport in a 3x3m shipping hold. This paper presents results from an investigation into the application of the times to ignition (t_{ad}) concept to a range of New South Wales coals using data obtained from adiabatic oven self-heating tests. There is a strong association between t_{ad} and the R_{70} self-heating rate of a coal. The geographical location of the coal has a dramatic effect on the t_{ad} value as the initial coal temperature significantly changes the results by varying degrees.

INTRODUCTION

The coal mining industry will always be faced with the potential of a spontaneous combustion event, whether it is in an underground environment, in a coal stockpile, or during coal transport. There are many methods used for assessing the propensity of coal to spontaneously combust. One of the more common ones used in Australia is the adiabatic oven self-heating test (Humphreys, Rowlands and Cudmore, 1981; Beamish, Barakat and St George, 2000). Data obtained from this test is also amenable to reaction kinetic analysis, particularly once the coal temperature exceeds 70°C. Parameters obtained from this analysis can be used as input to a 'times to ignition' concept proposed by Jones (2000) for assessing the risk of spontaneous combustion.

Adiabatic self-heating data was selected from the University of Queensland's Spontaneous Combustion Testing Laboratory database for analysis. This data consisted of seven samples from four New South Wales coalfields with ash contents in the range of 8-12%.

This paper presents the results of times to ignition analysis of these samples and shows the significance reaction kinetics has on spontaneous combustion behaviour.

TIMES TO IGNITION BACKGROUND

The original times to ignition concept was derived by Boddington, Feng and Gray (1983), with the purpose of providing an analytical treatment for systems with distributed temperatures in which heat-transport is controlled by conduction. More recently, Jones (2000) used the concept in the calculation of coal transport ignition times. Equation (1) shows the expression used to calculate the times to ignition (t_{ad}):

$$t_{ad} = (RT_R^2/E) \times (c/QA) \times \exp\{E/RT_R\} \text{ (seconds)} \quad (1)$$

where the times to ignition, t_{ad} is in seconds (s) of a reactant of specification A (pre-exponential factor) and E (activation energy) at initial temperature T_R , and c is the specific heat, Q is the heat of reaction and R the Universal gas constant. The expression QA/c can be obtained directly from the test data.

TIMES TO IGNITION CALCULATIONS

Test data

Adiabatic oven self-heating test results were obtained on pulverised dry coal from a start temperature of 40°C. The test procedure used is adequately described by Beamish (2005). All data was stored in an Excel spreadsheet for further analysis.

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Times to ignition data analysis

Jones *et al.* (1996, 1998) used the Arrhenius equation for obtaining kinetic parameters from oxidation of Scottish bituminous coals. Under adiabatic conditions this equation can be expressed as:

$$\ln [dT/dt] = \ln (QA/c) - (E/RT) \quad (2)$$

A plot of $\ln [dT/dt]$ versus $1/T$ will produce a straight line of slope $-(E/R)$ and intercept of $\ln (QA/c)$. An example of this plot for one of the samples tested from the Gunnedah Coalfield is shown in Figure 1. Jones (2000) also showed that under adiabatic conditions a "times to ignition" value can be obtained from some initial temperature (T_R) if E/R (slope from Figure 1) and QA/c (y-intercept from Figure 1) are known for a particular sample and the values substituted into Equation (1).

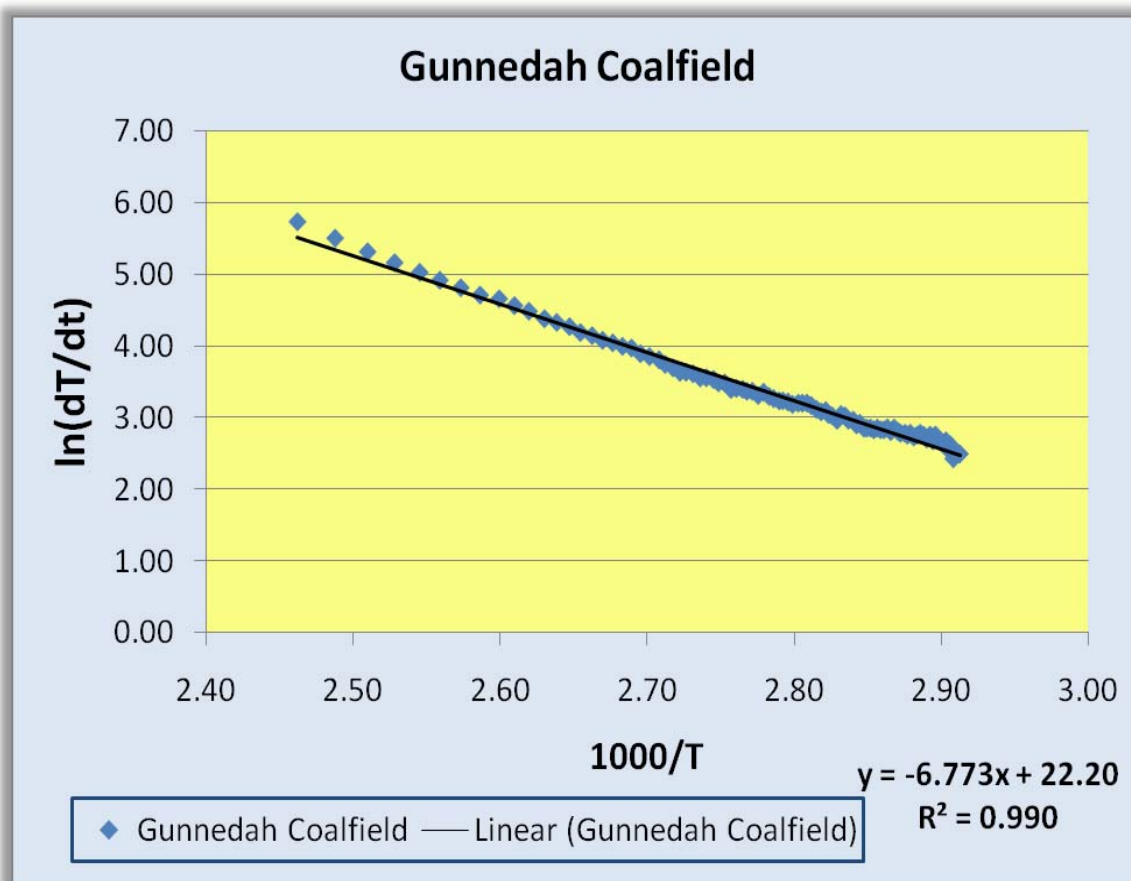


Figure 1 - Example of Arrhenius plot for adiabatic self-heating results from a sample of Gunnedah coal

RESULTS OF TIMES TO IGNITION ANALYSIS AND DISCUSSION

Variation in times to ignition

The range of t_{ad} values is shown in Table 1. The Southern Coalfield sample has a t_{ad} value of 57 days compared with 0.92 days for the Gunnedah Coalfield sample. It is interesting to note that the Southern Coalfield sample is a high rank coking coal, whereas the Gunnedah Coalfield sample is a lower rank steaming coal. Both coal rank (Beamish, 2005) and coal type (Beamish and Clarkson, 2006) have previously been linked to differences in spontaneous combustion propensity. It must also be remembered that these samples have been tested in a completely dry state and the influence of moisture in the coal on the kinetics of the oxidation reaction have not been taken into consideration. Further work is in progress on this effect as it is known to have a major impact in terms of delaying thermal runaway.

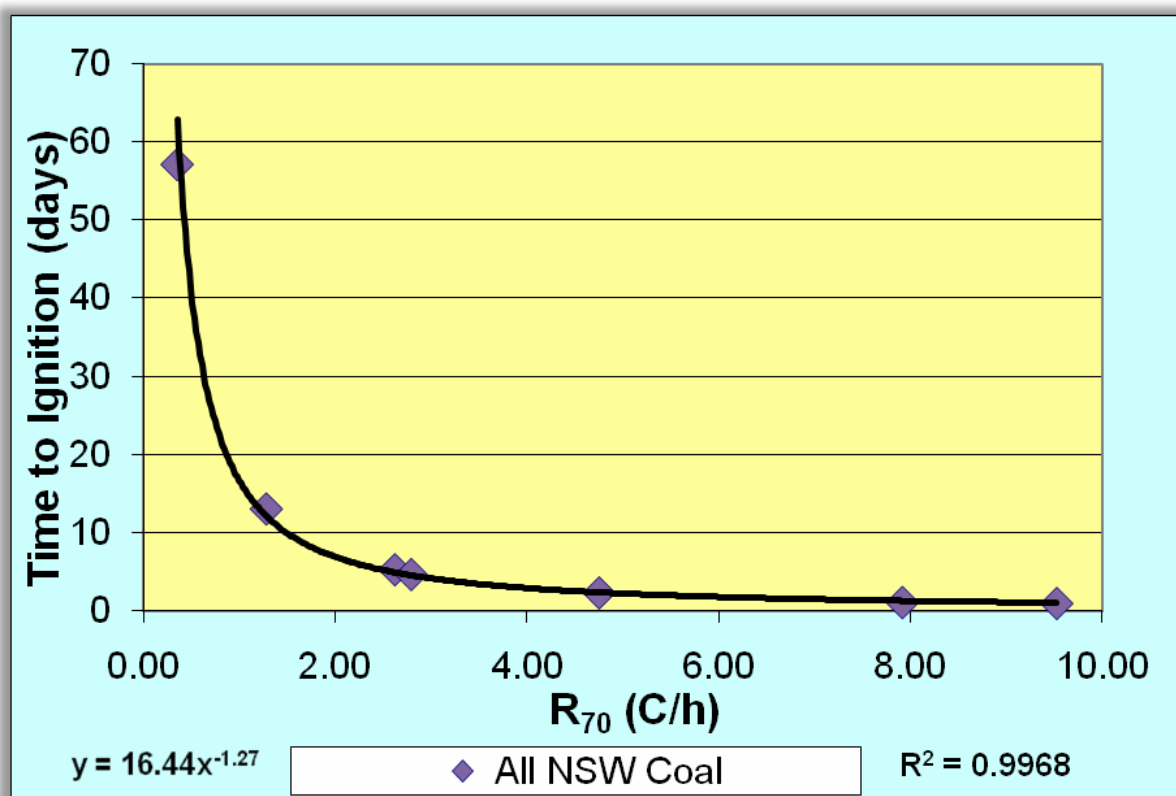


Figure 2 - relationship between t_{ad} and r_{70} self-heating rate

There is a strong non-linear relationship between t_{ad} and R_{70} self-heating rate (Figure 2). This is somewhat surprising as the t_{ad} calculations are based on the portion of the adiabatic self-heating curve above 70°C and the R_{70} value is obtained from the 40-70°C portion of the curve.

Table 2 - Calculated t_{ad} values by NSW coalfield

Location	T_{ad} (days)	R_{70}	Ash % db
Southern Coalfield	57.15	0.35	9.7
Hunter Coalfield 1	13.03	1.28	10.1
Newcastle Coalfield	5.24	2.62	9.3
Hunter Coalfield 2	2.27	4.75	10.9
Hunter Coalfield 3	1.03	7.91	11.8
Gunnedah Coalfield	0.92	9.52	8.1
Newcastle 1	4.63	2.79	11.6

Effect of start temperature on times to ignition

The t_{ad} start temperature has a large impact on the resultant time to ignition, which is shown in Figure 3. The value of 25°C used for NSW conditions was compared to 40°C, to replicate Queensland conditions and outline the difference a simple change in temperature has on t_{ad} . The Southern Coalfield sample t_{ad} value reduces from 57 days to approximately 17 days. It is clear from Figure 3 that a coal mined in New South Wales may not create a spontaneous combustion issue, but the same coal mined in Queensland would create a problem.

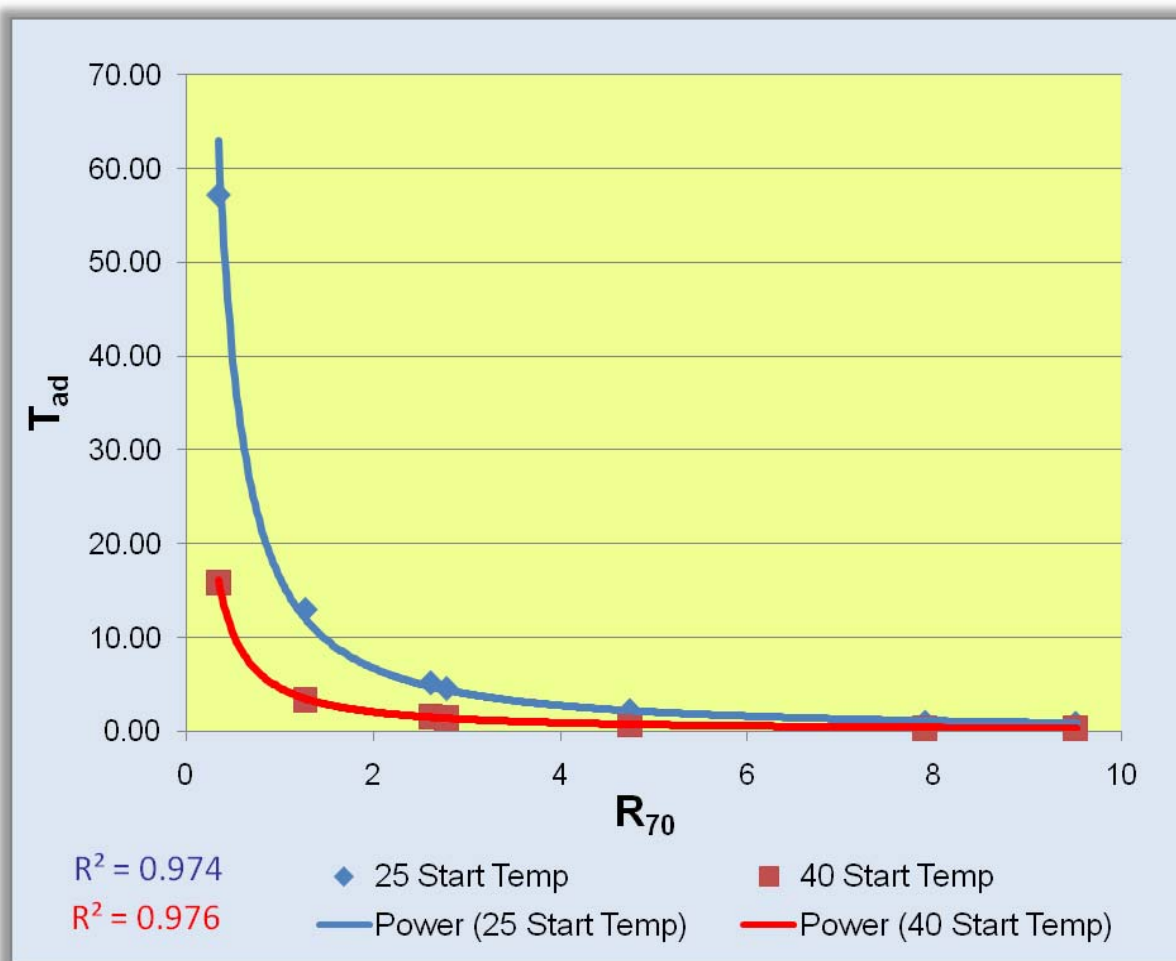


Figure 3 - Comparison of start temperature effect on t_{ad} values

CONCLUSIONS

The times to ignition (t_{ad}) concept has been applied to New South Wales coals using data obtained from adiabatic oven testing. Generally, high rank coking coals have the highest t_{ad} values and lower rank steaming coals have the lowest t_{ad} values. This is due to the significant difference in activation energy required for the oxidation reaction to take place in each of these coal types. There is a strong non-linear relationship between the times to ignition and R_{70} self-heating rate obtained from the same test data. More importantly the time to ignition analysis emphasises the importance of the coal start temperature on the time to ignition due to the exponential effect of the Arrhenius kinetics of the oxidation reaction, whereby as the temperature increases the rate of reaction increases. Therefore it is crucial that mines obtain coal temperature data to be input to any spontaneous combustion propensity assessment.

ACKNOWLEDGEMENTS

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ASSESSMENT OF AN UNDERGROUND COAL MINE FIRE: A CASE STUDY FROM ZONGULDAK, TURKEY

Alaaddin Cakir¹ and Kemal Baris¹

ABSTRACT: This paper aims to evaluate an underground coal-mine fire detected on November, 11th 2007 in Gelik Mine of Karadon Colliery of Turkish Hardcoal Enterprise (TTK), Zonguldak, Turkey. Several techniques were employed by TTK to fight the fire including sealing, filling the mine with water and pumping nitrogen. The mine atmosphere was continuously monitored and gas samples were collected for analysis using a gas chromatograph. In this study, following a description of the fire fighting efforts, the interpretations of well-known fire indices applied to the different stages of the incident were given and attempts were made to compare and to test the reliability of these indices.

INTRODUCTION

Fire in coal mines is a serious problem in Zonguldak Basin and worldwide. Spontaneous combustion is the main reason in most of the mine fires detected within the mines of TTK. Mine fires may lead to loss of life, stopping production, equipment loss and mine closure.

The mine fire to be investigated was detected in Gelik Mine of Karadon Colliery. There were six mine fires in Karadon Colliery due to spontaneous combustion between 1990 and 2000. In advancing longwall mining, coal left in the goaf (gob) and other geological conditions are considered as the main reasons for these fires. Other factors include air passing through the goaf, coal left at the roof or floor of the face, where seam thickness is high and air leakages due to the roof collapse.

This paper aims to evaluate the mine fire which took place at Gelik Mine-Karadon Colliery. Several gas indices were utilized to assess the underground atmospheric conditions as well as the continuity of the fire.

GENERAL INFORMATION ABOUT BASIN, TTK AND GELIK MINE

Zonguldak hardcoal basin in Turkey is the only basin with hardcoal deposits. Mining activities in the basin started in 1848 and several national and international companies operated various coal mines. In 1938, the basin was acquired by Eregli Coal Enterprise (EKİ) which operated these mines until 1983, when the Turkish Hardcoal Enterprise (TTK) was founded. Since then the mines in the basin have been operated by TTK.

TTK is a government organisation and is the dominant coal producer in the basin. At June 2008, TTK employed 9,857 workers and 1,940 officers, including engineers. Total coal production by TTK in 2007 was 2,423,719 tonnes run-of-mine coal, 1,675,372 of which was saleable. There are five main collieries operated by the organisation, namely, Armutcuk, Kozlu, Uzulmez, Karadon and Amasra collieries. Reserves of these collieries are presented in Table 1.

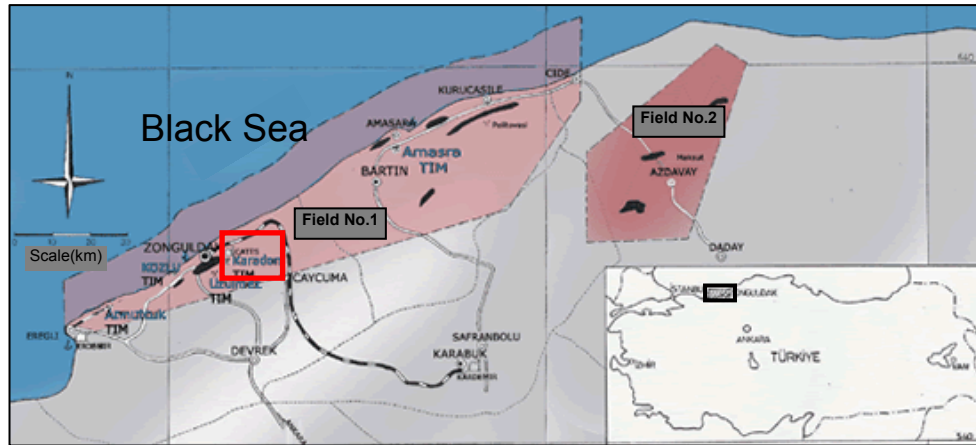
Gelik mine is one of the two hardcoal mines within Karadon Colliery which is located 12km east of Zonguldak and covers an area of 30 km² (Figure 1). Gelik mine employs a total of 1,228 persons; 1,210 are mine workers and 18 are administrative staff, including engineers. The estimated total reserve of the mine is estimated at 175,127,801 tonnes. Coal production is by conventional longwall mining using pick and shovel with timber supports and chain conveyors. There were ten longwall faces, located at -150/-260, -260/-360, -360/-430 and -430/-470 levels. Daily production of the mine was 2,200 tonnes before the fire took place. The information about the active longwalls is presented in Table 2.

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Table 1 - Reserves of five main collieries of TTK (TTK Annual Report, 2007)

Colliery	Reserves (tonnes)				
	Available	Proved	Probable	Possible	Total
Armutcuk	1,168,000	4,594,324	11,089,144	5,885,637	23,037,105
Amasra	233,700	171,873,195	115,052,000	121,535,000	408,693,895
Kozlu	3,353,714	68,729,821	40,539,000	47,975,000	160,597,535
Uzulmez	2,178,060	137,181,373	94,342,000	74,020,000	307,721,433
Karadon	2,924,000	136,905,375	159,162,000	117,034,000	416,025,375
Gelik Mine*	1,067,515	54,177,286	65,983,000	53,900,000	175,127,801
Total	10,157,474	519,284,088	420,184,144	366,449,637	1,316,075,343

* Included in Karadon Colliery reserves.

**Figure 1 - Location of Zonguldak Basin including the borders of TTK (TTK, 2007)**

THE INCIDENT- INITIATION OF FIRE AND FIRST ACTIONS

The first indication of the fire underground was when an underground worker reported smoke and a smoky smell at the end of No.41222 drift (-150/-260) and a smoke concentration at No.41303 drift at 17:00 on Sunday, 11th November, 2007. Afterwards, an investigation team went underground and confirmed a blazing fire on No.41300 drift (-260) at 21:00.

The reason for the initiation of the fire was not known at first. It was thought to be the result of either a spontaneous combustion event or an electrical contact. Though the reason was not exactly known, the mine management's response was to fight the fire by every means possible, which included sealing-off, pumping nitrogen and filling the area with water. This response was likely based on the history and previous experience in the basin, i.e. disasters and explosion risks due to high CH₄ content of the mines in the basin causing the management to panic. However, the mine inspection undertaken after discharging the water and opening the seals showed that the fire had been initiated by an electrical contact.

After the detection of the fire, two teams started to fight the fire with water at two different points at 00:30 on 12th November, 2007. The first team managed to advance to the entrance of No.41306 drift (-260) by extinguishing the fire. However, the team encountered a roof collapse and observed the fire continuing behind the fall at both No.41300 (-260) and No.41306 drifts(-260). Since it was not possible to pass through the collapsed area the first team stopped the fire fighting at 04:00 on 13th November, 2007.

While these events had been taking place the second team working at No.41303 drift (-260) was advancing up to the place where there was once a transformer station, 20 m away from No.41307 drift (-260). However, there was also a roof collapse in this area. Due to intense smoke and CO concentration, up to 2,000 ppm it became impossible to advance further, and a decision was made to pull out of the drift at 06:00 on 13th November, 2007 and seal the mine. The timeline showing the dates of the actions during the fire is presented in Figure 2.

Table 2 - Active longwalls in Gelik mine (TTK, 2008)

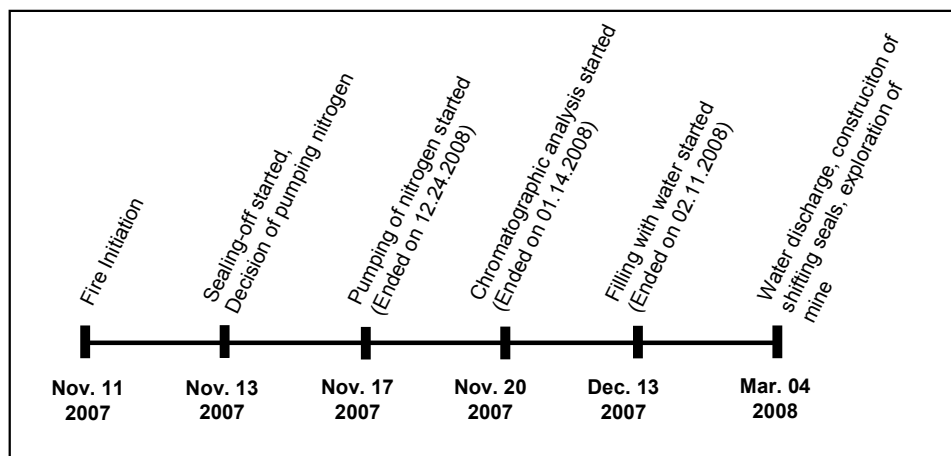
Name	Levels and Drift Codes	Production Method ³	Seam Thickness (m)	Seam Inclination (°)	Average Daily Production ⁴ (tonne/day)
Acılık ¹	-150 / 41230 -260 / 41307	Advancing Longwall	2.5 - 3.0	25	250 - 300
Sulu ¹	-150 / 41230 -260 / 41307	Retreating Longwall	2.0 - 2.5	35	150 - 200
Sulu ¹	-150 / 41228 -260 / 41306	Retreating Longwall	2.5 - 3.0	35	250 - 300
Cay ¹	-150 / 41306 -260 / 41406	Advancing Longwall	1.0 - 1.5	30	100 - 150
Acilik ¹	-260 / 41307 -360 / 41407	Advancing Longwall	2.0 - 2.5	35	150 - 200
Acilik ²	-150 / 41227 -260 / 41315	Retreating Longwall	2.5 - 3.0	25	250 - 300
Acilik	-260 / 41315 -360 / 41415	Retreating Longwall	2.5 - 3.0	35	250 - 300
Ozkan	-360 / 41409 -430 / 41509	Retreating Longwall	1.5 - 2.0	20	200 - 250
Nasufoglu	-360 / 41409 -430 / 41509	Advancing Longwall	2.0 - 2.5	20	150 - 200
Sulu	-430 / 41509a -470 / 41409b	Retreating Longwall	2.5 - 3.0	15	200 - 250

1 : Constrained in fire zone.

2 : Transport roadway is closed due to the fire.

3 : Back of the faces is caved.

4 : Changes according to the number of timbers.

**Figure 2 - Timeline of events in Gelik mine fire**

The sealing-off operation followed by pumping of nitrogen

The decision was made to erect the first seal at No. 41300 drift (-260) at approximately 11.00 am on Tuesday, 13th November, 2007. A further 19 seals were erected during the period to 05th January, 2008. The location and the details of the 20 seals are illustrated in Figure 3.

The connection of the fire area to the ventilation network was cut and it was thought that no fresh air was able to enter the area following the construction of the first seven seals erected on 13-14th November, 2007. However, on 14th November, gas measurements showed that the seals were not effective, possibly because of leakage (CO: 2500ppm, CH₄:3.6%, CO₂:3.63 and O₂: 14%). Mine management then decided to install additional seals and pump nitrogen into the area in an attempt to control the fire as quickly as possible.

Liquid nitrogen was obtained from nearby nitrogen production facilities and transported in tankers to Cumhuriyet shaft which was operated between the levels +126 and -668. Construction of a nitrogen transportation line was completed on 17th November, 2007 and nitrogen injection to the fire area from No.1 seal commenced by 23.45. Pumping continued for 37 days until 16.30 on 24th December, 2007. During that period, apart from small breaks, there was a 2.5 day delay in pumping nitrogen caused by severe winter conditions which hampered the transport of nitrogen to the mine site. A total of 1056.9 tonnes of nitrogen was pumped into the mine which was transported by 47 tankers whose capacities ranged from 13.1 to 30.2 tonnes.

Filling the fire zone with water

The mine management held a meeting to discuss the fire situation on 13th December, 2007. Although 16 seals had been constructed (3 in -55, 4 in -150, 5 in -260 and 4 in -360 levels, as shown in Figure 3) gas samples taken from the mine since 20th November, 2007 indicated that oxygen concentration were not decreasing (CO: 34ppm, CH₄: 4.3%, CO₂: 2.36%, O₂: 12.62), prompting the mine management to conclude the possibility of air leakage to the fire area. Air leakage was thought to have come from a number of old and abandoned workings close to the fire zone, and affected most of the -55, -160, -260 and -360 levels. Therefore, the mine management decided to fill the mine with water to prevent fresh air entrance to the mine and thus extinguish the fire.

The water filling operation commenced at 20.00 on 13th December, 2007 from the No.1 seal constructed on No.41300 drift (-260) by a 100mm-diameter pipe and No.10 seal constructed on No.41303 drift (-260) by a 50 mm-diameter pipe. The piped water was provided by pumps from the water pools located on the on-setting station which was 220m away from No.1 seal and on the bottom of No.41222 (-150/-260) incline.

Filling operations stopped on 26th December, 2007 at 16.00-24.00 shift. However, analysis of gas samples which were continuously being taken from the fire area indicated that there was an increase in CO concentration. Therefore, water filling operation was resumed on 27th December, 2007 until 3rd January, 2008. At this date no CO was detected in the mine atmosphere so the water filling operation was stopped. On 9th January, 2008 the seals were checked and it was determined that No.4 seal on No. 41303 drift (-260) was full of water but there was no water behind No.1 seal on No.41300 drift (-260). Meanwhile, gas analysis showed that CO concentration had exhibited no change since 3rd December, 2007 which was about 0-1 ppm. Thereupon, the water in No.4 seal was started to be discharged. After this date mine atmosphere was continuously monitored. On 14th January, 2008 gas analysis showed that CO concentration in the mine increased from 2 ppm to 3220 ppm in 17 hours. Thus, mine management realized that there was a heating in the mine again and decided to stop discharging the water from the seal and pump more water into No.1 and No.10 seal. Water pumping continued for 28 days and stopped on 11th February, 2008.

During the 28 day period, actions which were taken by the mine management included:

- Continuously monitoring mine atmosphere by analysing gas samples using a gas chromatograph, 5 times a day
- All seals were checked for leakage and any leaking seals were recoated
- Water level behind the seals was monitored. Leaking seals were recoated
- In the other parts of the mine, old and abandoned panels in particular, air leaks were investigated and caissons were applied to regions where there was air leakage.

Water discharge, construction of shifting seals and exploration of mine

Water discharge commenced on 4th March, 2008 from No.1 seal. Then, No. 4 and No.13 seals were opened and these places were ventilated on 5th March and 6th March, 2008, respectively. Meanwhile, No.1 seal had to be stopped since discharged water obstructed the transportation through -360 trolley drift. Having taken all the necessary precautions No.1 shifting seal was constructed on No.41228 drift and closed.

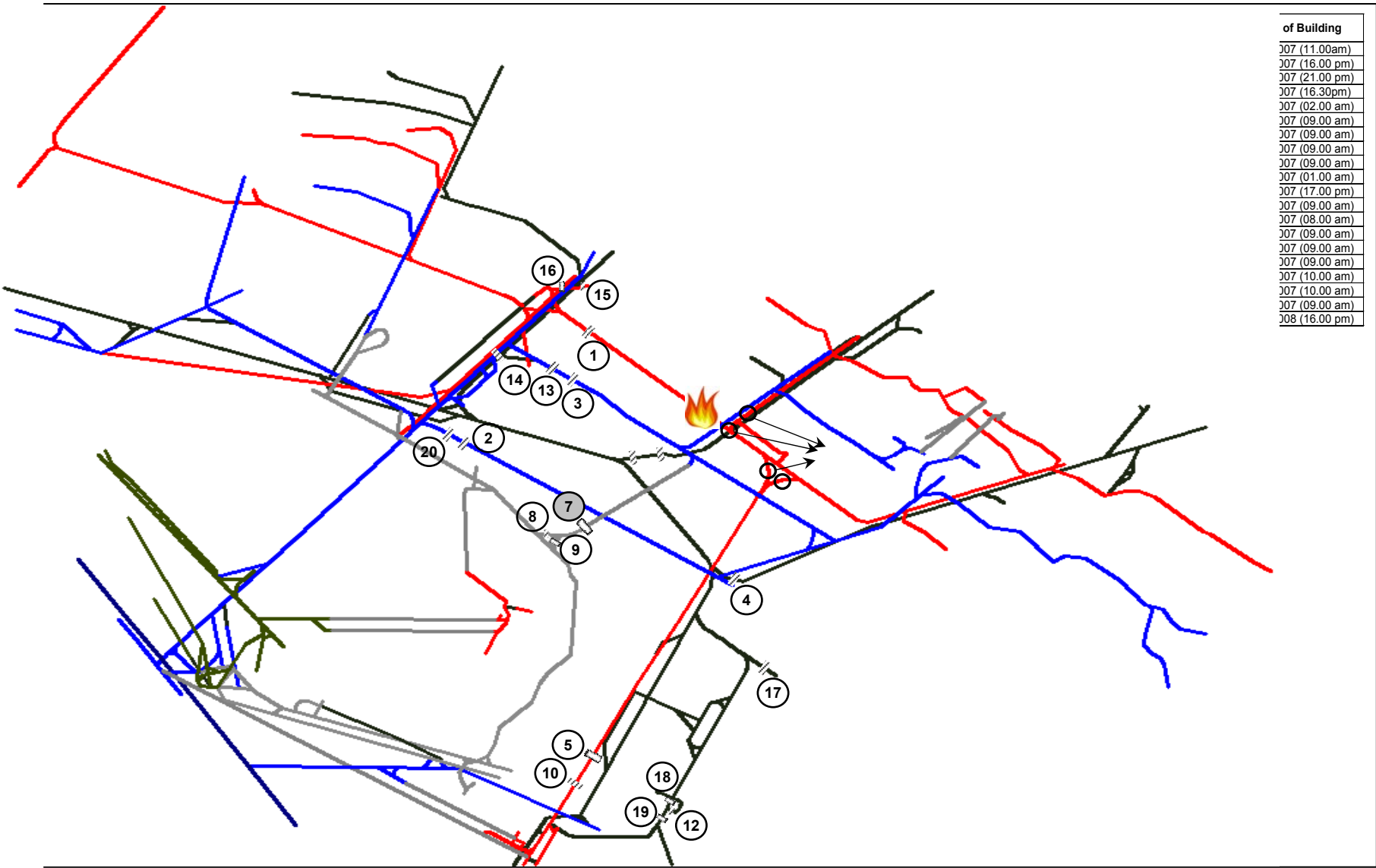


Figure 3 - Mine plan showing the location of the seals

No. 20 and No.2 seals were opened and these places were ventilated on 7th March, 2008. No. 2 shifting seal was constructed on No.41230 drift and closed. Finally, on 9th March, 2008, No.10 and No.5 seals and on 10th March, 2008 No.1 seal were opened and ventilated.

Opening of seals, construction of shifting seals and water discharge operations were performed gradually and in coordination to prevent air from entering the panels which possess spontaneous combustion risks.

An inspection of the mine was undertaken after discharging the water and opening of the seals. During the inspection it was found that:

- There had been two large roof falls in the junction of No.41300, No.41303 and No.41307 drifts. One of them was at the cut-through which connected No.41300 and No.41303 drifts and the other was between No.41300 and No.41307 drifts.
- There were two more collapses, with one at the junction connecting No.41300 and No.41306 drifts and the other 20 m Northeast of No.41306 drift entry.
- The fire had taken place in the region around 20m northwest, 200m northeast and 200m southeast of No.41306 drift. All timber supports, conveyor belts and cables were burnt out.
- No fire indication was observed in roadways connecting No.41303 drift to No.41300 and No.41307 drifts.

MONITORING OF MINE ATMOSPHERE OF GELIK MINE FIRE

Mine atmosphere was continuously monitored during the fire. The first gas samples were taken on 20th November, 2007 from No.7 seal which was the only seal where sampling was possible at that time. Gas sampling continued for 55 days until 14th January, 2008. During this period a total of 435 samples were taken. 118 samples were analysed by gas chromatograph at the Department of Safety and Education of TTK. The rest of the samples were roughly analysed in the laboratories of Gelik Mine. All data obtained from the gas chromatograph analysis were taken into account to validate the scientific approach. Figure 4 shows the trend in the concentrations of CO, CO₂, CH₄ and O₂ during the incident.

Fire gas indices applied to Gelik Mine fire

The mine fire was monitored after sealing to check whether it was being controlled or progressing. Status of mine fires is usually assessed by different indices or ratios, such as production of CO, Graham's ratio, C/H ratio, CO/CO₂ ratio, Willet ratio, Trickett ratio, etc. together with the estimation of temperature of fire area. However in this case, none of the indices or ratios used was capable of giving a precise and certain picture of the status of fire within the sealed area. Therefore, several different indices/ratios should be used together to achieve more precise results. The indices and ratios generally used in Turkish coal mines are briefly discussed below.

Oxygen Consumption

The flaming combustion in a fire is expected to cease when the oxygen concentration is below 12.4%. However, fire can be sustained for a long time even at 1-2% oxygen in the atmosphere. Successful extinguishing of a mine fire can occur if no re-ignition occurs with the introduction of sufficient amount of oxygen to fire zone. However, in most cases as in this present case study, it was extremely difficult to get an accurate oxygen consumption rate since there was some air leakage into the sealed area.

Emission of CO and Graham's ratio

CO is considered to be an effective way of detecting the heat or a fire in a mine. With modern instruments and monitoring techniques it is possible to measure even minute traces of CO emission. If correctly interpreted, the formation rate of CO along with its ratio to the consumption of oxygen, i.e. CO/O₂ deficiency ratio, is a useful guide to assess the extent and course of fire.

Graham (1920) offered a method of calculating the degree of heating by comparing the formation rate of CO or CO₂ with that of the O₂ consumed. He observed the increase in oxygen adsorption in coal with increased temperature. This is known as Graham's ratio whose formula is given by Equation 1 and is still a widely used tool to detect and assess a mine fire. If the value of this ratio is around one it indicates a risky situation and if it is about three then it may indicate that the heating may turn into an extensive fire. In the case of an extensive fire the ratio gives a value higher than seven.

$$GR = \frac{CO}{0.265N_2 - O_2} \times 100 \quad (1)$$

Figure 5 and 6 presents the graphical illustration of the variation in O₂ deficiency and Graham's ratio for Gelik mine fire.

Willet's Ratio

Willet (1952) proposed a ratio based on the analysis of samples taken behind sealed off areas in some coal seams in which the CO produced by oxidation did not disappear at all, or disappeared at a very slow rate with the progressive extinction of fire. He used the ratio below to assess the magnitude and extent of the fire along with the analysis of CO. The ratio shows slow decreases during extensive fires.

$$WR = \frac{CO_{produced}}{Black\ damp + Combustible\ gas} \% \quad (2)$$

Variation in Willet's ratio values for Gelik mine fire is presented in Figure 7.

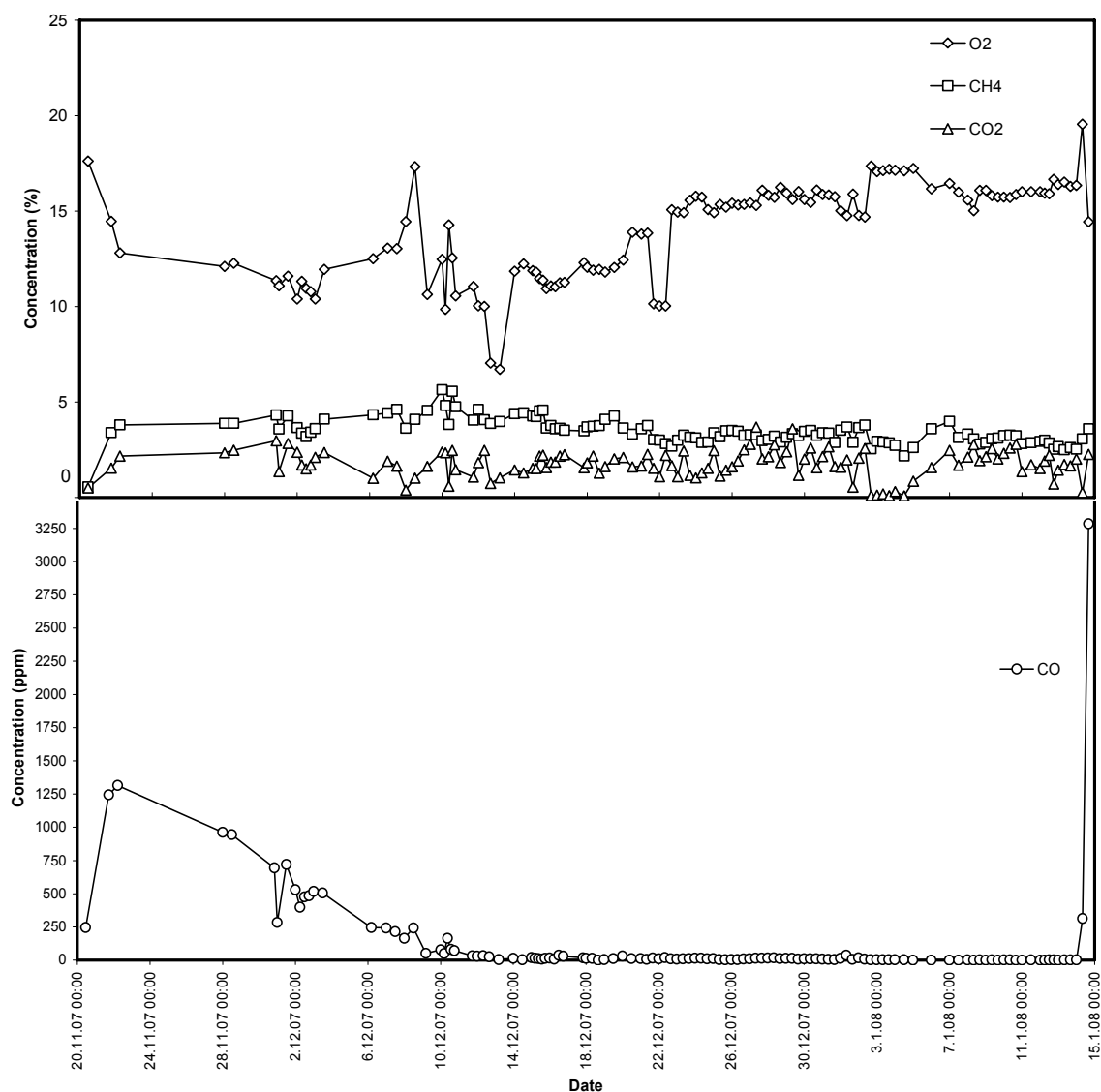


Figure 4b - Variation in the concentrations of CO, CO₂, O₂ and CH₄

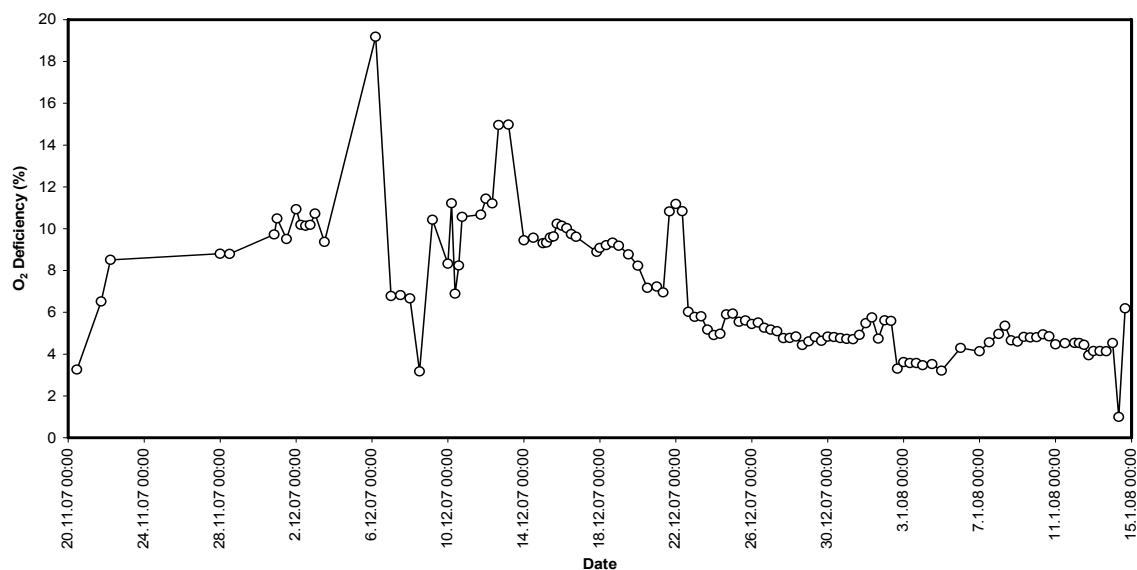


Figure 5 - The variation in O₂ deficiency values for Gelik mine fire incident

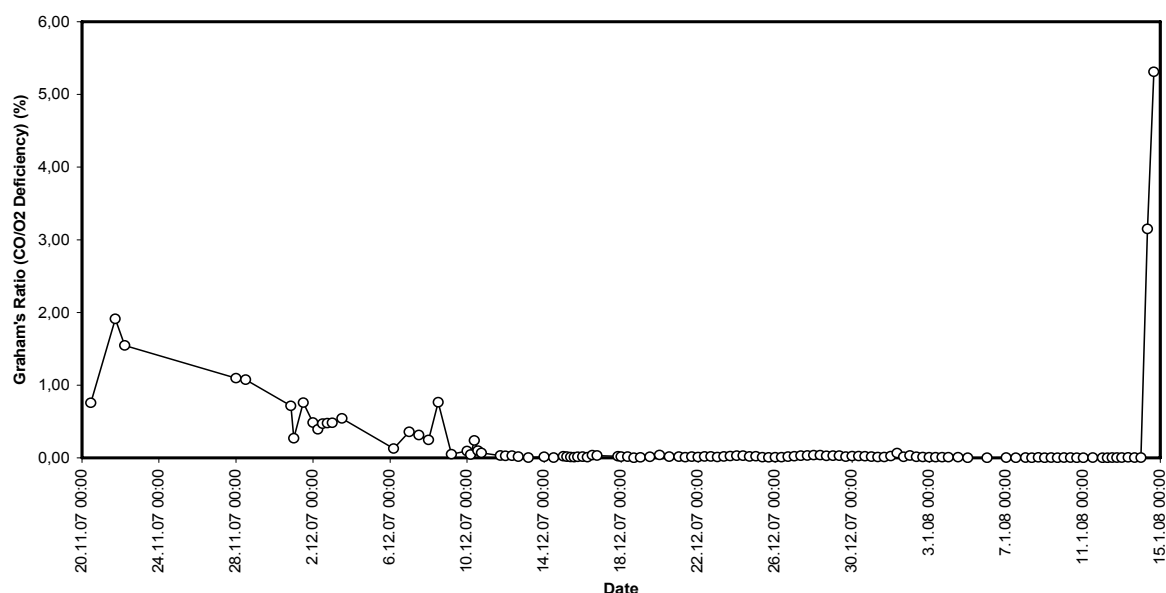


Figure 6 - The variation in Graham's ratio for Gelik mine fire

C/H Ratio

Ghosh and Banerjee (1967) introduced C/H ratio to assess the intensive character of a fire. Assuming coal to be a fuel of the type $C_xH_yO_z$ the following relationship was established:

$$C/H = \frac{6(CO_2 + CO + CH_4 + 2C_2H_4)}{2(0.265N_2 - O_2 - CO_2 + C_2H_4 + CH_4) + H_2 - CO} \quad (3)$$

By combining the above C/H ratio and oxygen consumption rate Ghosh and Banerjee (1967) offered a method to determine and characterise the extent and intensity of a fire in a sealed area (Table 3). Ghosh et al.(1980) also reported a comparison between Graham's Ratio (CO/O₂ deficiency) and C/H ratio of the fire gases in Jharia field (Table 4). The application of C/H ratio used to assess Gelik mine fire is given in Figure 8.

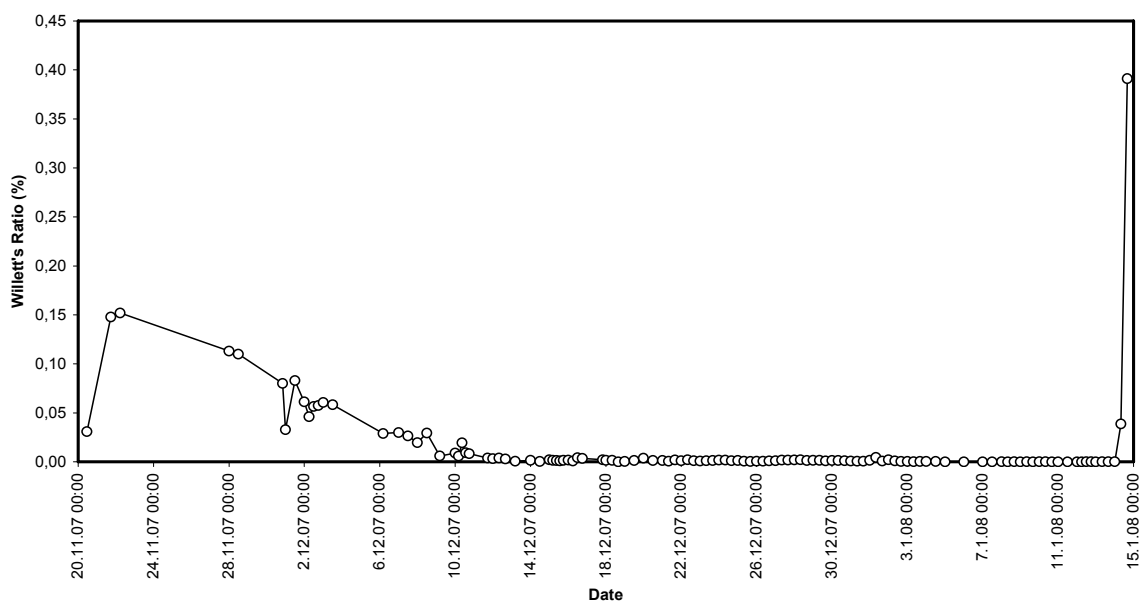


Figure 7 - Variation in Willett's ratio for Gelik mine fire

Table 3 - Application of C/H ratio in the assessment of an underground mine fire

C/H values, from analytical data of fire areas	Rate of oxygen consumption as observed from periodic analysis	Remarks on the nature of fire
1. High and very near to that of coal i.e. nearly 20	(a) Fast	Blazing and extensive burning of coal
	(b) Slow	Blazing but localised burning of coal
2. Appreciably higher than that of coal i.e. more than 20	(a) Fast	Blazing and extensive fire associated with burning of props etc.
	(b) Slow	Blazing fire associated with burning of props etc., but a localised one
3. Appreciably lower than that of coal i.e. much below 10	(a) Fast	Superficial fire but covering an extensive area
	(b) Slow	Superficial fire and a localised one

Table 4 - Comparison between CO/O₂ deficiency and C/H ratio in Jharia field (Ghosh et al., 1980)

Description of fire areas	CO/O ₂ Deficiency	C/H Ratio
Fire of recent origin, slight heating noticed, area kept sealed off	0.20-0.90	0.25-0.75
Heating in an advanced stage, area kept sealed off	1.75-3.00	1.01-1.45
An old abandoned long-standing fire, hot fumes observed coming out from overhead surface cracks	0.22-1.16	20 and above

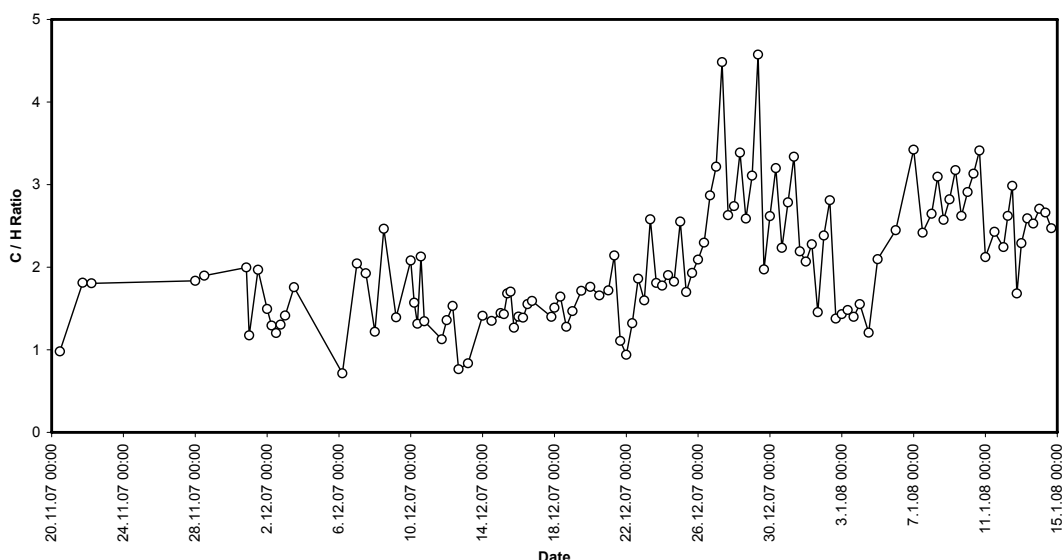


Figure 8 - The variation in C/H ratio for Gelik mine fire

Jones-Trickett Ratio

The Jones-Trickett ratio resulted from numerous examinations of the gases formed as by-products of colliery explosions and mine fires. This ratio can be used to indicate the type of explosion that has occurred in a mine and the level of activity or passivity of a fire behind seals. It was originally developed to help researchers differentiate between gas or coal dust explosions. It has been adopted to mine fire situations. Jones and Trickett (1954) suggested that analysis of the gases in a sealed area may indicate whether methane or coal dust had been involved in an explosion. They treated the combustion of methane and of coal theoretically, in each case deriving a relationship between the amount of oxygen used in the reaction and the amounts of carbon dioxide, carbon monoxide, and hydrogen produced. All these quantities can be determined when a sample of gas is analysed. The relationship for methane is different from that for coal due to the difference in chemical composition; hence, observation of the relationship makes it theoretically possible for the products of a methane explosion to be distinguished from those of a coal dust explosion. The ratio is expressed as:

$$TR = \frac{CO_2 + (3/4)CO - (1/4)H_2}{0.2647N_2 - O_2} \quad (4)$$

The ratio varies with the type of fire, depending on the fuel. If the ratio is less than 0.4, no combustion exists. When the ratio is between 0.4 and 0.5, methane is the fuel. The values between 0.5 and 0.9 indicate that the fuel is coal, oil, conveyor belt, insulation. The value of the ratio between 0.9 and 1.6 implies that wood is burning. Values above 1.6 generally occur in laboratory conditions. Thus, the gas chromatograph or sampling device(s) should be carefully examined if the ratio results above 1.6 during a possible mine fire. Figure 9 shows the trend in the Jones-Trickett ratio for the Gelik mine fire.

RESULTS AND DISCUSSION

The Gelik mine fire is not a spontaneous combustion event but resulted from an electrical contact. TTK applied several fire fighting actions including sealing-off the area, pumping nitrogen and filling the area with water to control the fire as quickly as possible. Seven seals were built initially but the expected decrease in the concentrations of CO and O₂ did not occur, possibly due to leakage. Thus a further 13 seals were constructed to prevent air from entering the fire zone. At the same time nitrogen was pumped to the zone to control the fire. However, the authors expected that this was not going to be effective since the volume of the void into which the nitrogen was being injected was too great. The result was as expected; pumping of nitrogen was not as efficient as the mine management desired. It was also decided to fill the fire zone with water to control the fire.

Continuous sampling from the fire area provided data for examination. The data obtained from the analysis, using a gas chromatograph, was examined and interpreted for different gas indices. CO

concentration is generally used in Turkish coal mines to monitor the mine atmosphere during mine fires. Graham's ratio (CO/O_2 deficiency) is another method used to assess the condition of mine fires.

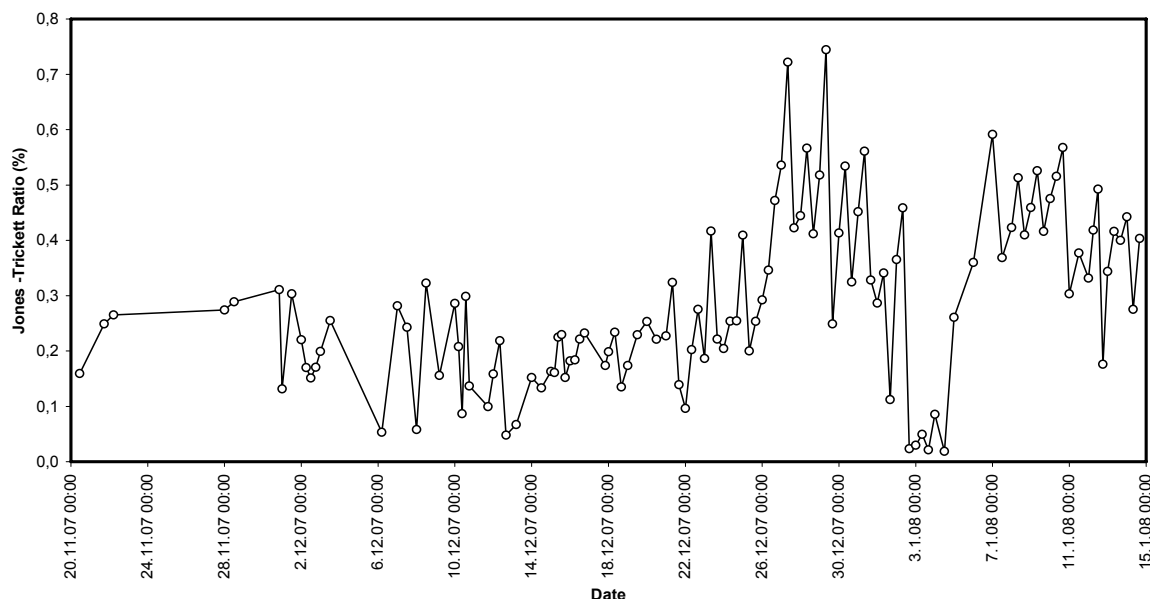


Figure 9 - The variation in Jones-Trickett ratio during Gelik mine fire

In the Gelik mine case, CO concentration at the beginning of the fire was measured as 1,250 ppm and decreased to normal levels (below 50 ppm for Turkish coal mines) after control actions (sealing-off, filling with water and pumping nitrogen) were implemented. However, CO concentration climbed to 3,250 ppm following the opening of the No.1 seal on drift No. 41300 (-260) in 1st January, 2008. As soon as the seal was opened the CO concentration started to increase dramatically and the area was resealed.

Although there was some observed decrease at the beginning of the fire, the oxygen concentration showed an increase in general. It can be considered that the precautions taken to combat the fire were effective at early stages, thus causing a decrease in the oxygen concentration but since there was air leakage into the fire zone it was not possible to reduce the oxygen concentration below 15-16%. It is considered that the air leakage is the result of either unfavourable geological conditions (faults, cracks etc.) or workings of adjacent private mines.

A sharp increase (from normal value to two) in Graham's ratio implied that there was an advanced heating which may turn into an extensive fire. After sealing, filling the fire zone with water and pumping nitrogen Graham's ratio reduced to its normal value indicating heating had cooled down. Nevertheless, after No.1 seal on drift No. 41300 (-260) was opened, there occurred a dramatic increase in Graham's ratio since CO concentration reached to 3,200 ppm on 14th January, 2008.

Willet's ratio followed the same trend as Graham's index showing an increase at the beginning and then reducing, implying that the fire was extinguished but then increasing sharply on 14th January, 2008, as result of the sharp increase in CO concentration.

While the indicators above implied that the fire seemed to be extinguished, the C/H ratio value approaching five suggested that there was possibly a localised, superficial fire behind the seals. This was proved after measuring high CO concentrations as soon as the seal was opened.

The Jones-Trickett ratio suggested that no combustion existed until the date 23rd December, 2007 when the ratio had a value of 0.42. After that time the ratio increased, ranging between 0.5 and 0.9, indicating there was a fire in which coal, oil, conveyor belts and insulation were the fuel. This suggestion proved to be true after a subsequent mine inspection confirmed that coal, conveyor belts and cables had burned.

CONCLUSION

Monitoring of the mine atmosphere is very useful in mine fire incidents. Relating the concentrations of individual gases to time, applying gases to equations, and examining their change over time are tools that have been successfully used to determine if a heating exists and, if so, the extent of the emergency. Hence, fire ratios (indices) play a very important role in assessing the status of a sealed-off mine fire. In this study, concentrations of gases in the mine atmosphere were monitored and several indices were utilized to assess the Gelik mine fire, Zonguldak, Turkey. Although the variation in CO concentration, Graham's ratio (CO/O_2 deficiency) and Willet's ratio indicated that the fire had been extinguished, the variation in O_2 concentration, C/H ratio and the Jones-Trickett ratio suggested that heating and/or the fire was still continuing. Opening of the seal No.1 and the resultant dramatic increase in CO concentration in a short period proved that the fire hadn't been extinguished. Therefore, it is concluded that it would be more precise to evaluate the ratios together instead of treating them alone since the uses of ratios may vary from case to case. Furthermore, attention must be paid to understand the individual fire and/or heating cases and to get more reliable results by carefully assessing different ratios or indicators.

ACKNOWLEDGEMENTS

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CRITICAL APPRAISAL TO ASSESS THE EXTENT OF FIRE IN OLD ABANDONED COAL MINE AREAS – INDIAN CONTEXT

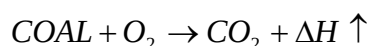
Niroj Mohalik¹, Ran Singh¹, Virendra Singh¹ and Durga Tripathi¹

ABSTRACT: Mine fires in Indian coal mines have a long history of over 140 years and major causes of fire are considered to be spontaneous heating of coal. Regular thermo compositional monitoring plays an important role for assessment of fire in abandoned coal mines and therefore different fire indices assist to categorise location and extent of fire. The paper highlights different methodologies to know extent of fire in old abandoned areas and reviews different fire indices for interpretation of status of mine fires with suitable case studies.

INTRODUCTION

Indian coal mines have a historical record of extensive fire activity in Raniganj coal field, 1865 (Dhar, 1996). The coal mine disasters in India revealed that fire and explosion contributed to about 41% of total fatalities (Sinha, et al. 2001). The major coal fires of India are located in Jharia, Raniganj, Singrauli and Singareni coalfields and about 160 mine fires had been detected in 1997. Out of these, more than 70 were reported from Jharia coalfield itself and total area affected extends over 6300 ha which threaten the township of Jharia (DLR 2005). It is estimated that 75 % of India's coal fires result from mining activities mainly due to the prolonged exposure of coal to atmospheric oxygen. The fires in Jharia and Raniganj Coalfield are mainly due to unprofessional mining and the past extraction of coal (Figure 1). The incidences of fire (about 70 %) in Indian coal mines are mainly due to spontaneous combustion of coal (Zutshi et al. 2001).

Spontaneous combustion in coal is an outcome of heating through oxidation processes, which may be aided by catalytic effects of other compounds (e.g. water, pyrite, etc.). In the process of oxidation, coal interacts with oxygen of air at ambient temperature, liberating heat, which if allowed to accumulate ultimately, would enhance the rate of oxidation and lead to devastating fire, known as spontaneous combustion. The oxidation of coal starts with exothermic chemical reactions and oxidation can be described as a process of three sequential steps. These are (i) physical adsorption, (ii) chemical adsorption or chemisorption, forming coal-oxygen complexes, and (iii) chemical reaction. The chemical reaction breaks down less stable coal-oxygen complexes and results in the formation of gaseous products such as CO, CO₂, and H₂; it can be simplified as follows :



Coal fires include surface coal seam fires, underground coal seam fires, old abandoned coal mine fires and carbonaceous dump material, as well as fires in stored coal or coal which is being transported. If a coal seam catches fire and remedial measures fail to be taken at an early stage, the seam may continue to burn for tens to hundreds of years, the time depending primarily on the availability of oxygen. So, demarcation and delineation of coal fire is a major threat to Indian coal mining industries. The paper highlights different methodologies to know extent of fire in old abandoned areas and reviews different fire indices for interpretation of status of mine fires with suitable case studies.

AVAILABLE TECHNIQUES FOR DETERMINATION OF EXTENT OF FIRE

Abandoned coal mine fires in India were mainly due to spontaneous combustion of coal and unscientific mining before nationalization in 1971. The process of spontaneous heating in coal depends upon several inherent parameters, namely mining, geological and chemical. Other external factors, include conveyor belt frictions, electrical short circuits, and explosions, as well as Bantulsi conflagration, dumping of hot ash, and illicit distillation of liquor.

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Fires in abandoned mines and overburden dumps are a relatively common occurrence in coal-producing areas. Abandoned mine fires present serious health, safety and environmental hazards due to the emission of toxic fumes, subsidence and the deterioration in air quality. Such fires usually depress property values for affected land and for adjacent areas. Although there are several fire extinguishment methods, in many cases the high cost and low efficiency are related to the inability

- (1) to accurately determine the location and extent of the combustion zones within an abandoned mine and
- (2) to identify some point at which the fire can be reliably considered extinguished.

To locate this type of remote, subsurface fire, it is necessary that suitable methods should be adopted and available data are interpreted according to an appropriate algorithm. The different techniques available so far for demarcation and delineation for fire areas include:

1. Field observations and surface thermal mapping of the area,
2. Thermo-compositional studies,
3. Geophysical methods and remote sensing, and
4. Data analysis and interpretation.

Field observations and surface thermal mapping of the area

The emission of smoke, fumes and goaf stink smell at surface fractures and vents is the conventional indicator of an abandoned mine fire. Because hot gases follow the path of least resistance, the surface evidence of fires may not be related by straight line paths to the source of combustion. Aerial infrared photography, through thermal scanner or infra red gun, determines temperature variations within a few inches of the surface, but is inappropriate when heated areas lie about 40 m beneath the surface. Interpretation of aerial infrared can also be complicated by the presence of heat absorbing surface features.

Thermo-compositional studies

In thermo-compositional studies temperature and gas analysis are carried out to determine the state of fire in abandoned mines. Borehole temperature measurements were the principal tool for detection of subsurface coal fires until the 1960s. Temperature and gas samples taken at boreholes are point source measurements; their accuracy is limited to a very small volume of 0.03 to 0.06 m³ (1 to 2 ft³), within a radial distance of 0.25 m (10 in) at the base of borehole. Their advantage lay in the fact that they could be performed in close proximity to the fire centre.

Subsurface changes in the concentration of O₂, CO₂, CO, CH₄, H₂ and other higher hydrocarbons have been used as the basis for geochemical combustion indicators. Although changes in the concentration of these gases may be related to combustion, they may also be produced by processes other than combustion. In contrast, previous work by the Bureau of mines USA has shown that the desorption of low molecular weight hydrocarbon gases from coal is strongly temperature dependent. A Mine Fire Diagnostic (MFD) Methodology was developed to determine the location and extent of combustion zones in abandoned underground coal mines. In Indian coal mines, carbon monoxide concentration is most widely used as an indicator for spontaneous heating/fire in combination with Graham's ratio. But, in the actual practice this is not the only ratio, on that basis the fire position can be interpreted and every ratio can't be used in all cases as it will vary case to case depending upon the extent and condition of fire. Always trends of fire ratios/indices are to be given due importance and not the absolute values.

Geophysical Methods

In geophysical methods remote thermal characteristics or variations in the gross composition of the mine atmosphere have not been routinely successful at locating isolated combustion zones. Core drilling and near-surface geophysical imaging techniques will produce adequate information on structural features, but are less reliable for indicating combustion areas. Elevated temperatures can alter the mineralogy of iron bearing rocks, but magnetic anomalies are more likely to be associated with areas that have been heated and cooled than with active combustion. Electrical terrain conductivity surveys may indicate water flow, i.e., areas where combustion is unlikely. Near surface seismic surveys and ground penetrating radar can indicate subsidence areas and changes in subsurface structure, but these are not necessarily related to combustion.

In the early 1960s, increasing availability of airborne and satellite-borne thermal scanner data made remote sensing a better tool for coal fire detection and monitoring on surface only. Infrared photography discriminates temperature variations only within a few centimeters of the surface, usually indicating heated vents and fractures. In-depth studies using Landsat TM and/or airborne thermal data were performed then in various countries. These studies applied an average emissivity value (0.96) to represent all land cover. Geographical information systems helped to store and analyse the data generated.

Data analysis and interpretation

The most common techniques of early detection of fire in Indian context are the thermo-compositional analysis. In order to locate a fire, it is necessary that (1) a fire have a measurable characteristic, (2) the characteristic be detectable through appropriate sampling methods, and (3) the sampling data are interpreted correctly. The procedure to determine the extent of fire consists of determining a unique feature for heated coal, a procedure for obtaining samples from abandoned mines and an algorithm for interpreting the data and relating it to a subsurface location. The diagnostic procedure used to locate underground fire zones is based upon two primary assumptions: (1) changes in the concentration of product of combustion gases are due to the presence or absence of heated coal and (2) the temperature recorded with depth in surface borehole near fire zone. The data obtained from these analyses are important for interpretation to know various stages of coal mine fire.

COLLECTION OF MINE AIR SAMPLES

Mine air generally contains CO₂, O₂, CH₄, N₂ and in special cases CO, H₂, H₂S, NO_x and other higher hydrocarbons. Sampling of mine air is required for qualitative assessment for safety of miners and mines. Composition of mine air samples not only indicates the presence of explosive or obnoxious gases but also enables adoption of safety measures in time and also provides valuable information about the condition of the mine during or after explosion. Determination of status of mine atmosphere depends mainly on the collection of representative samples and if sampling is faulty, the painstaking accuracy of the analysis is a waste. Proper sampling mainly depends upon the knowledge and alertness of the sampler, correct sampling procedure and proper preservation of the samples. In addition to this, sampling point, sampling site, sample container and sampling techniques are important in deciding the correctness of representative sample. In practice - sampling pipe becomes too large due to depth of mining. In such circumstances, purging should be at least fifteen times of the volume of the sampling pipe, so that stagnant gases within sampling pipe purge out properly. In fire areas it is very difficult and time consuming to collect representative samples in fire event analysis. So a large number of gas samples are required for an extended period for assessment of the area.

Analysis of mine air samples

The normal atmospheric air consists of N₂, O₂ and CO₂ in the proportion of 79.04%, 20.93% and 0.03% respectively. The deviation of the proportions of these gases in mine-air is very useful for early detection of spontaneous combustion or assessment of fire in underground areas. During any incidence of fire the thermal splitting of coal is set-up and carbon monoxide together with other gases and smokes are produced at their characteristic temperature. The hierarchy and order of appearance of gases may vary from coal to coal. For old abandoned mines there is no distinct technology used in Indian condition except thermo compositional analysis from boreholes. Different gases such as CO, CO₂, CH₄, H₂ and few other lower unsaturated hydrocarbons, which are product of the combustion due to mine-fire activity, are collected from the fire areas and are analysed.

Since the beginning of the twentieth century a number of fire indices/ratios have been suggested to assist in the interpretation of status of fires in underground coal mines (*J. L. Graham, 1914, Timko et al. RI-9362, Litton, C.D RI-9031, Ghosh and Banerjee et al. 1967, Jones & Trickett, 1954, Mitchell, D.W, 1984, Morris, R, 1986, Willet, H.L, 1961, Tripathi et al. 1996*). A brief description of different fire indices/ratios used worldwide for detection and assessment of mine fire are given in Table 1. All the fire indices/ratios are mainly based on different gases produced during fires. These ratios are based on two assumptions

1. The air which has been supplied to mine having 20.93% O₂, 79.04 % N₂ and 0.03 % CO₂ (excluding other gases)

2. Nitrogen is neither added to the air nor taken from the air concerned in the oxidation.

CASE STUDY

Satgram Fire Project

Sitaldas Opencast Project under J. K. Nagar Fire Project of Satgram, Area, ECL, is located in the central part of Raniganj Coalfield for extraction of 6 – 6.5 m thick Nega Seam (R-VIII). The major outcrop of the Nega seam is exposed in Nimcha Village (population approximately 3000) and average gradient of seam varies between 2° to 5° towards S44°E. The dip side of Nega seam (R-VIII) was worked out using bord and pillar method of mining under shallow depth cover surrounding Nimcha village before nationalisation in 1971. Cracks and fractures were developed because of unscientific extraction of coal under shallow cover and resulted in spontaneous heating/fire and subsidence in the area, due to air ingress. It was observed that fire was within 60 to 90 m of the periphery of Nimcha village. So, trench cutting to floor of Nega seam encircling the Nimcha village was appropriate technology instead of normal blanketing and was filled with incombustible material. Maximum width of the trench was 195 m at top and 145 m at the bottom where trench was divided into three patches along the length namely patch 1, patch 2 and patch 3. Patch 1 and patch 2 could not be completed due to middle district road connecting to grand trunk road. The gestation period of trench cutting of patch 2 made a scope for illegal miners to enter into Nega seam towards Nimcha village and galleries were developed towards Nimcha village to extract coal at different places of the trench. A pothole of 10 m diameter took place on 4th of January 2008 near southern part of Nimcha village. Smoke and toxic gases were coming out from potholes, which is endangering the village and affecting environment. Thermal mapping was carried out by Heat Spy Infra red thermometer in early morning near the subsided area and surrounding village. The temperature variation of 5° to 7° was found near by pothole and some of places surrounding to Nimcha village. In the first phase nine boreholes 150 mm (6") diameter with casing and leads, at least up to 3 m were made at judiciously selected location (Figure 1). The location of maximum boreholes were drilled where temperature was found more than ambient by thermal mapping and giving due importance to periphery of Nimcha village. Subsequently two more boreholes were drilled near borehole no. 3 and observed no symptoms of fire.

Result and discussion

Thermo-compositional studies from all the borehole shows that fire does not exist except at borehole no. 3. With the exception of borehole no. 3, all the boreholes were full of strata water. The temperature was found to be ambient (Table 2). High percentage of CH₄ (14%) and CO (1400 ppm) was found in borehole no. 3 which showed that a moderate fire existed in localised area. The temperature in borehole no. 3 was found to be 80°C and Graham's ratio, CO₂/O₂ ratio indicated that fire existed in the localized area surrounding it (Figure 2). The decreasing trend of oxygen percentage during investigation period along with marked presence of CO and CH₄ in borehole no. 3 confirmed that the extent of fire towards Nimcha village was up to the borehole no. 3 during the investigation period. About 9000 m³ of top soil was used for blanketing the surface area and edges of trench as per site specific condition. Fire fighting operation was carried out through borehole no. 3 to control and combat fire in the localised area by chemical means, subsequently filling with incombustible material to stabilize the area outside the trench limit line. The remaining part of the trench i.e. North East side of trench towards village was also completed for the safety of village.

Instrument: Infrared Thermometer
Date: 11/01/2008, Time: 5:30am
Ambient Temperature: 11°C /12°C
All temperatures are at surface
BH 1-11 – Borehole No 1 to 11

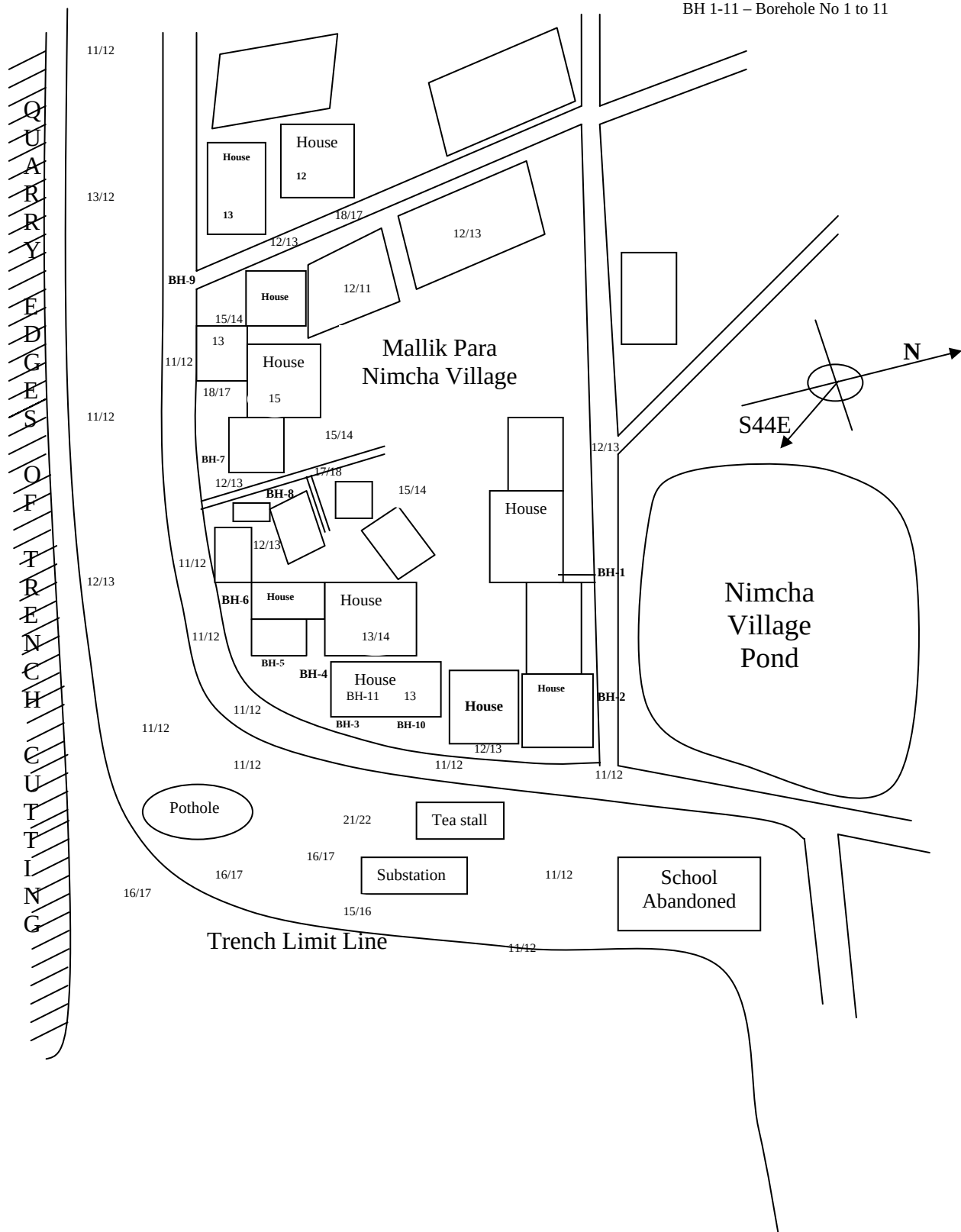


Figure 1 - Location of boreholes and thermal mapping of the area towards Nimcha village

Table –1 Different fire indices/ ratios to assess the status of fire

Fire Indices/Ratios	Significance/Interpretations	Advantages	Limitations
Graham's Ratio (GR) $= 100\text{CO}/(0.265 \times \text{N}_2 - \text{O}_2)$ and (J. L. Graham, 1914-15, 1920-21) In case of failure of GR, where CO extinction is not indicative of status of fire, than the CO/CO ₂ def.* Ratio can also be applied. $\text{CO}/\text{O}_2\text{de}^* = 100\text{CO}_2/(0.265 \times \text{N}_2 - \text{O}_2)$	< 0.5 - Normal	- Since both numerator and denominator are affected, so the ratio remains independent to dilution of air or methane. - Gives early detection of fire if it increases continuously. - Assessing the status of fire.	- Does not give extent of fire i.e. amount of coal involved. - Disappearance of CO does not mean complete extinction of fire - If the products of combustion are diluted by black damp (N₂) or O₂ deficient air, the ratio would be affected.
	>0.5 - < 1.0 - Existence of Heating		
	>1.0 - < 2.0 - Advance Stage		
	> 3.0 - Active Fire		
	> 7.0 - Blazing Fire		
Young's Ratio (YR) = $100(\text{CO}_2 - 0.03)/(0.265 \times \text{N}_2 - \text{O}_2)$ (CMRI Report, 1991)	< 25 - Normal	CO₂ found, where fire exists in advance stage.	Extraneous origin of CO₂ and its solubility in water render misinterpretation.
	25-50 - Superficial Heating		
	> 50 - High Intensity Fire		
JTR = $(\text{CO}_2 + 0.75 \text{CO} - 0.25 \text{H}_2) / (0.265 \text{N}_2 - \text{O}_2)$ (Jones & Trickett, 1954)	< 0.4 - No fire	It can differentiate the nature/type of fire.	- Does not provide status of fire. - Cannot provide the early detection of fire.
	0.4 – 0.5 – Methane Burning		
	0.5 - 1.0 - Coal, Oil, Conveyor Belt Burning		
	0.9 - 1.6 – Timber Burning		
	> 1.6 – Error in Analysis		
Willets Ratio = $\text{CO}/(\text{Excess N}_2 + \text{CO}_2 + \text{Combustible Excess (N}_2 + \text{Ar)})$ $= \text{Sample (N}_2 + \text{Ar)} - (3.8 \times \text{Sample O}_2)$ (Willet, H.L, 1951-52)	Actual figure vary from seam to seam. The ratio increases as temperature increases. It also depends upon the magnitude and extent of fire.	Gives early detection of fire in presence of production of CO.	- Does not provide status of fire. - If CO disappears then prediction of the status of fire is very difficult.
C/H Ratio = $6(\text{CO}_2 + \text{CO} + \text{CH}_4 + 2\text{C}_2\text{H}_4) / 2(0.265 \times \text{N}_2 - \text{O}_2 - \text{CO}_2 + \text{CH}_4 + \text{C}_2\text{H}_4 + \text{H}_2 - \text{CO})$ (Ghosh & Banerjee et al., 1967)	0 – 2.0 - Normal	- It can characterise intensity and extensity of fire. - Can distinguish between coal and wood fire. - Large range of value provide a better tool for characterisation of a fire	
	3.0 – 4.0 - Superficial Heating		
	> 5.0 - Active Fire		
	> 20.0 - Blazing Fire		
Morris Ratio = $\text{N}_2 - \text{C}_{\text{NO}} \text{O}_2 / (\text{CO} + \text{CO}_2)$ and $\text{N}_2/(\text{CO} + \text{CO}_2)$ (Morris, R, 1986)	Actual figure vary from seam to seam. The ratio increases as temperature increases. It is sensitive to change in the state of the gases within a sealed fire zone.	- For a particular seam it is very useful - It is used to estimate the temperature of a heating. - It may be used to warn mine personal that an explosion may occur within the fire seals.	- Does not provide status of fire. - Ratio varies randomly with high temperature depending upon the production of CO and CO₂.
Litton's Ratios (R) = $1/3 \times \text{CO}_s \times \text{R}_g^{-3/2} \times \text{O}^{-1/2}$ $\text{R}_g = 100 - 4.774 \text{O}_2 - \text{CH}_4 - \text{C}_2\text{H}_2$ $\text{CO}_s = \text{CO in ppm}$ (Litton, C.D, 1986)	≤ 1.0 – Equilibrium Exists for 30 days then ambient temperature reached	- Provides better information for sealed-off area. - Very useful for reopening of sealed-off area.	- Does not provide status of fire. - It cannot provide the early detection of fire.
	> 1 – Smouldering combustion or above ambient temperature (oxidation takes place)		

Desorbed Hydrocarbon Ratio (RI) = $\{1.01 \cdot (\text{CH}_4 + \text{C}_2\text{H}_2 + \text{C}_2\text{H}_4) - \text{CH}_4 / (\text{CH}_4 + \text{C}_2\text{H}_2 + \text{C}_2\text{H}_4) + 0.01\} \cdot 1000$ (Kim, A.G, 1988)	For Bituminous Coal	• Can assess the status of fire. • Gives a better result for sealed-off area having a no of sampling locations. • Can find out the location of fire if sampling point is evenly distributed.	• It cannot provide the early detection of fire. • Without hydrocarbons the value will be zero, which gives sometimes no indication of fire.
	0 – 50 - Normal		
	50 - 100 – Possible Source of Heating		
	> 100 - Hot Zone		
Oxides of Carbon Ratio = CO/CO_2 (Kuchta et al. Al. 1982, Mitchell, D.W, 1989)	< 2.0 - Normal	• Gives early detection of fire if it increases continuously. • Assessing the status of fire.	• Extraneous origin of CO_2 and its solubility in water render misinterpretation
	2.0 – 13.0 - Superficial Heating		
	> 13 - Blazing Fire		
H_2/CH_4 (Mitchell, D.W, 1990)	The ratio increases as temperature increases. It also depends upon the magnitude and extent of fire.	• Gives better information when fire is in advance stage or blazing fire.	• Cannot provide the early detection of fire.
RI/RE Ratio Relative Efficiency (RE)= $(0.265\text{N}_2 - 3.83\text{O}_2) / (\text{CO} + \text{CO}_2)$, Relative Intensity (RI)= $\{(1 - 3.83\text{O}_2)/\text{N}_2\} / [\text{CO}/(0.265 \cdot \text{N}_2 - \text{O}_2)]$ (Mitchell, D.W, 1984)	Increase in RI/RE value indicates fire is an ascending order of moving towards the sampling locations or conversely.	• Gives better result in locating the proximity of fire, when sampling locations spread throughout the area.	• Does not provide status of fire. • Cannot provide the early detection of fire.

Table – 2 Temperature measurements in different boreholes

Date	Time	Borehole Number (temperature in °C)											Ambient Temperature
		1	2	3	4	5	6	7	8	9	10	11	
17.01.08	10.30am	--	31	89	--	--	--	--	--	--	--	--	31
19.01.08	12.10pm	32	32	83	34	36	--	36	--	36	--	--	32
22.01.08	11.30am	--	--	82	28	26	--	27	--	24	28	26	27
29.01.08	11.55am	21	22	81	28	26	28	27	28	27	28	27	27
06.02.08	11.15am	24	24	81	16	14	22	31	17	29	31	31	30

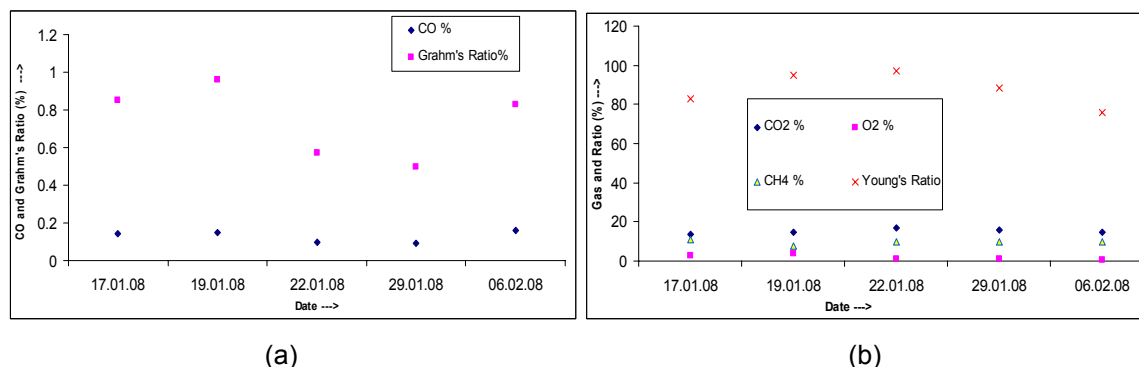


Figure 2 - (a) Graham's ratio and CO percentage of borehole no 3
 (b) Young's ratio, CO₂, O₂ and CH₄ percentage of borehole no 3

Mudidih Colliery

Mudidih colliery lies in Sijua Area at a distance of about 15 km towards west from Dhanbad railway station and this colliery consists of three sections namely Tetulmari, Mudidih and Jogta, which were amalgamated during take over by BCCL in 1971. Numbers of major faults of 3.36 m to 128.0 m of throw are passing through the colliery properties. Besides these major faults, large numbers of localized faults of small throw are also found frequently. All seams were developed on board and pillar mining. Fire in IX (T) and IX (B) seam in the adjoining colliery has been in existence since 1960s. In the dip side, the smoke and fume were seen in the recent past in adjoining area beyond western boundary of 7/8 incline of Mudidih colliery. In the year 2000, isolation stoppings of VIII seam goaf area below the above area of Tetulmari colliery were reported for temperature increase. Suddenly huge amount of black smoke coming out of the incline mouth and accordingly stoppings were erected. It was suspected that fire was coming from an old isolation stopping. Accordingly all the openings of the VIII seam leading to the incline were sealed, and inspection to the fire/stopping was ceased. Also, the progress of fire towards the boundary of 7/8 incline of Mudidih colliery from Tetulmari colliery became difficult to monitor.

Result and discussion

Five boreholes (Borehole No. 1 to 5) were available in the western side of the incline of Mudidih colliery. Borehole No. 2 was drilled up to IX (T) seam at a depth of 27.64 m and borehole No. 1, 3, 4 and 5 were drilled up to VIII seam at a depth of 50, 43, 53 and 60 m respectively. Huge amount of hot gases were observed coming out from borehole No.2, indicating positive pressure, while borehole No. 1, 3, 4 and 5 were showing negative pressure. After carrying out thermo-compositional investigation of the different boreholes, no CO trace was detected in the borehole, but CH₄ has been found in the borehole Nos 1 to 5, which indicated the presence of heating/fire. The ratio of CO₂/O₂ def. value indicates that heating is in advanced stage in the sealed off area. Also, Jonnes Tickett Ratio (JTR) value was between 0.4 to 1.1 in different boreholes indicating that the fire was due to coal and wood

burning. It was concluded that the gas sample data indicates that fire is present in the surrounding area.

Table – 3 Temperature measurements in different boreholes

Date	Borehole Number (temperature in °C)					Ambient Temperature
	1	2	3	4	5	
04.05.06	44	63	50	48	36	36
18.05.06	38	59	33	37	35	35
06.06.06	37	54	34	36	36	34
20.06.06	39	65	35	40	38	36
04.07.06	33	68	35	32	39	30
13.07.06	35	86	35	35	32	31
27.07.06	36	98	36	35	34	32

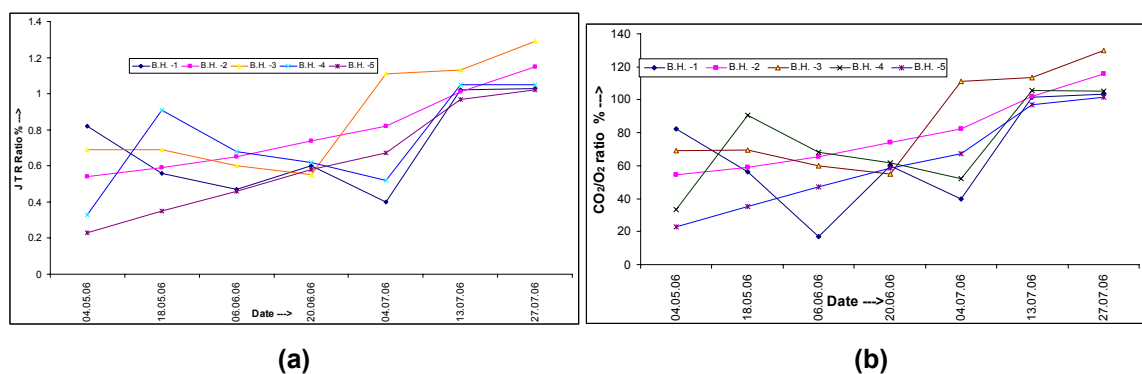


Figure 4 - (a) JTR ratio in different borehole (b) Graham's ratio in different borehole

CONCLUSION

The abandoned coal mine fires blazing in the Indian coalfields are not only consuming huge quantity of coal but also preventing exploitation of coal in the adjoining areas and in the underlying coal seams. Mine fires also affect the environment, due to release of noxious and toxic gases from abandoned coal mine fire areas. Also emitted are greenhouse gases (GHG) which contributes to global warming.

In Indian coal mining industry, the general trend for assessment and extent of fire is determined from thermo-composition studies i.e. interpretation of Grahams ratio i.e. CO/O_2 def. ratio only if sampling from fire area is proper. The trends of fire indices/ratios are to be given due importance for determination of extent of fire in abandoned coal mine and not the absolute values. Every ratio can't be used in all cases, as it will vary case by case, depending upon the extent and condition of fire.

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SPONTANEOUS COMBUSTION MANAGEMENT – LINKING EXPERIMENT WITH REALITY

David Cliff¹

ABSTRACT: Despite the best efforts of researchers to try to understand spontaneous combustion it still affects many mines. Laboratory testing and modelling have been available for many years and yet they are still not able to reliably predict the propensity for a coal seam to spontaneously combust.

The complexities of the spontaneous combustion process are explored by delving into the chemistry of the oxidation process. It is able to demonstrate why the testing and modelling of spontaneous combustion can be of limited accuracy. Laboratory tests and simulations are carried out under conditions cannot reflect the full complexity of the underground environment. This does not mean that the experimental work is meaningless, but the results need to be included as part of a proper risk assessment that includes the contributions and influences of other parameters currently not able to be adequately modelled or simulated in the laboratory. In addition laboratory testing can offer insights into the influence of such things as water content, ash content and particle size.

It is recommended that the Trigger Action Response Plans (TARP) in place in underground coal mines go beyond detection of spontaneous combustion and include indicators that identify the increased likelihood of spontaneous combustion and allow for controls to be put in place in time to prevent spontaneous combustion from occurring.

INTRODUCTION

Researchers have been attempting to develop methods for predicting the likelihood of spontaneous combustion in coal mines for over 100 years (there are some records going back to the Middle Ages). Despite this, major spontaneous combustion events continue to occur in Australian Underground Coal Mines roughly once a year. These events have not caused any loss of life since Moura No.2 in 1994 but they still necessitate the evacuation of the mine and many days lost production, not to mention the costs of remediation. If the problems in coal stockpiles and open cut mines are added then the severity of the problem escalates significantly.

It is standard practice to establish a spontaneous combustion management plan supported by a risk assessment (RA). This RA is based at least upon prediction of the likelihood of the coal to spontaneously combust. The starting point for this prediction is laboratory testing sometimes coupled with computer modelling. Why are these predictions of limited accuracy?

Consider this example. A consultant provided an assessment of the spontaneous combustion potential of the roof coal above a longwall block for an underground coal mine. This assessment was based upon a combination of small scale laboratory testing of R_{70} indices for the various coal bands in the roof and computer model that attempted to simulate the goaf conditions. The conclusion from the study was that "there does not appear to be any spontaneous combustion potential associated with any of the bands tested". Within two years, for the first time in the history of the mine there was a major spontaneous combustion event that took nearly 12 months to control and cost many millions of dollars in lost production and direct control costs. Neither the mine nor the consultant have been identified, it is not the purpose of the paper to criticise them. There are many examples like this.

To understand why there is often a significant disparity between assessments and reality it is necessary to delve into the fundamental processes involved, appreciate the complexity of the spontaneous combustion process and recognise the multitude of influences and parameters.

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THE CHEMISTRY OF SPONTANEOUS COMBUSTION

The structure of coal

Conventionally spontaneous combustion is thought of as simply coal reacting with oxygen to form carbon dioxide and carbon monoxide. This reduces coal to a simple uniform molecule, which of course it is not. Coal is a macro-molecule which has many different components, each with their own reactivity. Figure one depicts a typical model structure. It shows that coal contains a wide range of functional groups including those containing purely carbon and hydrogen as well as many including oxygen, such as aldehyde, alcohol, ketone, ether, ester, carboxylic acid. These latter functional groups are much more reactive than the pure hydrocarbon groups (Kalema and Gavalas, 1987).

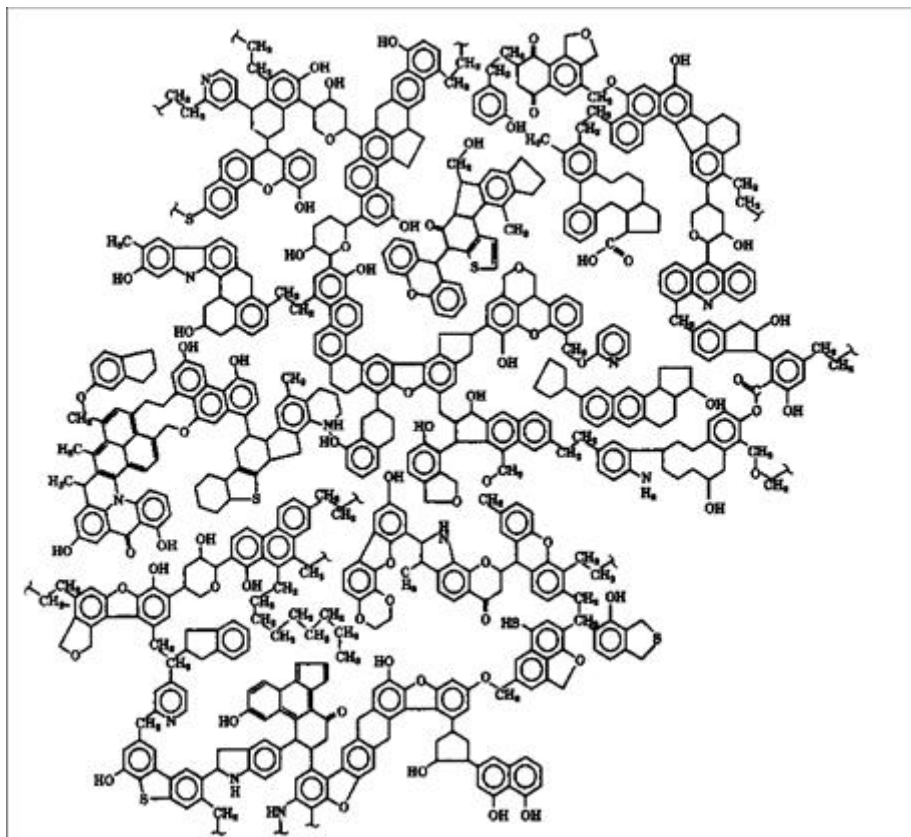
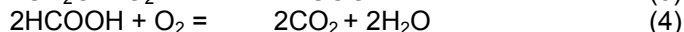
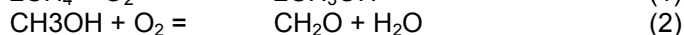
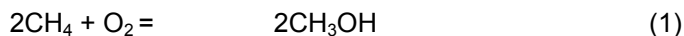


Figure 1 - Coal Model (Wells and Smoot 1991).

The oxidation process

Much can be learnt from mainstream organic chemistry. For example: consider the oxidation of methane to carbon dioxide (Chang, 1994).



The activation energy of each reaction decreases from (1) to (4), this means that less energy is required to initiate reaction (4) than to initiate reaction (1). However the amount of energy released by reaction (4) is much less than if the oxidation process started at reaction (1) and ran through all four reactions. Relating this to coal suggests that coal with significant numbers of functional groups aligned to reactions (3) and (4) are likely to be more reactive than those without them. Conversely the latter coals will generate more heat when they do react with oxygen. The activation energy required to break aromatic hydrocarbon bonds is even higher than for aliphatic hydrocarbon bonds. Generally high rank coals are low in oxygen content and have few reactive functional groups, they are the most

aromatic. Therefore generally they are not highly prone to spontaneous combustion and produce the most heat when burnt. Low rank coals, contain more oxygen than high rank coals, and have significant concentrations of oxygenated functional groups. These are generally more reactive and prone to spontaneous combustion but do not generate as much heat when burnt.

The role of water

Of course for coal the oxidation process is a much more complicated chemical system than the simple one outlined above and would involve hundreds of interrelated chemical reactions. Each one has its own temperature and reactant concentration dependence. A further complication to the reaction scheme is the role of water. Water is often only included in spontaneous combustion for its heat transfer capacity – heat of vaporisation and heat of condensation. However it is now clear that water also can become involved in the chemical reactions. Water produces highly reactive free hydroxyl and hydroperoxy radicals, HO and HO₂ (Ford, 1981). These radicals can act as catalysts in the oxidation process changing the balance between the various reaction paths and mechanisms. Stoichiometry or the ratio of fuel to oxygen also shifts the balance of the reactions.

Other influences

A further complication is that the reactions will occur in four dimensions – three orthogonal spatial dimensions and varying also with time as the nature of the coal changes with time. The concentrations of the reactants and products will vary in each of the dimensions and so for any simulation to be of value it must account for this.

Add to this a whole raft of intrinsic and extrinsic parameters that can influence the spontaneous combustibility of a coal sample. Examples are listed in table 1 below. These affect the ability of the coal to react with oxygen and for heat to be retained or removed.

Table 1 - Examples of intrinsic and extrinsic parameters (from Cliff and Bofinger 1998)

Intrinsic (Properties of the coal)	Extrinsic (external to the coal)
Friability	Thickness of coal seam
Caking property	Ventilation patterns
Heat capacity	Gas drainage
Thermal conductivity – coal and adjoining strata	Geological features – faults, folds and dykes, strata conditions
Coefficient of oxygen absorption	Seam dip
Surface area	Multiple seams
Gas content	Petrography

SMALL SCALE LABORATORY TESTING

Small scale oxidation tests remove most of these variables because the samples are:

- crushed – removing the effect of surface area, and seam gases
- dried – removing the effect of moisture,
- subject to high air flows – reducing residence time and altering the chemical balance – this favours fast reactions and single stage chemistry
- reacted in uniform temperature environments – removing the effects of ranges in temperature

- taken to represent an entire coal seam – not allowing for natural variation in ash content and chemical composition within the seam (Cliff and Bofinger, 1998).

The oxidation occurs in a uniform environment and only single stage reactions are observed. Despite this there is a broad correlation between the reactivity determined by small scale testing and the historical occurrence of spontaneous combustion in coal mines. Small scale tests only attempt to identify the inherent reactivity of the coal sample. The measure of reactivity is generally relative to other coals with a known history of spontaneous combustion activity. In Australia the most common test is the R_{70} test which determines the rate of rise of temperature of a sample under control conditions between 40 and 70 °C. Details and critiques of this method can be found elsewhere. As such the determination is the starting point for any assessment of the propensity for spontaneous combustion, not the end point (see for example: Beamish and Arisoy, 2008).

MEDIUM SCALE LABORATORY TESTING

Medium scale tests attempt to bridge the gap between the small and the large scale testing, using larger sample sizes (70 kilograms vs 70 g) and some of the conditions found to stockpiles and goaves (samples not dried nor finely crushed). This method is showing promising results and correlation with events in stockpiles. Unfortunately even this degree of complication is demonstrating reaction processes that we are only starting to understand. This larger scale reactor does allow multi stage reactions to occur in a linear dimension. This allows the study of the interrelationship between the components, and the impacts of moisture, and ash. By understanding these factors the correlation between testing and reality can be improved (Hitchcock and Beamish, 2008).

LARGE SCALE LABORATORY TESTING

Large scale tests use run of mine coal samples of up to 16 tonnes and allow more complex conditions closer to the mine environment to be established but suffer from the time it takes for spontaneous combustion to occur and presupposes that the conditions used in the test simulate the goaf of a coal mine. The large scale reactor tests demonstrate how complex the oxidation process is as it allows for the three spatial dimensions as well as variability in gas concentrations and moisture in each of the dimensions (Clarkson, Beamish and Cliff, 2005).

COMPUTER MODELS

Current computer models are still in their infancy and do not allow for the complexities outlined above. Indeed it is hard to model all the processes involved in spontaneous combustion as some are still unknown. There have been some attempts to allow for the conditions found in a goaf but as described above they can give the wrong answer. Attempts to calibrate the models against experiments have met with limited success (Humphreys, 2008).

DISCUSSION

Laboratory testing should not be abandoned. Testing is providing valuable information on the effects of ash content and other coal composition variables. The medium and large scale tests will give insights into the effects of the other parameters and allow for adequate computer models to be developed. Any assessment of spontaneous combustibility must recognise the limitations of the process used and embody processes to monitor and identify factors that will alter the validity of the assessment.

It is very naïve to expect one single test or calculation to give a universally applicable outcome. When a NERRDC funded study (McGowan, 1987) on thick seam mining was carried out at Ulan Colliery in the late 1980's over 60 R_{70} tests were carried out on the working seam over its 10 m height. Approximately 20 % of the tests were classified as highly prone to spontaneous combustion. The vast majority of samples were determined to have low to moderate reactivity. If only one sample was taken then depending on where it was taken from the reactivity would have been assessed as anything from negligible to highly prone. Testing is the starting point, once this assessment is done then the influence of the extrinsic factors needs to be applied, by experienced mine site personnel, as part of

the risk assessment process. This process that should be ongoing, being modified as parameters change or new information comes to hand.

For example: in the case study above, it became evident after mining that unlike previous goaves, the atmosphere in the goaf was remaining close to fresh air, and not going inert. With hindsight this was an indicator that something was different. No indicators of active spontaneous combustion were observed in the goaf for many months after mining, which reinforced the unlikelihood of an event occurring. The assumptions about spontaneous combustibility for this coal are hedged around the atmosphere going inert within a relatively short period of time. The laboratory tests did indicate that the roof coal had a high propensity for spontaneous combustion ($R_{70} > 1$). Historically this had been offset by the inert goaf atmosphere.

Laboratory experiments and models can also give an indication of control measure effectiveness, eg inertisation, or replacing the water content. In addition they can give an estimate of the significance of a change in a key parameter, such as seam thickness, gas drainage or ventilation patterns.

Testing is also giving us insights into the fundamental processes of spontaneous combustion. For example: Recent work by Hitchcock et al has shed new light onto the complex formation process of hydrogen during coal oxidation. This suggests that there is at least two different formation paths for hydrogen during coal oxidation, one relating to the direct oxidation process and the other relating to an anaerobic reaction process downstream of the oxidation zone, at lower temperature, where the coal is still drying out and may contain oxygen absorbed onto the surface of the coal (Hitchcock, Cliff and Beamish, 2008).

TRIGGER ACTION RESPONSE PLANS (TARPS)

Testing and modelling can provide the triggers for detecting abnormality prior to excess oxidation occurring. Triggers do not need to be just the products of oxidation, rather they could include:

- the presence of oxygen where it should not be
- unexpected geology – eg increased seam thickness
- changes to mining method – reduced retreat rate or increased retreat rate
- strata issues that impact upon ventilation of the goaf or mining method
- ventilation changes that affect the flow of gases in the goaf
- changes in coal quality

From geology and core samples zones of higher propensity can be identified, where additional care should be taken. This can include areas where mining may be slow due to difficult terrain, or where extra coal is left in the goaf due to faulting or folding.

Characterisation of the gas atmosphere throughout the goaf that should exist and being able to determine deviation from this early, will enable proactive controls to be put in place. It is better to monitor too much than not enough.

CONCLUSION

Spontaneous combustion management is a complex process and should not be oversimplified. Placing too much reliance on small numbers of tests or computer simulations should be avoided. They provide vital information to assist in the assessment of spontaneous combustion risk. The results of such studies should be included in the overall assessment process, along with knowledge of local conditions and intrinsic and extrinsic factors. More research and testing under controlled conditions is required to allow for the refinement of models and to identify all the factors that can influence spontaneous combustion. Management is an ongoing process.

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COMPARISON OF LABORATORY BULK COAL SPONTANEOUS COMBUSTION TESTING AND SITE EXPERIENCE – A CASE STUDY FROM SPRING CREEK MINE

Basil Beamish¹ and Robin Hughes²

ABSTRACT: As part of an on-going commitment to leading practice in spontaneous combustion assessment and management, Spring Creek Mine has adopted a strategy of bulk coal testing to obtain data on hot spot development in broken coal, including the associated gas evolution pattern. This approach has been successfully integrated into the spontaneous combustion management plan for the mine and has enabled appropriate actions to be taken in response to the coal behaviour during a heating. Site experience gained at Spring Creek is presented to compare with the laboratory scale testing results. A key feature that has been identified for the Spring Creek coal in the laboratory is the stage of hot spot development associated with moisture evaporation as the hot spot prepares to migrate towards the air source. This stage of delayed thermal runaway provides a lead time for appropriate actions to be taken to control the heating.

INTRODUCTION

Spring Creek mine is located on the west coast of New Zealand's South Island near Greymouth (Figure 1). The mine commenced development into the Upper Rewanui series of coal seams in 1999. The Upper Rewanui coal measures have been exploited by several large mining operations in the area over the past 100 years, all of which have experienced considerable difficulty in controlling spontaneous combustion. In their preparations for the commencement of extraction in 2004 mine management recognised that the propensity for spontaneous combustion would impact significantly on mine design, particularly the effect that the mining method (high pressure water monitors) would have on the ventilation circuits within the extraction panels. To facilitate the early recognition of the on-set of a spontaneous combustion situation, assistance was sought from the University of Queensland to accurately determine the issue of constituent gasses under various conditions. A 2-metre column test (Beamish *et al*, 2002) was conducted on a sample of Spring Creek coal in March 2005. The results of this test have assisted in the early recognition of spontaneous combustion potential allowing appropriate and timely intervention. One spontaneous combustion event occurred at Spring Creek mine in July 2008. This event was recognised and dealt with at a very early stage as a result of the gas signatures gained from the 2-metre column test, resulting in minimal lost production time and virtually no loss of reserves.

This paper presents the bulk self-heating test results for Spring Creek coal and places them into perspective with the experience gained from a heating event at the mine.

UQ 2-METRE COLUMN TESTING

Samples

Fresh ROM coal was supplied from the Spring Creek Mine in 4 x 20L sealed buckets. Prior to loading into the column, a size distribution of the coal was determined. The average particle size of the sample was 6.39 mm, based on the procedure described by Kunii and Levenspiel (1991) for estimating the surface-volume average particle size from the size distribution of the coal. Three samples were taken during the initial size distribution analysis to obtain data on the as-received moisture of the coal. The average test moisture of the batch was 11.6%, which indicated that the coal had retained its as-mined moisture. In addition, three lump samples were taken and stored in the laboratory freezer until they could be tested in an adiabatic oven to determine the R_{70} value of the coal. It should be noted that upon opening the 20 L buckets significant out-gassing occurred, which

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proved to be rich in methane as indicated by a Minigas handheld gas detector. Hence, the buckets were effectively self-inertised from seam gas desorption.

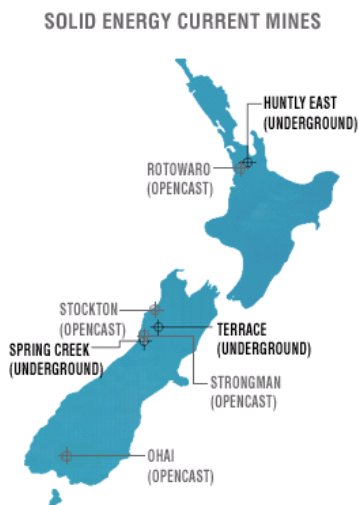


Figure 1 - Location of Spring Creek Mine

Self-heating test procedure

Beamish *et al.* (2002) describe the basic operation of the UQ 2-metre column. The column has a 62 L capacity, which equates to 40-70 kg of coal depending upon the packing density used. The coal self-heating was monitored using eight evenly spaced thermocouples along the length of the column that were inserted into the centre of the coal at each location (Figure 2). Eight independent heaters correspond to each of these thermocouples and were set to switch off at 0.5°C below the coal temperature at each location so that heat losses were minimised radially.

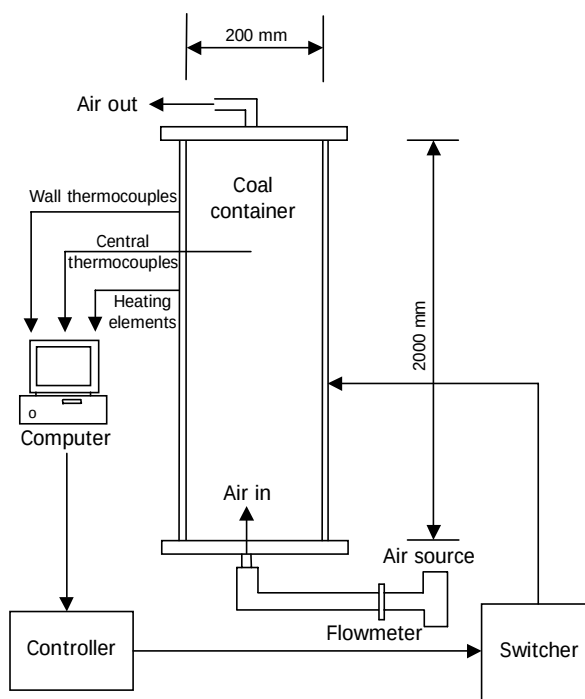


Figure 2 - Schematic of the UQ 2-metre column self-heating apparatus (modified from Arief, 1997)

The starting conditions of the test were agreed upon with the mine so that the test was performed as close as possible to the mine environment. Once all the coal was in the column it was sealed and the heaters used to set the starting coal temperature, which for this test was 25°C . This was achieved

overnight. A sample of the gas evolved in this static condition was taken with a peristaltic pump before air was introduced to the coal at 0.25 L/min. The inlet air temperature was maintained between 23-24°C for the duration of the test. A computer recorded the temperature data from each thermocouple at ten-minute increments. The column has several safety devices including computer-controlled trips on the external heaters and a temperature trip on the air inlet line. These were set to ensure maximum safety during operation of the column. Gasbag samples were taken from the exhaust at regular intervals as self-heating progressed. All gasbags were analysed by a registered laboratory (Simtars) using an HP Quad Gas Chromatograph to determine the concentrations of oxygen (O₂), nitrogen (N₂), hydrogen (H₂), methane (CH₄), carbon monoxide (CO), carbon dioxide (CO₂), ethane (C₂H₆) and ethylene (C₂H₄).

RESULTS AND DISCUSSION

Coal rank and self-heating rate values

The proximate analyses data (Table 1) from a nearby borehole (DH851) has been used to assess the ASTM rank and Suggate rank classification of the coal (Suggate, 2000). According to the ASTM rank classification the coal is high volatile B/A bituminous, which is in agreement with the Suggate rank of 10.4. The R₇₀ values for the three samples taken from the batch ranged from 4.71 - 4.87°C/h. These R₇₀ values show that the coal is very reactive towards oxygen and the coal is classified as having a high spontaneous combustion propensity.

Table 1 - Average analytical data for Spring Creek column sample	
DH851	Main Upper Seam
Coal analysis	
Inherent moisture (% , adb)	3.6
Ash (% , adb)	1.6
Volatile matter (% , adb)	41.7
Fixed carbon (% , adb)	53.1
Sulphur (% , adb)	0.39
Swelling index	3.0
Calorific value (MJ/kg, adb)	32.15
Rank parameters	
VM (% , dmmsf)	43.7
CV (Btu/lb, dmmsf)	14648
Suggate rank	10.4
Romax (%)	0.61
CV (Btu/lb, mmmf)	14006
ASTM rank	hvbB/hvAb

The Spring Creek coal is of similar rank to the Hunter Valley coals mined at Dartbrook, which experienced heating problems on more than one occasion (Moreby, 1997 and 2005). Hence, it displays a similar reactivity to these coals. However, it must also be remembered that the self-heating rate of the coal, and hence its propensity for spontaneous combustion is significantly affected by the presence of moisture in the coal as shown by Beamish and Hamilton (2005). Therefore to gain a better indication of propensity for spontaneous combustion the coal must be tested in an as-mined state, where the moderating effect of moisture in the coal can be taken into consideration. This is achieved with the UQ 2-metre column test.

Hot spot development rates for Spring Creek coal

The hot spot development pattern for the Spring Creek column test is typical of the development of a heating in broken coal with a high as-mined moisture content (Figures 3 and 4). By day 12, a hot spot has been established 145 cm from the air inlet, and moves slightly downwind before the maximum coal temperature plateaus at approximately 90°C for a couple of days (Figure 4). This plateau is in response to the coal needing to discharge adsorption-bound moisture (Evseev and Voroshilov, 1986) to provide access of the air to oxidation sites allowing accelerated heating to develop. The hot spot then begins to migrate towards the air source as the coal dries out on the leading edge and by day 20

a defined hot spot tries to develop 73cm from the air inlet (Figure 3). The hot spot continues to migrate upwind as the coal dries out and eventually reaches a maximum temperature of approximately 160°C just 37cm from the air inlet after 28 days.

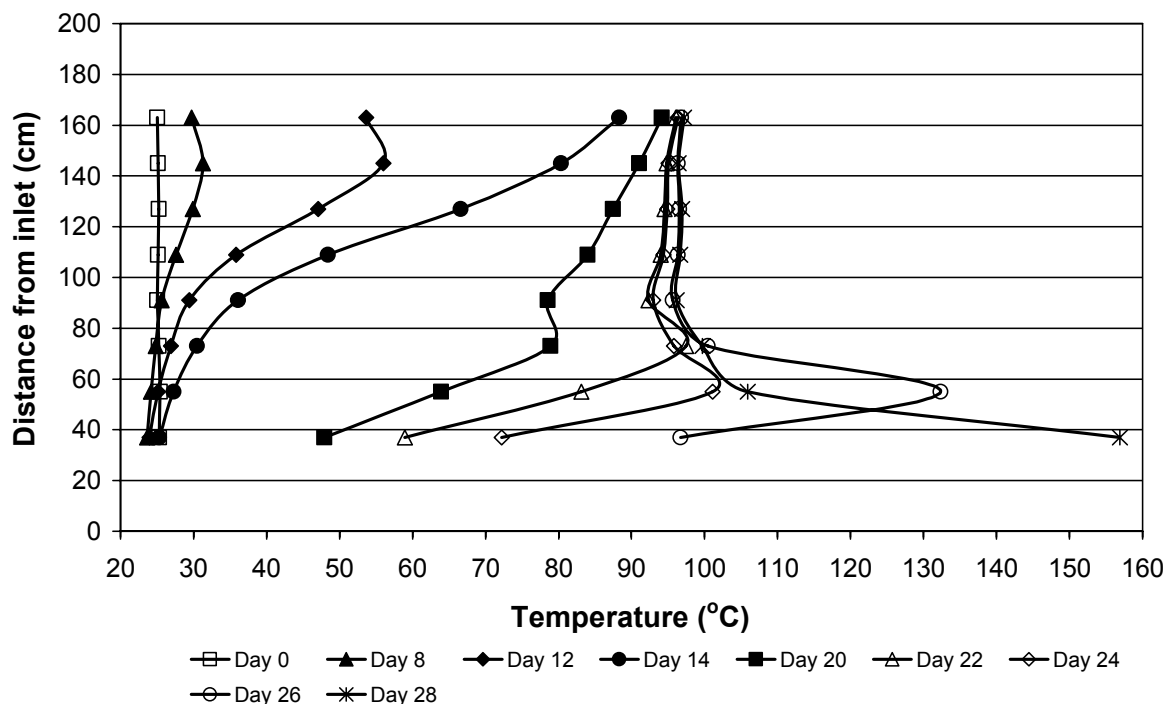


Figure 3 - Self-heating profile showing hot spot development in Spring Creek bulk coal

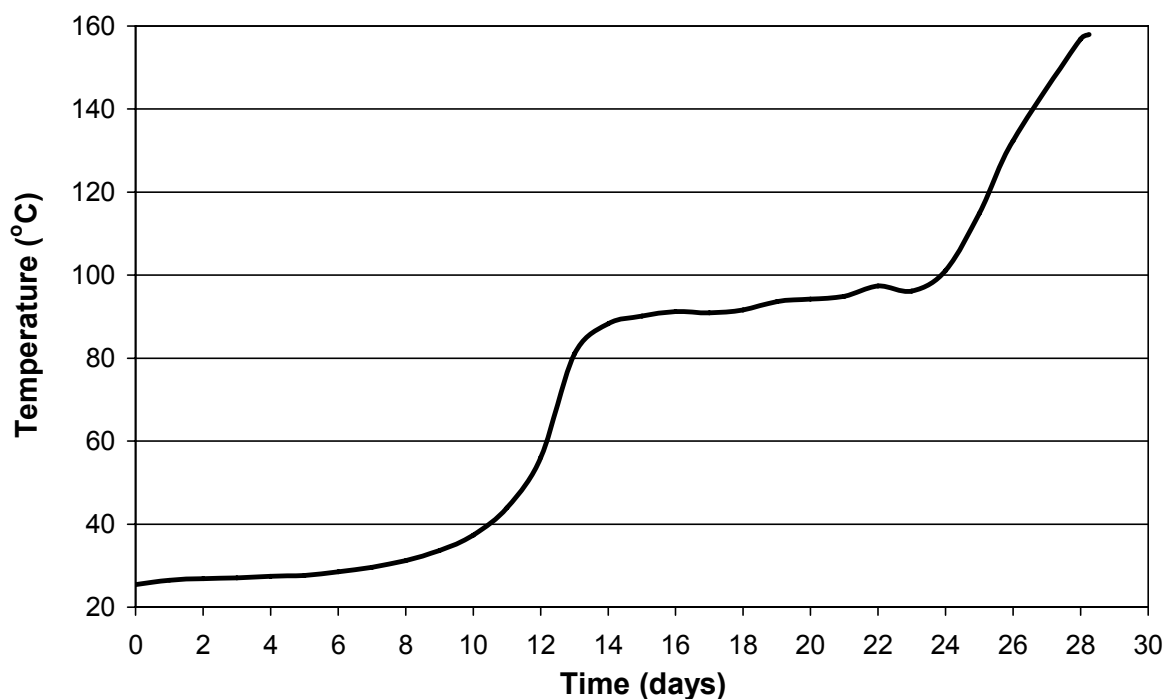


Figure 4 - Maximum self-heating temperature showing stages of hot spot development

For safety reasons, the hot spot is not allowed to progress beyond this point in the column. Instead, the heater at 55 cm from the air inlet was set at 200°C to heat the coal at this point in the column to obtain further gas evolution data. This had the effect of reactivating the hot spot at the 37 cm level in the column and a final maximum temperature of 202°C was achieved, before the test was terminated.

Gas evolution under static conditions at 25°C

The initial equilibration of the coal in the column to the start temperature of the test is equivalent to sealing the coal in a large gas desorption canister. Hence, the results obtained indicate the presence of any seamgas in the coal. The ROM sample still contained a substantial amount of seamgas, which was predominantly methane (8.3 %) with a minor amount of carbon dioxide present (1.7%) at 25°C. Subordinate amounts of ethane (25ppm), carbon monoxide (294 ppm) and hydrogen (58 ppm) were also present. The carbon monoxide and hydrogen can be considered to be products of low temperature coal oxidation. However, the ethane is present as seamgas. The oxygen content in the column had fallen to 5.4%, indicating the coal had used up a substantial amount of the air present, which is consistent with the high reactivity of the coal shown by the R_{70} test results.

Gas evolution during hot spot development

The evolution patterns of the major gases and spontaneous combustion indicator ratios corresponding to the column self-heating are shown in Figure 5. Both methane and carbon dioxide show a gradual decline in concentration during the early part of the test as the coal initially warmed. This is consistent with seamgas desorption. Similarly, ethane disappeared altogether in the early stages of the test. However, both carbon monoxide and hydrogen consistently increased as the temperature of the coal increased even in the early stages of the self-heating.

The carbon monoxide evolution closely tracks the hot spot development and even plateaus in response to the hot spot reaching the stage where moisture needs to be removed from the coal for further hot spot development and migration to take place. Similarly, the Graham's Ratio (carbon monoxide/oxygen deficiency) follows the progress of the heating.

The carbon dioxide evolution also closely tracks the hot spot development once the initial gas desorption phase dissipates. Unfortunately, the carbon monoxide/carbon dioxide ratio appears anomalous in responding to the self-heating, but this is masked by the falling carbon dioxide concentration in the early stages in response to gas desorption.

The hydrogen evolution shows a significant maximum evolution at the point where the hot spot begins to plateau, but then begins to increase again once the hot spot begins to migrate towards the air source. Hitchcock, Cliff and Beamish (2008) have shown that this pattern is consistent with zones of hydrogen evolution in response to initially being generated in an oxygen rich environment followed by generation in an oxygen deficient environment. More detailed studies of this effect are in progress.

The appearance of ethylene is very distinctive and only occurs once the temperature of the hot spot exceeds 90°C. In this case it appears several days after the hot spot reaches the moisture plateau. As such it is a good indicator of advanced hot spot development and clearly if it shows a sustained accelerating trend then the hot spot is rapidly migrating towards the air source at elevated temperatures in excess of 120°C.

Ethane and methane evolution patterns are strongly controlled by gas desorption mechanics. Both show a rapid decline in the early stages of the heating, but at approximately 40°C the methane begins to increase and the presence of ethane is again detected and the concentrations of both gases rise sharply from this point until the moisture plateau is reached. The sympathetic relationship between these two suggest that there is a fundamental physical change in the coal pore structure in this temperature range that reactivates the gas desorption of these two gases. The increasing temperature of course exacerbates this mechanism. There appear to be several cycles of this phenomenon as the hot spot migrates towards the air source. It is interesting to note that on each successive cycle the methane evolution decreases, whereas the ethane evolution increases slightly or reaches a similar maximum level on each cycle. This may be a function of differential desorption between the two gases. Given the characteristic behaviour of these two gases for this particular coal they can be used as early warning indicators of a heating event.

It is also, interesting to note that as the methane evolution is decreasing in the latter half of the column, the hydrogen evolution is increasing. It has been suggested that the hydrogen response may be simply a gas density separation mechanism because the column is operated in a vertical mode. However, due to the fact that methane (another lighter than air gas) is decreasing and carbon dioxide (a heavier than air gas) is increasing, density separation of the gases in these experiments is clearly

not happening and the observed gas evolution is simply responding to the general body airflow through the coal allowing the oxidation reactions to take place.

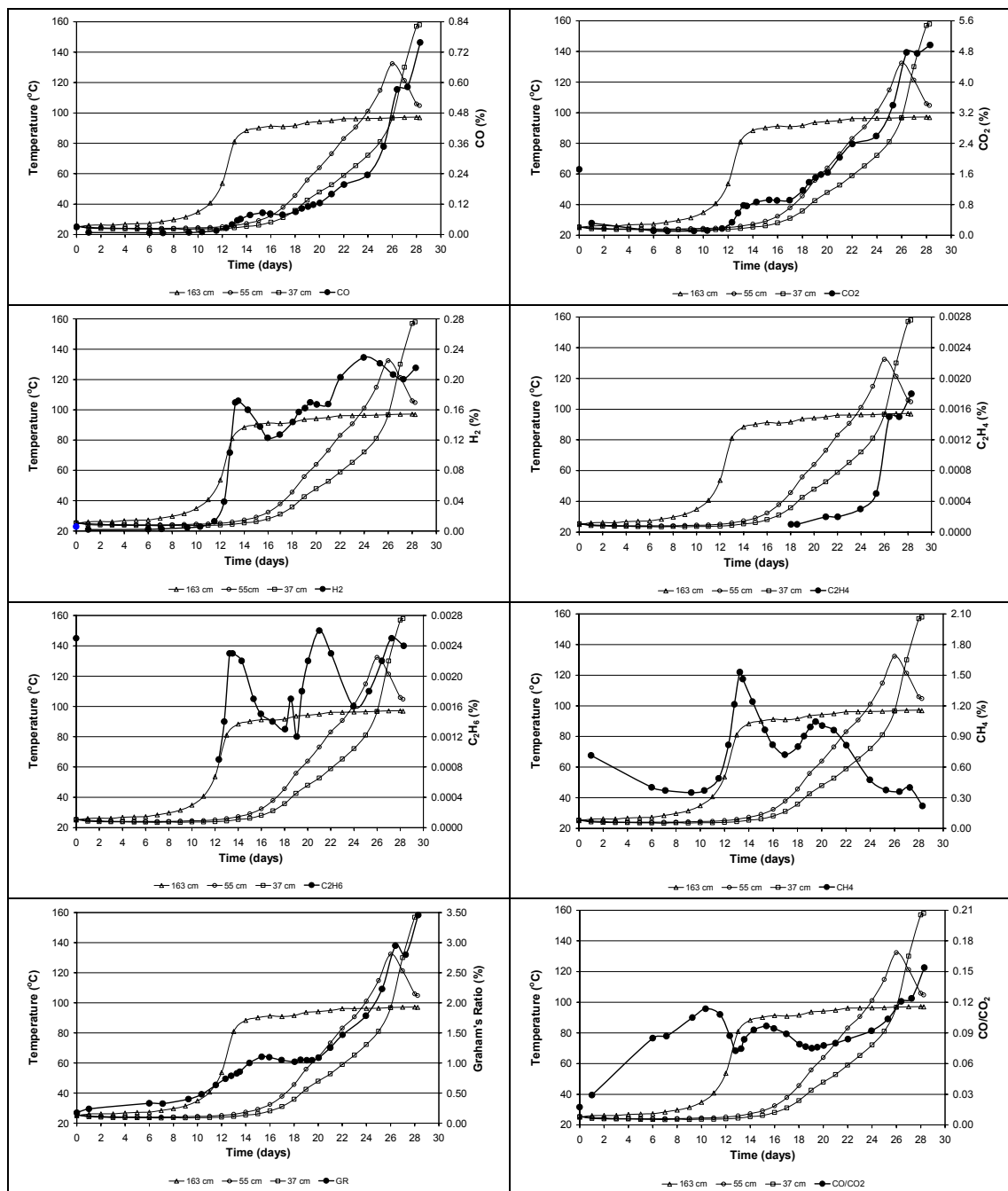


Figure 5 - Gas evolution patterns from Spring Creek bulk coal self-heating

Spring Creek minesite experience

Prior to the commencement of extraction at Spring Creek mine, tubes are set up in the intake (1 point) and the return (2 points) and data collection via Safegas software begins immediately. Gasbags are taken from the extraction section return and the Main return weekly in the initial stages of extraction, then daily as the goaf size increases. The information is used to trend a number of indicators: Graham's Ratio, Jones-Trickett Ratio and CO/CO₂ Ratio. For Spring Creek conditions, CO make is not a good indicator of sponcom and is clearly a function of goaf size with 35 – 40 litres/minute being recorded in previous panels nearing completion. This point is illustrated in Figure 6.

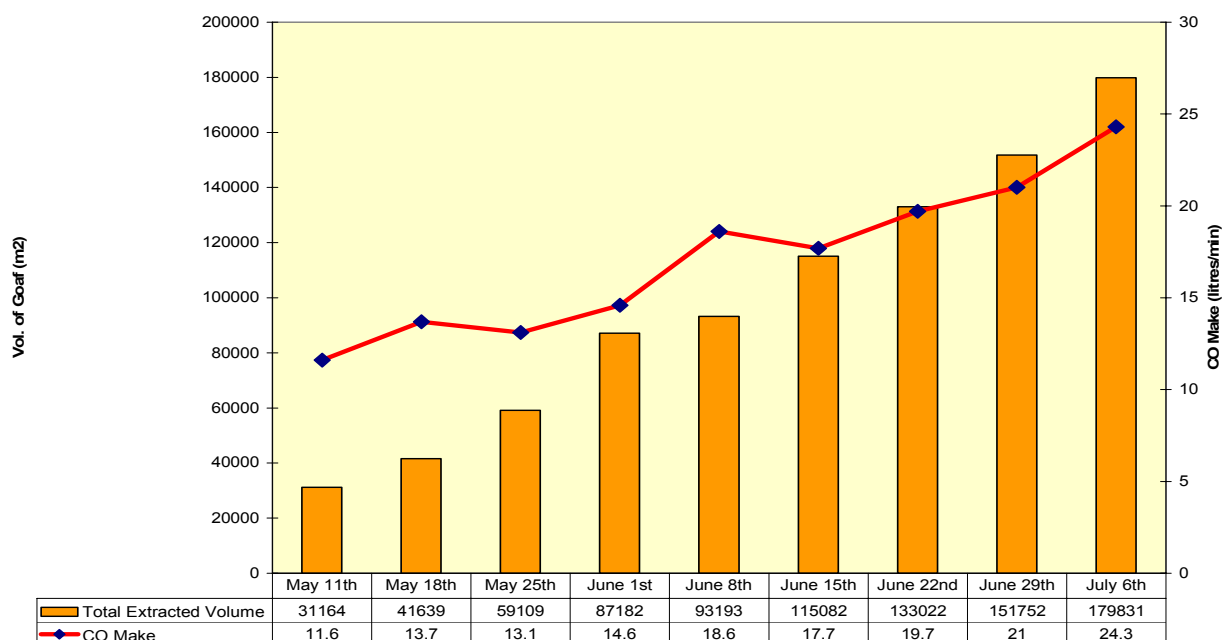


Figure 6 - Comparison of goaf size and CO Make

Information from regular gasbag sampling is compared closely to the results from the 2-metre column test carried out in March 2005. Of particular interest is the appearance of ethane and hydrogen. While ethane is a seamgas, it is undetectable in the gasbag samples at ambient temperatures. The appearance of ethane is coincident with an increase in return air temperature and the 2-metre column test indicates that the coal sample has reached 40°C. Although there are a number of variables that may affect this information, it offers the earliest indicator of a potential heating for Spring Creek and allows time for appropriate intervention. Further, the 2-metre column test has provided certainty around the appearance of ethylene. The ethane and ethylene plots from the July 2008 event are shown in Figure 7. Hydrogen in the gasbag samples under such circumstances is a clear cause for concern and significant acceleration of hydrogen evolution becomes apparent following the appearance of ethane (ie at 40°C) as shown in Figure 8.

The sponcom event in July 2008 occurred 60 days after the commencement of the extraction of old mains that had been developed four years earlier. The event was dealt with initially by reducing the ventilating pressure to minimise the flow of air through the goaf, and by short circuiting the air at the panel entries to allow the goaf area to inertise itself naturally. These actions successfully terminated the event and based on the UQ 2-metre column test this was achieved while the heating was still in the moisture plateau stage, just before it could migrate towards the outer surface of the coal pile where thermal runaway to ignition would have been inevitable.

Mining has now moved to another extraction area and samples are being tested at UQ for gas evolution and hot spot development patterns in combination with a new adiabatic oven testing procedure for as-mined coal. This information will be used to modify interpretation of minesite behaviour due to any changes detected in the coal performance.

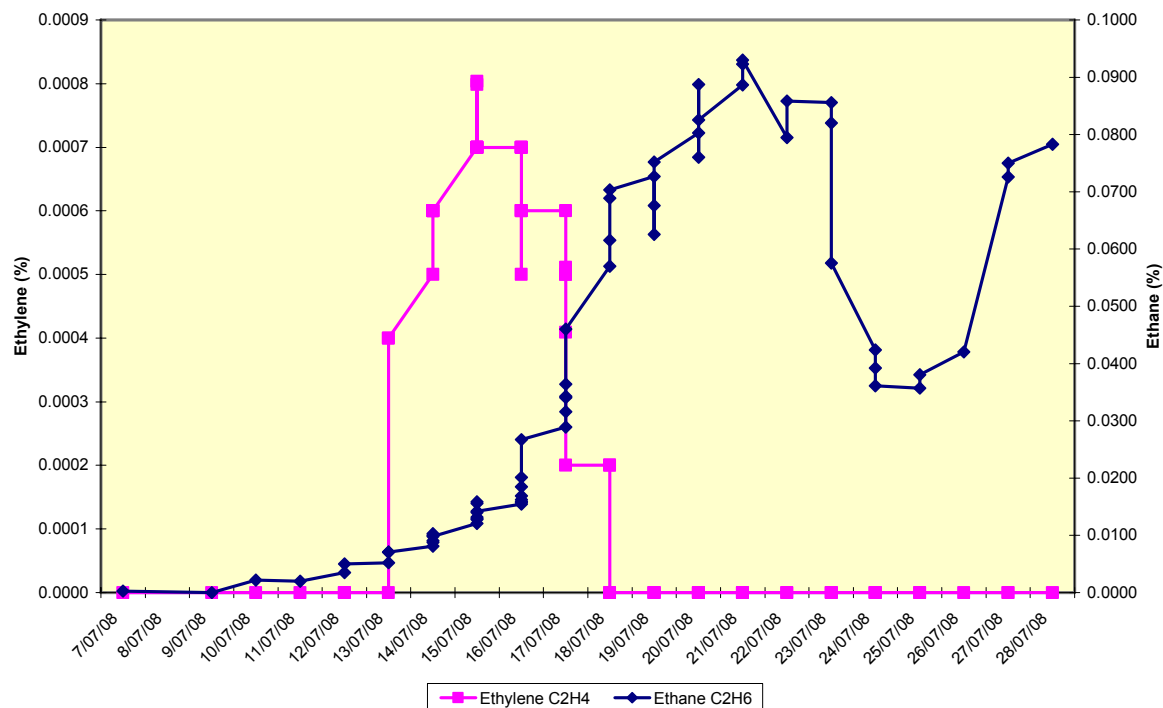


Figure 7 - Ethane and ethylene in static atmosphere throughout the sponcom event

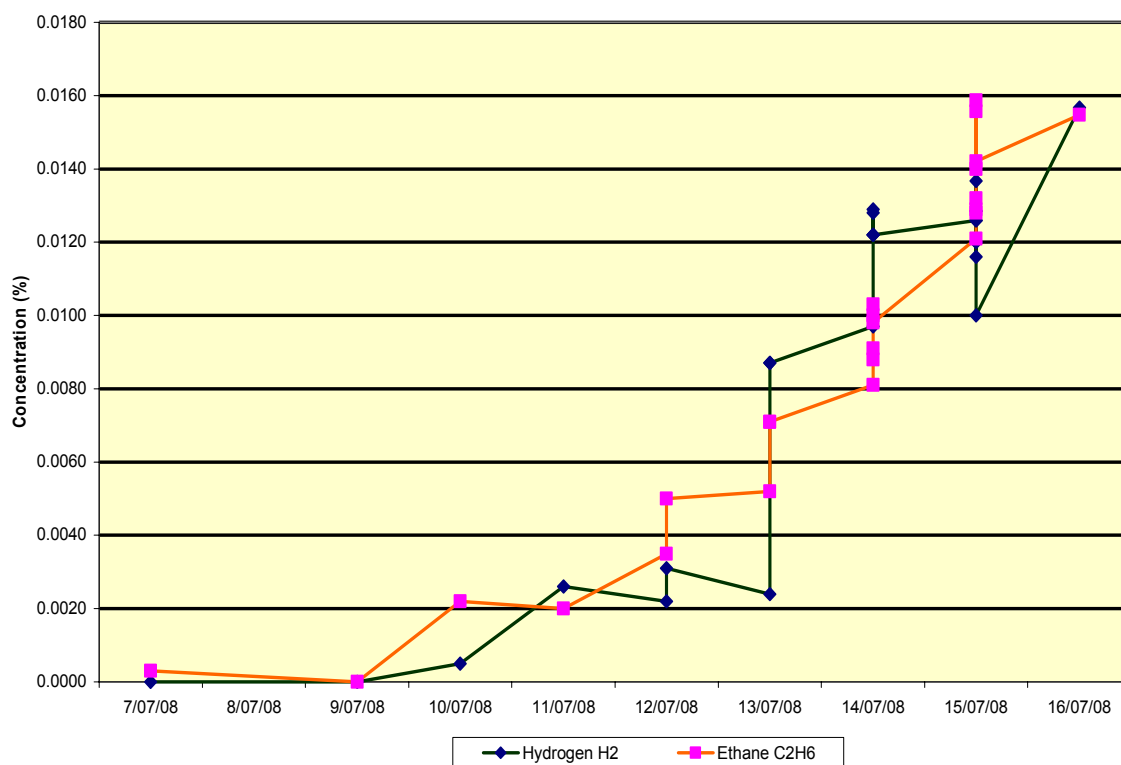


Figure 8 - Hydrogen and ethane evolution during the July 2008 heating at Spring Creek Mine

CONCLUSIONS

The occurrence of spontaneous combustion events in gassy mines often have severe outcomes with major disruption to production and in some cases, loss of life, loss of resource, loss of equipment and the loss of the mine. At Spring Creek the preparation for such an eventuality through detailed risk assessment and well communicated response plans proved to be crucial. An understanding of the intrinsic characteristics of the coal seam was at a level that provided recognition of the onset of

spontaneous combustion at the earliest possible stage of self-heating. This, in turn, allowed for an intervention that resulted in minimal loss of coal (<2000 tonnes) and minimal loss of time to recommence production (16 days). The results from the 2005 UQ 2-metre column test provided management with information that was not obtainable by any other means. This allowed a high level of confidence in the decision to inertise the affected area by natural means.

More detailed laboratory testing of Spring Creek coal is in progress and new results from this work is providing the opportunity for accurate benchmarking to assist the coal industry maintain leading practice in spontaneous combustion management planning.

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LOW TEMPERATURE OXIDATION OF A HIGH VOLATILE BITUMINOUS TURKISH COAL

EFFECTS OF TEMPERATURE AND PARTICLE SIZE

Kemal Baris¹ and Vedat Didari¹

ABSTRACT: Low-temperature oxidation of a high volatile bituminous Turkish coal was studied using an isothermal flow reactor technique under different experimental conditions. Coal samples were ground and sieved to three different particle sizes namely -850 μm , -425 μm and -300 μm . The samples were oxidised at 40, 60 and 90°C with an oxygen flow rate of 45 mL/min. CO₂ and CO emissions were analysed using an HP 5897 Series II type gas chromatograph. Temperature was found to have a definite effect; as the temperature increased rates of formation of CO₂ and CO also increased significantly. Experimental results showed that the rates of formation of CO₂ and CO are independent of the particle size of the samples.

INTRODUCTION

Low-temperature oxidation of coal has been studied by a number of investigators since the reaction with oxygen may lead to self-heating and subsequent spontaneous combustion of coal (Polat and Harris, 1983; Kaji, Hishinuma and Yoichi, 1985; Itay, Hill and Glasser, 1989; Clemens, Matheson and Rogers, 1990; Krishnaswamy et al., 1996; Wang, Dlugogorski and Kennedy, 1999, 2002, 2003a, 2003b). Along with the safety problems the oxidation of coal also affects the molecular structure of coal and this causes negative effects on the mass and elemental composition of coal.

The spontaneous combustion of coal causes serious problems in the coal industry during mining, transportation, storage and treatment. It is reported that every year between 100 and 200 Mt of high quality coal is consumed by spontaneous combustion in China (http://www.itc.nl/~coalfire/problem/china_coalfire.html). Six spontaneous combustion events were reported in Karadon Colliery of Turkish Hardcoal Enterprise in the Zonguldak Basin between 1990 and 2000. Since has the potential to cause loss of property and life, special attention has been being paid to spontaneous combustion of coal.

Wang, Dlugogorski and Kennedy (2003a) stated that coal oxidation at low temperatures (i.e. below 100°C) is a complicated process and involves four phenomena: oxygen transport to the surfaces of coal particles, chemical interaction between coal and O₂, release of heat and emission of gaseous products. The diversity of chemical composition, physical properties (such as heat capacity and thermal conductivity) and porous structure of coal enhances the complexity of this phenomenon.

The reaction between coal and oxygen at low temperature depends on many factors including temperature, particle size, surface area, coal pore structure, moisture content, coal rank and the composition of air. This paper presents results of the effects of temperature and particle size on low-temperature oxidation of a high volatile Turkish bituminous coal by examining the characteristics of gaseous products, CO₂ and CO in particular, emitted during the oxidation process.

EXPERIMENTAL

A high volatile bituminous coal was obtained from Armutcuk Colliery operated by Turkish Hardcoal Enterprise, Zonguldak Basin, Turkey. Proximate, ultimate and petrographic analysis of the sample is presented in Table 1. Coal samples were crushed, ground in a sealed ball mill and then sieved to three different particle sizes of -850 μm , -425 μm and -300 μm (sieving was performed in a sealed glove box to minimise the contact of particles with air). Particle size distributions were determined using a Malvern Laser Beam Particle Sizer. Figure 1 shows the particle size distribution of the coal samples. d₅₀ and d₈₀ values are 0.389 and 0.512 mm for the -850 μm , 0.297 and 0.215 mm for the -425 μm and 0.112 and 0.179 mm for the -300 μm particle size, respectively. After comminution and

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sieving 125 g of sample from each size group were weighed and put in glass jars in a sealed glove box under nitrogen flow and then evacuated to be used in oxidation tests.

Table 1 - Proximate, ultimate and petrographic analysis of the coal sample

Proximate analysis (dry basis)			Ultimate analysis (dry ash free basis)		
Ash (%)		5.1	C (%)		81.7
Volatile Matter (%)		37.3	H (%)		5.0
Fixed Carbon (%)		57.6	N (%)		1.3
Total Sulphur (%)		0.5	S (%)		0.93
Calorific Value (kcal)		7606	O (%) by diff.		7.3
Petrographic composition					Reflectance (R _m) (%)
Vitrinite (%)	Liptinite (%)	Semi-Inertinite (%)	Inertinite (%)	Mineral Matter (%)	0.768
52.2	8.4	15.8	23.0	0.6	

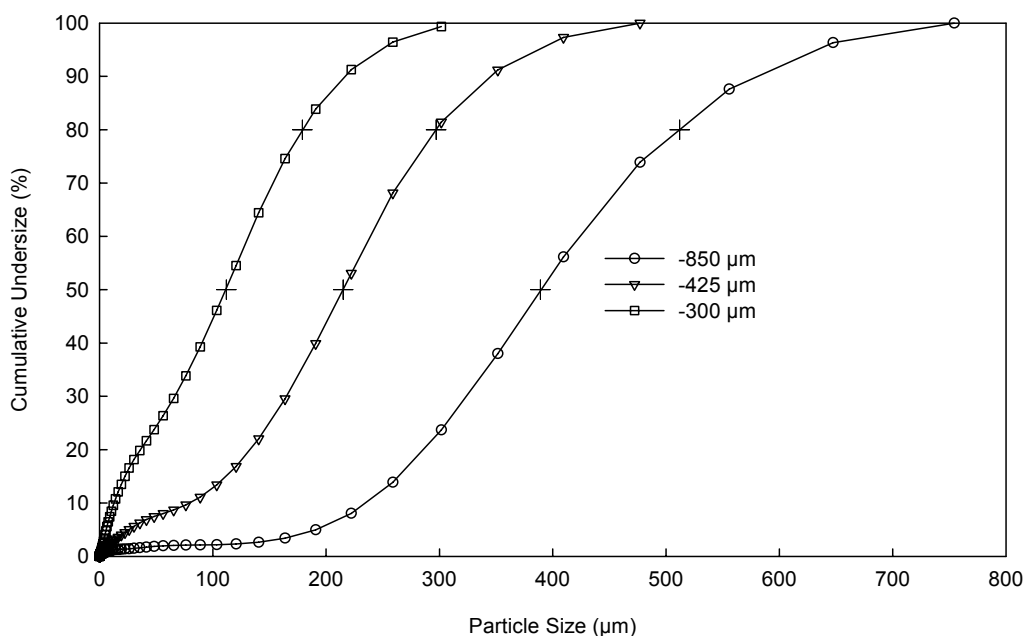


Figure 1 - Particle size distribution of coal samples

An isothermal flow reactor technique was used for the oxidation tests. Figure 2 shows the detail of the setup used in the experiments. Before starting an experiment, 125 g of sample was placed into the reactor in a sealed glove box and dried by purging ultra-high-purity nitrogen under 65°C for 15 hours. After drying the oven was set to the desired temperature. As soon as the temperature of the coal sample in the reactor and the temperature of the oven equilibrated a gas stream of oxygen (99.6 % purity) was introduced to the system at a flow rate of 45 ± 0.5 mL/min. In order to monitor gas emissions (CO_2 , CO and hydrocarbons) an HP 5897 Series II type gas chromatograph equipped with a flame ionization detector (FID) and a thermal conductivity detector (TCD) was used. For each oxidation test 15 consecutive analyses were performed by using GC. Total oxidation period was 337.5 minutes. The variation of oxygen concentration was monitored by a Rapidox 3100 oxygen analyser.

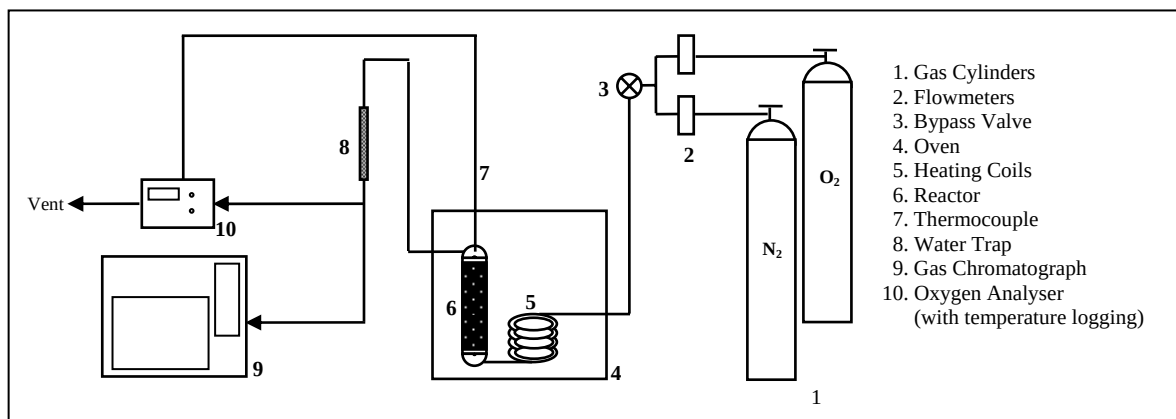


Figure 2 - Experimental setup used for oxidation tests

The oxidation tests were repeated for each size by modifying the heating temperature to investigate this effect on oxidation of coal at low temperatures. The temperature of coal samples was monitored for each oxidation test. Table 2 shows the experimental conditions for each size.

Table 2 - Experimental conditions for each size fraction tested.

Size (μm)	Flow rate (ml/min)	Temperature ($^{\circ}\text{C}$)
-850	45	40
		60
		90
-425	45	40
		60
		90
-300	45	40
		60
		90

The rates of formation of CO_2 and CO were calculated using Equations (1) and (2):

$$R_{\text{CO}_2} = \frac{C_{\text{CO}_2,o}}{W} V_{\text{gas}} \quad (1)$$

$$R_{\text{CO}} = \frac{C_{\text{CO},o}}{W} V_{\text{gas}} \quad (2)$$

where $C_{\text{CO}_2,o}$ and $C_{\text{CO},o}$ are concentrations of CO_2 and CO at the reactor exit, V_{gas} denotes the flow rate of the gas stream and W represents the dry mass of coal sample. W was measured after each experiment and it was observed that the increase in the mass of coal sample is almost negligible.

RESULTS AND DISCUSSION

Typical experimental results showing the time-dependent rates of CO_2 and CO formation are presented in Figure 3. Note that the first experimental data were removed from the graphs since it is considered that dilution of nitrogen remaining in the reactor could mask any trends in the formation of carbon oxides.

It is observed that the rate of formation of CO_2 decreases gradually with time. Rate of production of CO also exhibits a similar behaviour. It is known that the reason for the decay in formation rate of CO and CO_2 is because of the gradual accumulation of stable complexes at the surface of coal pores

during coal oxidation as stated in the literature (Carpenter and Giddings, 1964). In their studies Wang, Dlugogorski and Kennedy (2002) also argued that especially in the first few hours of experiments, rapid decrease in formation rates may be due to distinct reaction pathways for the liberation of these products. Although the observed decrease in the production rates of CO_2 and CO are not so sharp a substantial decrease shown in Figure 3 supports this argument.

Temperature has a significant effect on the production rates of CO_2 and CO . As the temperature increases the level of emission of CO_2 and CO also increases. Figure 4 shows the effect of temperature on low-temperature oxidation of the coal sample. Although the formation rate of carbon oxides increase with time for a few experimental results the rates in general exhibit a decreasing trend for most of the experimental results. Even if the formation trend is increasing with time oxidation tests performed at higher temperatures always result in higher level of emissions of CO_2 and CO . This result is in agreement with other experimental findings (Carpenter and Giddings, 1964; Kaji, Hishinuma and Yoichi, 1985; Wang, Dlugogorski and Kennedy, 2003b).

Since the heating temperatures were rather low the temperature of coal samples showed rather a quick increase at the beginning of each test due to the rapid oxidation, then temperature rise had almost a constant rate.

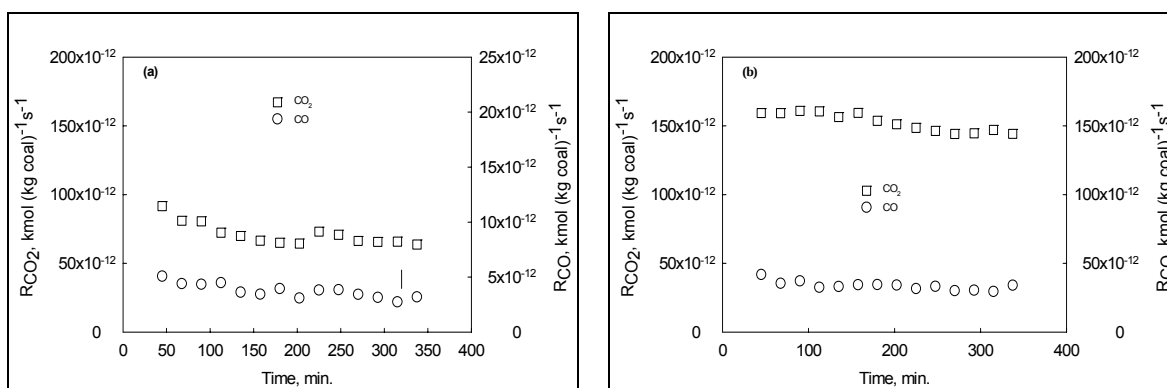


Figure 3 - Variation in the rates of production of CO_2 and CO (a) with a particle size of $-300 \mu\text{m}$ at 60°C (b) with a particle size of $-425 \mu\text{m}$ at 90°C

Early investigators reported that CO is the main product of coal oxidation at low temperatures (i.e. below 70°C). However, present results do not support these observations. It is clear that the formation rate of CO_2 is always much higher than that of CO in all experimental conditions even at 40°C . It seems that CO_2 is the major product of oxidation at low temperatures. This argument is supported by a number of researchers (Carpenter and Giddings, 1964; Kaji, Hishinuma and Yoichi, 1985; Wang, Dlugogorski and Kennedy, 2002).

The experimental results show that the production rates of CO_2 and CO exhibits no dependence on the coal particle size although a few experimental results seems to show a dependence on the coal particle size. Figure 5 illustrates a typical trend for the effect of particle size on low temperature oxidation of the coal sample. As seen in Figure 5, the rate of formation of CO_2 does not seem to have a dependence on the particle size at 60 and 90°C although there seems particle size dependence at 40°C with some scatter in the experimental data. However, there is no obvious dependence on particle size for the rate of formation of CO in all experimental conditions. These results may indicate that the production of carbon oxides is not controlled by mass transport of these species in coal pores but it is controlled by chemical reactions as confirmed by several researchers (Kaji, Hishinuma and Yoichi, 1985; Wang, Dlugogorski and Kennedy, 2002).

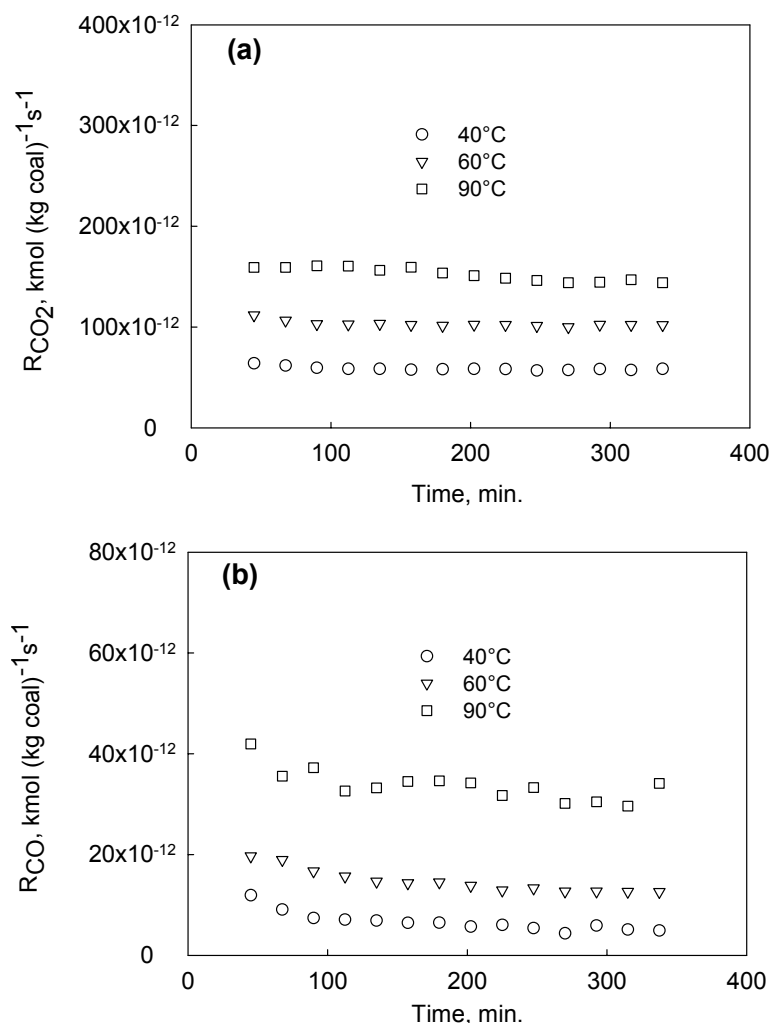


Figure 4 - The effect of temperature on the formation rate of (a) CO₂ and (b) CO with a particle size of -425 μm

CONCLUSIONS

The effects of temperature and particle size on the low-temperature oxidation of a high-volatile Turkish bituminous coal were examined at 40, 60 and 90°C and for the particle sizes of -850, -425 and -300 μm by measuring CO and CO₂ formation rates. It is observed that CO₂ is the major product of oxidation at low temperatures rather than CO. The formation rate of CO₂ is always much higher than that of CO in all experimental conditions. The formation rates of CO₂ and CO decrease with time though there was some scatter evident the experimental data. This may be attributed to deactivation of active sites in coal structure. Temperature has a pronounced effect on the rates of formation of CO₂ and CO. When the temperature is increased the production rate of carbon oxides also increases. Furthermore, the formation rate of CO₂ and CO is independent of particle size which may indicate that the production of CO₂ and CO is controlled by chemical reaction rather than mass transport in coal pores.

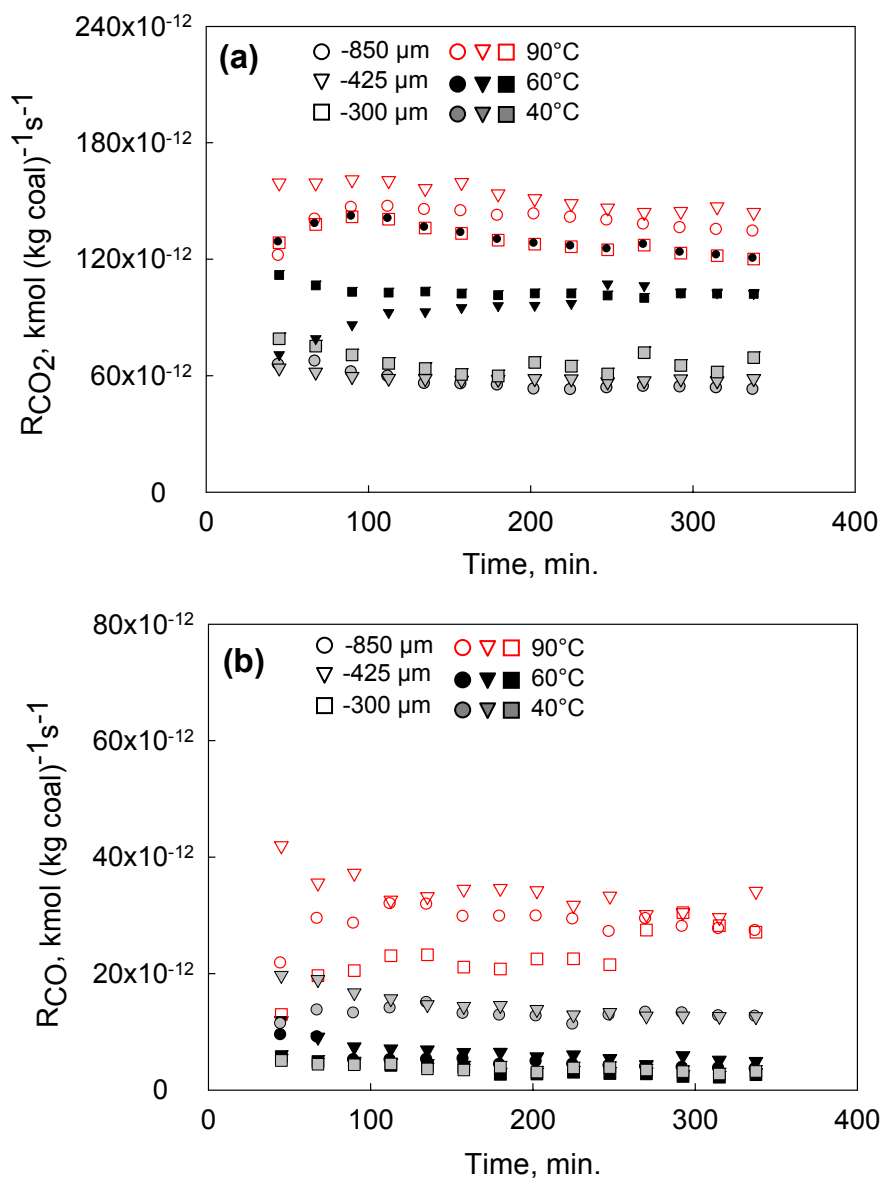


Figure 5 - The effect of particle size on the production rates of (a) CO₂ and (b) CO at 40, 60 and 90°C

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ASSESSMENT OF SPONTANEOUS HEATING OF COAL BY DIFFERENTIAL SCANNING CALORIMETRIC TECHNIQUE – AN OVERVIEW

Niroj Mohalik¹, Durga Panigrahi¹, Virendra Singh¹ and Ran Singh¹

ABSTRACT: Differential Scanning Calorimetric (DSC) applications in coal sciences have expanded rapidly. Various researchers observed that some of the methods to determine spontaneous heating are time consuming, tedious and do not give reproducible results. DSC instruments are usually employed under identical experimental conditions to track the reaction type (i.e., endothermic or exothermic), to calculate kinetics and heat flow rates. Application of DSC for determination of spontaneous heating of coal and suitability in this area is described.

INTRODUCTION

The spontaneous heating resulting into mine fire is an inherent problem in coal mining industry, which endangers lives of men in a mine. It also causes serious environmental pollution and economic losses to industry. Therefore, determination of susceptibility potential of coal due to spontaneous heating and their classification are essential to plan the production activities and storage capabilities in a coal mine. Different methods of spontaneous heating susceptibility can be broadly grouped under three headings; examination of chemical constituents of coal, oxygen avidity studies and thermal studies. In chemical composition of coal, attempts have been made to determine the spontaneous heating tendencies of coal based on their constituents obtained from proximate and ultimate analyses. The maceral composition of coals and their susceptibility to spontaneous heating have led to the development of petrological classifications. The oxygen avidity studies include; proxy complex analysis rate study, Russian U-index and other oxidation methods. In thermal studies, different methods are attempted, which include; initial temperature, crossing and ignition point temperature, modified crossing point temperature, puff temperature, Olpinski index, adiabatic calorimetry, thermo-gravimetric (TG) analysis, differential thermal analysis (DTA) and differential scanning calorimetry. Spontaneous combustion of coal is influenced by the nature of the coal, particle size, geological condition and mining environment, all of which govern the thermal processes occurring in the coal. However, generalization of the results on the thermal behaviour of coal has been difficult. In recent years the application of thermal analysis techniques to the study of combustion pyrolysis behaviour and kinetics of coals has gained a wide acceptance among various research workers.

LITERATURE REVIEW

The earliest study on spontaneous combustion was carried out by Mahajan, Tomita and Walker (1976), using DSC technique. They reported differential scanning calorimetry curves for 12 coals of various ranks in a helium atmosphere with a flow rate of 1 ml min⁻¹ and temperatures between 100 to 580 °C at a constant heating rate of 10 °C min⁻¹. The amount of sample used in the study varied between 12 and 20 mg with reference material being alumina. They concluded that the thermal effects of coals, ranging in rank from anthracite to bituminous, were endothermic. Exothermic heats were observed only in the case of sub-bituminous coals or lignite. The net thermal effects were found to be strongly rank dependent. Rai and Tran (1977) conducted a kinetic study on catalyzed and non-catalyzed coal. In their kinetic model, the apparent activation energy was measured to be a rectilinear function, describing the extent of the pyrolysis reaction of the of Hanna coal. The order of reaction was found to be about 0.3 for the pyrolysis step and 0.67 for the hydro-gasification step.

Gold (1980) demonstrated the occurrence of exothermic processes associated with the production of volatile matter in or near the plastic region of the coal samples studied. He observed that the temperature and magnitude of the exothermic peak were strongly affected by the heating rate, sample mass, and particle size. Rosenvold, Bubow and rajeshwar (1982) analysed 21 bituminous coal samples from Ohio by DSC and non-isothermal thermogravimetric. Three regions of endothermic

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activity were distinguished in DSC scans in an inert atmosphere. The first peak (25–150 °C) corresponded to devolatilisation of organic matter, and a partially resolved endotherm occurred at temperatures above 550 °C probably correspond to cracking and coking processes subsequent to pyrolysis step.

Rajeshwar (1983) applied differential scanning calorimetry and thermogravimetry to study of coal, oil shale, and oil sands. DSC has been used to characterise 12 U.S. coals of varying rank from anthracite to lignite. Elder and Harris (1984) investigated the thermal characteristics of Kentucky bituminous coals undergoing pyrolysis in an inert atmosphere at three different heating rates and determined the specific heats of the coals. The specific heats of the dry coals lie in the range of 1.21 to 1.47 Jg⁻¹K⁻¹. The exothermic heat flow from 300 to 550 °C, where the major weight loss occurred, has been associated with the primary carbonisation process, the development of the plastic state, and the onset of secondary gasification, which is responsible for coke formation. Ismail and Walker (1989) studied oxygen chemisorption on a number of coal chars in oxygen atmosphere at 100 °C by using DSC and TGA. They studied 16 coal samples having particle size of -100 mesh. The samples were heated in nitrogen atmosphere with a flow rate of 45 ml min⁻¹ between ambient to 600 °C. They correlated heat release and weight of oxygen chemisorbed as a function of reaction time with Elovich plots. They observed that the rate of heat release and weight increase generally decreased with increasing rank of coal. They also observed that the rate of heat release upon oxygen chemisorption generally increased as rates of gasification of chars at higher temperatures in air increased.

Alula, Cagniant and Laver (1990) used thermo-gravimetric and differential scanning calorimetry to characterise low and high-temperature coal tar and petroleum pitches and their fractions, thermal methods to the characterization of pyrolysis coal products. Kok (1997) investigated thermal behaviour of lignite by DSC, TG/DTG, High Pressure Thermo Grvimetric (HPTGA), and combustion cell experiments. He studied one coal sample of particle size of – 60 mesh in air with a flow rate of 50 ml min⁻¹. Different heating rates were applied, i.e. 5, 10, 15, 20 and 25 °C min⁻¹ in a temperature range between 20 to 600 °C during experimentation. He used different models (Arrhenius, Coats and Redfern, Fassihi and Brigham) to obtain kinetic parameters. The heat flow rate recorded at different temperatures showed that oxidation reaction started around 250 °C and reached a maximum rate at 410 °C. Higher heating rates resulted in higher reaction temperatures and heat of reactions. Distinguishing peaks in the DSC curves shifted to higher temperatures with an increase in heating rate.

Garcia, Halla and Fanoor (1999) found that differential scanning calorimetry is a useful technique to investigate early stages of the oxidation of coal. Experiments were carried out on three coal samples of 10 mg each in oxygen atmosphere, at flow rate of 20 ml min⁻¹, heating rate 10 °C min⁻¹, temperature range ambient to 600 °C, and particle size of – 100 mesh. They proposed that the onset temperature was a better indicator of propensity of coals to oxidation and this parameter agreed with rank of coals investigated and increased with time of oxidative weathering. Ozbas, Kok and Hieyilmaz (2003) determined combustion behaviour and kinetic analysis of raw and cleaned coal samples of different sized fractions by using DSC. They studied four coal samples of 10 mg each. Experiments were carried out in air atmosphere with a flow rate of 50 ml min⁻¹, at the heating rate of 10 °C min⁻¹ and temperature range between 20 and 600 °C. DSC curves of three coal samples showed two reaction stages. The first stage of reaction was due to moisture loss (endothermic) and observed in temperature range of ambient to 150 °C. The second stage was exothermic region due to combustion and observed in the temperature range of 150 to 600 °C. Kinetic parameters of the samples were determined using Roger and Morris kinetic model. Panigrahi and Sahu (2004) used DSC, DTA, wet oxidation potential, crossing point temperature for determination of susceptibility of 31 Indian coals to spontaneous heating. They used 10 mg coal sample of particle size -72 mesh, , in oxygen atmosphere with a flow rate of 20 ml min⁻¹, the heating rate was at 30 °C min⁻¹, between ambient to 500 °C and with alumina as reference material.

Exhaustive correlation studies between susceptibility indices and intrinsic properties were carried out for identifying the appropriate parameter to be used for classifications. The identified parameters were used as inputs to ANN. Adoptive resonance theory of Artificial Neural Network ANN has been applied to classify coal seams into four different categories. Kok (2005) used differential scanning calorimetry and thermogravimetry to obtain information on temperature-controlled combustion characteristics of 17 coals of different origin from Thrace basin of Turkey. Using DSC experiments were performed on 10 mg coals, in air atmosphere with a flow rate of 50 ml min⁻¹, particle size of 60 mesh, ,temperature range between 20 to 600 °C at a heating rate of 10 °C min⁻¹ by. Kot calculated the kinetic parameters from Arrhenius and Coats–Redfern plots. The study revealed that two temperature regions of evident

The chemical reactivity, elimination of water and primary carbonization were also evident in all of the coal samples. It was observed that activation energies of samples were varied from 54 to 92 kJ mol⁻¹. Elbeyli and Piskin (2006) determined thermal characteristics and kinetic parameters of cleaned Tunçbilek lignite by using differential thermal analysis (DTA) and thermogravimetry (TG) DSC thermal analysis system both for combustion and pyrolysis reactions. The experiments were carried out in a 10 mg lignite sample of particle size of - 65 mesh. The tests were carried out in air/nitrogen atmospheres with a flow rate of 100 ml min⁻¹, heating rate 10 °C min⁻¹ up to 1000 °C.

Krzesinska *et al.* (2008) studied three Polish coals of varying rank (82.7, 86.2 and 88.7 wt.% carbon content) and caking ability (weak, moderate and strong) of the Krupinski, Szczygłowice and Zofiowka mines, respectively by TG, DSC and dynamic mechanical analysis (DMA) method. The amount of sample used in their experiment was 7 mg and sample was heated at 10 °C min⁻¹ up to 520 °C in nitrogen atmosphere with a flow rate of 50 ml min⁻¹. The weight loss and heat flow during pyrolysis, and storage/loss elastic modulus measured as a function of increasing temperature were related to the caking ability of coals. Parameters determined with the TG and the DSC methods in the binary and ternary blends were correlated with the proportion of strongly-caking-coal concentration in the blend. The weight loss of coal blends was found to be additive parameter. The DSC thermograms of binary blends were found to be different from those of the ternary blends, which suggested a different course for this blend pyrolysis.

A critical study of the work carried out by different researchers indicates that the different experimental conditions for DSC have been adopted by them for studying the spontaneous heating susceptibility of coal. In order to draw a suitable conclusion from their studies the experimental parameters used by them have been compiled in tabular format and presented in Table 1, which clearly reveals the followings:

- The particle size of coal samples varies from – 60 mesh to - 100 mesh.
- Variation of heating rate was between 5 °C min⁻¹ to 30 °C min⁻¹
- The sample was studied in nitrogen, atmospheric air and oxygen.
- The amount of sample used by them varies from 7 mg to 20 mg.
- The range of flow rate used by them was between 1 ml min⁻¹ to 100 ml min⁻¹.
- The reference material was alumina.
- The number of sample analysed by them was one to four except in four cases, i.e. 12, 16, 17 and 31 samples.
- The range of temperature was ambient to 1000 °C.

Thus, there is no unanimity on the experimental parameters used by different researchers. If the experimental parameters will be different, the results of two samples analysed by two experimental conditions will not be comparable for indexing the coal with respect to their spontaneous heating. Therefore, the future direction of research should be to finalise the experimental standards for this techniques.

DIFFERENTIAL SCANNING CALORIMETRY

a) Principles

Differential scanning calorimetry is a technique in which the difference in energy is put into a substance and a reference material is measured as a function of temperature while the substance and reference materials are subjected to a controlled temperature program. In this technique the ordinate value of an output curve at any given temperature is directly proportional to the differential heat flow between a sample and reference material and in which the area under the measured curve is directly proportional to the total differential calorific input. By this technique, coal samples can be studied under experimental conditions that simulate spontaneous heating process of materials.

b) Experimental Set-up

The complete experimental set up comprises the differential scanning calorimeter, sample holder, crucible sealing press (crimper), purge gas supply arrangement, a computer with software and a graphic plotter (Figure 1). In Mettler-Toledo DSC-821e differential scanning calorimeter sensors are single junction thermocouples. Any energy difference in the thermocouples to the sample and the reference is then recorded against the program temperature. Thermal events in the sample thus

appear as deviations from the DSC baseline, in either endothermic or exothermic direction, depending upon whether more or less energy difference to the sample relative to the reference material. In DSC therefore, endothermic responses are usually represented as being negative, i.e. below the baseline, corresponding to an increased transfer of heat to the samples compared to the reference. Purge gas is supplied from a cylinder equipped with suitable regulators. All the operations of the DSC-821e calorimeter are controlled from the personal computer through software STAR[®]. The software performs all the controls, calibration, data display, standard calculations, curve comparison and calculations etc.



Figure 1 - Experimental setup of Differential Scanning Calorimetry

c) Sample Preparation

Coal samples were collected from different Indian coalfields covering both fiery and non-fiery coal seams of different ranks. Approximately 5 kg of coal samples was collected by channel and chip sampling method and stored in plastic containers with de-aerated water. Before the experiments, representative samples were prepared from the coals obtained by applying closed circuit crushing and screening. The prepared samples were taken for thermal analysis.

d) Experimental Procedure

DSC apparatus was calibrated via the melting points of indium, zinc, lead metals under the same conditions as for the sample. During the experiments the heat flow as a function of time and temperature were recorded, while the sample was subjected to a computer controlled temperature programme. Sample was placed in the sample pan, covered with a lid and sealed with pressure using a crimper press. The crimped container with the sample was put on the sample furnace while the reference furnace was kept empty. Pinholes were made in the lid. The coal was in a monolayer at the base of the pan to ensure that the entire sample made good thermal contact with the bottom part of the pan and therefore good heat transfer with the sensors of the equipment. If the pinholes were too small oxygen diffused into the pans too slowly and this process determined the rate of oxidation, rather than diffusion into the pore structure of the coal itself. A number of experiments were carried out in the temperature range from ambient (30°C) to 600°C with varying atmosphere (Figure 2 to 5). The experiments were repeated under identical conditions to check the reproducibility of the results.

Table 1 - Experimental parameters used by different researchers in DSC studies on spontaneous heating of coal

Sl. No.	Name of the author	Year	Parameters							
			Particle Size (mesh)	Heating Rate ($^{\circ}\text{Cmin}^{-1}$)	Atmosphere	Sample Amount (mg)	Flow Rate (ml min^{-1})	No. of Sample Studied	Reference Material	Temperature Range ($^{\circ}\text{C}$)
1.	Mahajan, Tomita and Walker (1976)	1977	-	10	Helium	20, 12	1	12	Alumina	100 to 580
2.	Ismail and Walker (1989)	1989	- 100	10	N ₂		45	16		Ambient to 600
3.	Kok (1997)	1997	- 60	5, 10, 15, 20, 25	Air	-	50	1	-	20 to 600
4.	Garcia, Halla and Fanoor (1999)	1999	- 100	10	Oxygen	10	20	3	-	Ambient to 600
5.	Ozbas, Kok and Hieyilmaz (2003)	2003		10	Air	10	50	4	-	20 to 600
6.	Panigrahi and Sahu (2004)	2004	- 72	30	Oxygen	10	20	31	Alumina	Ambient to 500
7.	Kok (2005)	2005	- 60	10	Air	10	50	17	-	20 to 600
8.	Elbeyli and Piskin (2006)	2006	- 60	10	Air Nitrogen	10	100	1	-	Ambient to 1000
9.	Krzesinska <i>et al.</i> (2008)	2008	-	10	Nitrogen	7	50	3	-	Ambient to 520

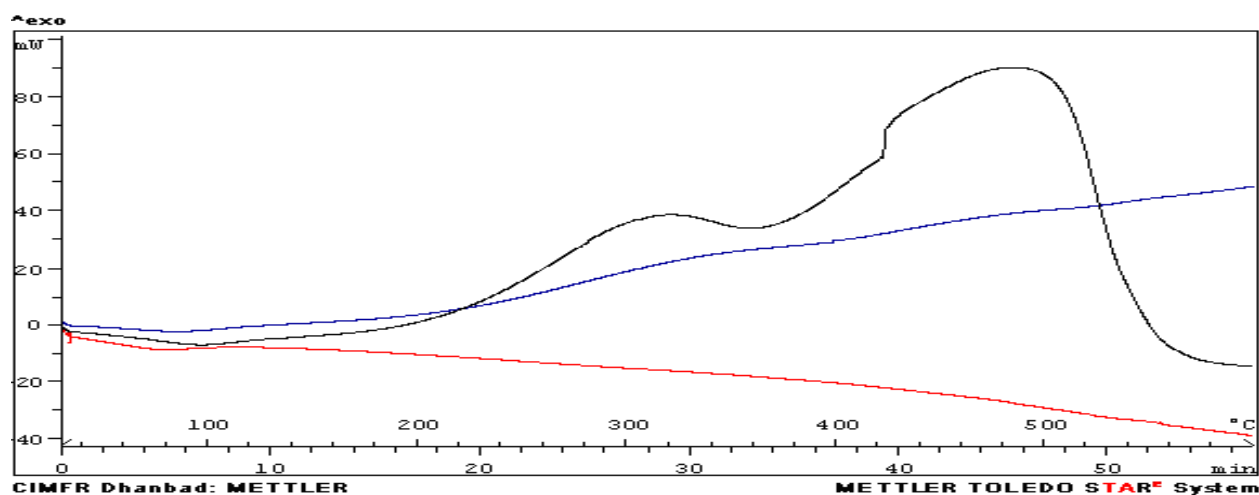


Figure 2 - DSC Thermo-gram of coal sample (Oxygen, Air and Nitrogen Atmosphere)

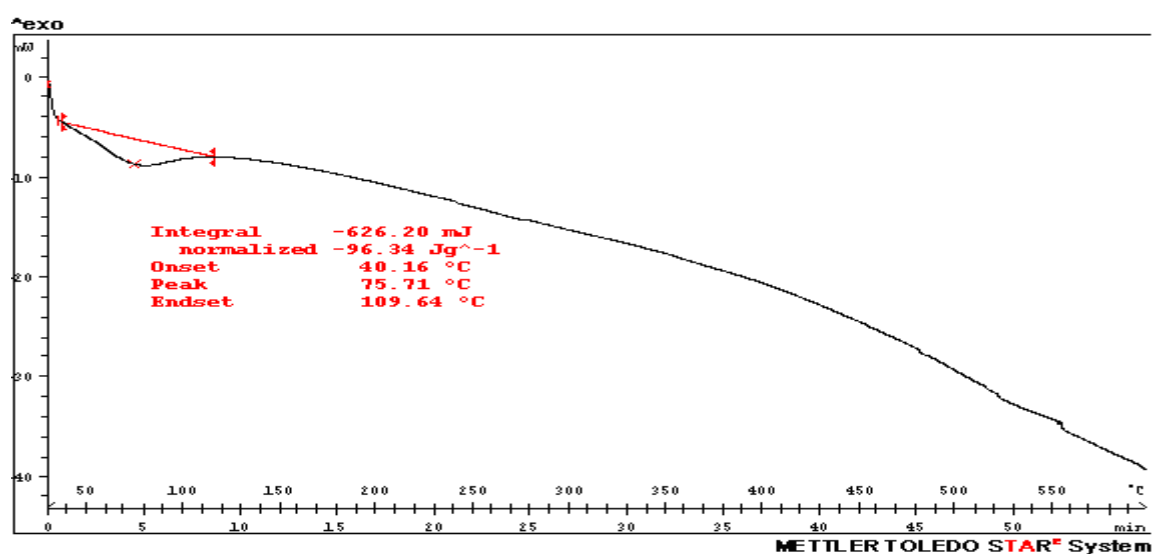


Figure 3 - Different stages of DSC Thermo-gram of coal sample (Nitrogen/Inert atmosphere)

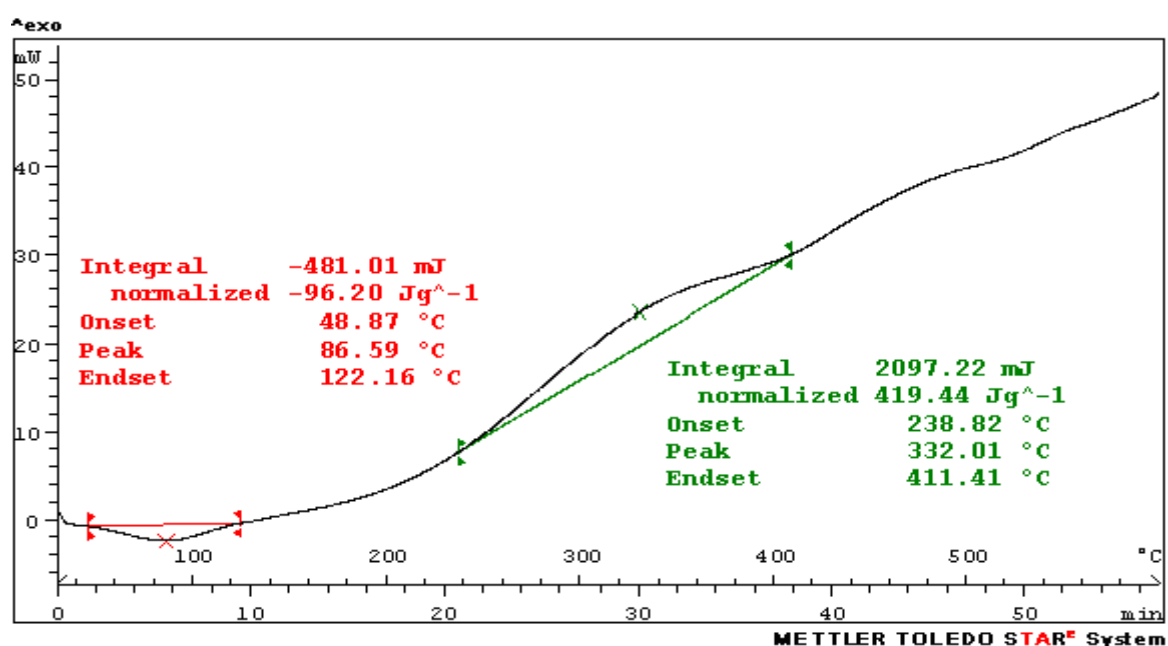


Figure 4 - Different stages of DSC Thermo-gram of coal sample (Air atmosphere)

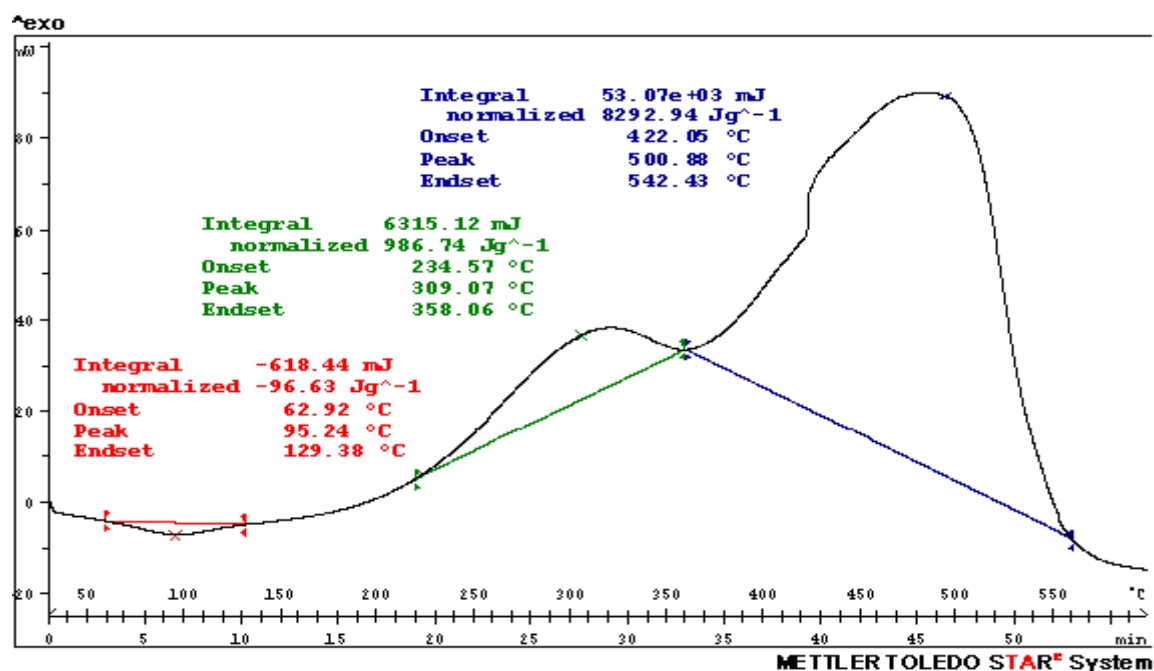


Figure 5 - Different stages of DSC Thermo-gram of coal sample (Oxygen atmosphere)

RESULTS AND DISCUSSION

The combustion process was exceedingly complex and many competing processes contributed to the behaviour of thermal analysis curves. Theoretically, the spontaneous combustion of coal can be initiated whenever oxygen comes in contact with the coal. So, experiments should be carried out in oxygen or air atmosphere because of easy oxidation of coal in air. In case of nitrogen or inert atmosphere, the study showed that oxidation may not take place, thus it was not suitable for the determination of spontaneous heating.

DSC provides both qualitative and quantitative information about material transition during the spontaneous heating process as compared to DTA and TGA study of coal. In the region of first endothermic peak of DSC, the enthalpy oxidation is mainly due to release of moisture. The onset temperature of first exothermic peak in oxygen atmosphere is possibly a measure of the susceptibility of coals towards spontaneous combustion. This has a number of potential advantages. Firstly, the overall weight change at this temperature is negligible, thus avoiding the problems inherent in exotherm integration. The heat evolved during the early stages is crucial to subsequent oxidation by inducing relaxation in the coal structure which enhances the mass transfer of oxygen. For some of these transitions, DSC can provide, not only the temperature at which the transition (reaction) occurs and how much heat is evolved, but can also provide valuable information about the rate (kinetics) of reaction. Furthermore, with the advent of easy-to-use computer based data analysis programs, the ability to obtain such kinetic information has become more practical. Differential scanning calorimetry will be a useful technique to investigate the early stage of oxidation of coal.

CONCLUSION

Differential scanning calorimetry techniques are useful tools to study the combustion, pyrolysis behaviour, kinetics and assessment of spontaneous heating of coal. The critical analysis of the literature survey on the application of DSC technique and summarised data presented in Table 1 clearly reveals the following salient features.

- DSC technique has in general had limited application for studying the susceptibility of coal to spontaneous heating and fire.

- There is no general agreement and unanimity regarding the experimental standards to be adopted for studying the susceptibility of coal to spontaneous heating and fire in above techniques.
- Thermal behaviour of coals depends on experimental conditions such as particle size, sample amount, heating rate, carrier gas and its flow rate.

The DSC study can be used as an indicator to determine the susceptibility of coals to self heating or spontaneous combustion. The acceptability of this method for determining spontaneous heating characteristics of coal mainly depends upon how closely it predicts the spontaneous heating in the field conditions. This will help to scientifically classify the coals with respect to their proneness to spontaneous heating, so that practicing mining engineers, mine planners and mine operators can formulate the ameliorative measures in advance. This will help to save coal from burning, and will contribute to improve production, productivity and safety in mines, as well as reduce atmospheric pollution

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OPPORTUNITIES FOR IMPROVEMENTS IN SAFETY AND HEALTH MANAGEMENT SYSTEMS FOR COAL MINES - AN AUDITOR'S PERSPECTIVE

Ian McDonell¹

ABSTRACT: Auditing of Safety and Health (or Health and safety in NSW) Management Systems (SHMS) is an integral and essential process to make sure that an operation demonstrates an understanding of legislative requirements and complies with both the written law and its intent.

Auditing SHMS in both NSW and Queensland has come a long way since the implementation of updated mining safety legislation that mirrors or relies on parent Workplace Health and Safety (WHS) or Occupational and Health Safety (OHS) laws. Intrinsic in this is the application of relevant Australian Standards on SHM and Quality Management.

The audits in use may not fully address these parameters, and this paper attempts to identify some of these concerns, and hopefully suggest some areas for improvement in the systems and their auditing.

The material for this paper comes from both the author's experience, and experience of other external auditors including members of the inspectorates in both states.

INTRODUCTION

The author has been involved since 1998 in the development, implementation and auditing of Safety and Health Management Systems (SHMS) in mine and compliance management roles, and has a concern that many organisations may not fully understand the depth or intent of legislation as it applies to a site.

The material presented is the personal opinions of the author, and does not reflect the corporate views of any organisation. It is the result of a number of years of analysis of legislation, preparation of systems, training and auditing. The paper is an attempt to assist others to reach understanding of the range of options for auditing of SHMS at mines, and points out what the author believes are commonly found opportunities for improvement in this process.. These opportunities are not specific to any organisation or any mine, but a gathering of data from the author and many other auditors working in this field.

The author has been involved since 1998 in the development, implementation and auditing of Safety and Health Management Systems (SHMS) in mine and compliance management roles, and has a concern that many organisations may not fully understand the depth or intent of legislation as it applies to a site.

BACKGROUND

The Operator's audit is a requirement of the Queensland Coal Mine Safety and Health Act 1999 section 41(1) (f) as part of the "Obligations of coal mine operators", in this case to "audit and review the effectiveness and implementation of the safety and health management system to ensure the risk to persons from coal mining operations are at an acceptable level".

Under the New South Wales Coal Mine Safety and Health Act 2002, the requirement is stated in section 23 "Contents of health and safety management system", specifically subsection (3) (a) "system elements which must include ... system evaluation".

Neither legislation specifies the need for external auditing, nor sets periods for such audits. Auditing and review are key elements of a Safety and Health Management System under the referred Australian Standard 4801:2001.

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In 2007 in Queensland it was recognised that the operators audit had been approached in many different ways that in most cases was not in agreement with the Inspectorate view. To attempt to get some alignment, a series of meetings and communications was implemented to set some concepts and standards to this process. Along with the Inspectorate, representatives of the coal holders and operators and the external auditors being used at the time agreed on a range of auditable elements of the systems that could be used to show the effectiveness of and continuous improvements in the systems. From this was created the Queensland Guidance Note (QGN09) "Reviewing the Effectiveness of Safety and Health Management Systems". In that document key system elements are identified as:

- Change management
- Work force involvement
- System performance: lead and lag indicators
- Causal analysis: repairing defences
- Audit and inspection findings
- Contractor safety and health
- Chronic exposures causing incapacity

Note that the Guideline does not mention "compliance" or "implementation" in its title. Auditing of these elements is discussed later.

SOME EXPERIENCE TO DATE

The responsibility for operator's audits in Queensland had been left to each site until 2006, either by way of the Site Senior Executive (SSE) appointing an internal or external auditor or by the operator's SSE doing the same for contractor operated sites.

In NSW the arrangements for audits have been at the discretion of the senior site management, and have used both internal and external personnel.

The difference between "compliance", "implementation" and "effectiveness" was not well recognised, so that most audits concentrated on achieving technical legislative compliance by having SHMS documentation that addressed every item to the standards required by that legislation.

Since then, it has been recognised that "implementation" is the process of system production, training personnel and embedding procedures into the workplace to ensure that all workers comply with those requirements. Further to this, "effectiveness" is the measure of how reducing risk to acceptable levels and creating continuous improvement has been developed.

Discussions across the industry and personal experience of the author have led to the conclusion that achieving consistency in approach and results is still problematic.

DEFINITIONS FOR AUDITING CONCEPTS

The following are not in any way legal definitions, but the author's attempts to identify the difference between the terms, so that ways to measure these for auditing may be developed.

Compliance

Those things that are required to meet the detail of the legislation such as:

- The combination of documentation, equipment and procedures that address the matters described by the legislative element.
- inclusion of risk management, technical analysis, and acquisition standards.
- "If the law states that you need three of something, and they are to be painted green, then compliance is reached when you can show you have three green painted ones."

Further detail is given Table 1

Implementation

Those things that are required to put the implementation into effect such as:

- The processes to train, assess, construct, maintain and use or otherwise deal with the procedures, equipment and materials that will be needed to achieve the total workforce understanding and usage of the risk control measures required for the particular legislative requirement.

“The workforce understands the need for the three green things, knows how to use them and maintain them, knows where they are to be used and when, and understands how they will reduce risk to acceptable levels.

Further detail is given in the Table 2.

Table 1 - Measures for Compliance

<u>Indicator</u>	<u>Criteria</u>
<u>Legislation identification</u>	• Relevant legislation identified • Legislation update process in place • Management team involvement
<u>Legislation analysis</u>	• Legislation analysed for compliance requirements • Management team involved • Compliance tracking tool used • Corrective action system used
<u>Legislation included in SHMS</u>	• SHMS addresses all legislation
<u>SHMS compliant with legislation</u>	• SHMS elements comply with legislative requirements
<u>Additional referred documents identified</u>	• Referred and referenced documents in legislation identified • Update process in place • Management team involvement
<u>Additional referred documents analysis</u>	• Documents analysed for compliance requirements • Management team involved • Compliance tracking tool used • Corrective action system used
<u>Additional referred documents included in SHMS</u>	• SHMS addresses all referred documentation
<u>SHMS compliant to additional documentation</u>	• SHMS elements comply with documentation requirements

Effectiveness

“The measurement of the result of compliance and implementation, which is the demonstrated lowering of risk to acceptable levels for that element of the legislation”

- Detailing the key performance indicators or other descriptors that will be used to show that the risk controls are fully implemented and are having a positive effect on the safety and health performance of the site.
- These may include reactive and proactive measurements, cultural surveys, audits, investigations, feedback or other measures determined by the mine management.
- The three green things have improved the Total Recordable Case Frequency Rate (TRCFR) by preventing incidents, the workforce has stated that they value them and audits show that they are used and maintained.

Table 2 - Measures for Implementation of the SHMS

Indicator	Criteria
<u>Legislation</u>	• Process to identify applicable legislation, including compliance registers and updates • Appropriate training and communication systems for all personnel
<u>Consultation and communication process</u>	• Formal documented system • Entire workforce coverage • Records of topics • Records of attendance • Absentee follow up • Includes process to manage concerns and objections
<u>Risk Management System</u>	• Complies with standards • Multi levels including broad brush, personal, team and site wide • Directs use of hierarchy of controls • Written acceptable risk limits • Training in risk management to appropriate persons and suitable skill levels
<u>Contractor Management Plan</u>	• Covers all contractors, consultants • Thorough pre-use review process • Induction covers all legislation
<u>Change Management System</u>	• Multi levels • Justification required • Formal documented system
<u>Hazard and inspection reports</u>	• Covers statutory and enterprise requirements, range of types • Based on situational risk • Risk based actions, both immediate and follow up • System to follow up and close out • Supervisors reviews
<u>External Information System</u>	• Formal documented check sheet system • Appropriate analysis system • Action planning and follow up • Dissemination system • Archive system
<u>Action Planning and Process</u>	• System to allocate actions, responsibility, timetable, and follow up / escalation process • System to review and close out
<u>Incident Management System</u>	• Formal documented system / template • Includes near miss, hazard reports and suggestions • Includes health and safety • Covers all data requirements • Detailed incident analysis leading to appropriate and accurate root causes • Action requirements / taken system based on risk • Communication strategy • Training of appropriate personnel to suitable standards
<u>Supervision management</u>	• Formal documented system • Based on situational risk • Appropriate documented reports include actions taken and required

Further detail of the measurement of the above concepts is presented in Table 3.

COMMON AREAS FOR IMPROVEMENT IDENTIFIED

The following items have been noted in audits as commonly being less than adequately handled in many safety and health management systems. These are listed in alphabetical order and no priority is given to any item as the author believes they have equal value to the whole SHMS.

Change management

The SHMS should include a formal documented process for management of change including;

- Risk based approach that evaluates business needs, benefits and resource requirements
- Action planning and tracking process
- Appropriate levels of management sign off
- Auditing of results for implementation and effectiveness

Acquisition, tracking and implementation of externally sourced safety and health information

The SHMS should include a formal documented process including:

- Sourcing of information – incident reports, safety alerts, notices from various authorities, inter-company and Original Equipment Manufacturer (OEM) releases, internet sites especially inspectorate
- Logging of receipt of information and use of tracking sheet
- Contents of tracking sheet including allocation of responsibilities for assessing, using and disseminating
- Determination of any actions required, implementation into action planning process, change management considerations
- Close out by senior management including filing process

Actions and controls under HSMS to match perceived or actual level of risk

This applies to any actions or controls applied from risk assessment, accident or incident action plan, and remedial action plan from inspections or external notifications so that,

- Criticality of matching level of control with level of risk
- Use of hierarchy of controls strictly in response to levels of risk
- Extreme risks require elimination or substitution controls only
- High risks require engineering solutions or higher
- Actions and controls need to be audited for completion and effectiveness

Action plan allocation and tracking to auditable close out by:

- Use of software systems that are document controlled
- Actions must be allocated to an appropriate person, not a position
- Actions must have a time frame relevant to the actual or perceived risk
- Actions must be tracked and reported on a regular (monthly?) basis
- Incomplete actions must escalate up the management tree
- There must be demonstrable and auditable close out of all action plans

Real root cause analysis of incident, accident and health reports to identify and action controls for repetitive trend events so that:

- All accidents and incidents must be investigated to a root cause or causes
- Causes must be relevant to the actual incident or accident, not generic
- Data logging into major causal types is essential to identify trends, and establish repetition of incidents for higher level analysis and action
- Health considerations are critical, including long and short term health effects of workplace and work conditions
- Actions identified to be managed by action planning process and to be relevant to risk

Table 3 - Measures for Effectiveness of the SHMS

Indicator	Criteria
Annual S and H planning and implementation process	• Formal documented system • Includes previous performance • Addresses areas for improvement • Uses consultative process to develop • SMT input and acceptance • KPIs developed to measure results of programs
Action and Control Risk Evaluation	• System includes methods to measure effectiveness of actions to address original problem and reduce risk • KPIs developed to measure results of programs
Consultation and communication process	• Communications address actual or perceived risk levels • System includes methods to measure effectiveness of actions to address original problem and reduce risk • KPIs developed to measure results of programs • Workforce concerns addressed in timely and risk based manner
KPI measurement and performance	• Range of proactive and reactive KPIs developed and used • KPIs to demonstrate continuous improvement
Health management	• Formal documented system to identify health matters • Systems to measure and monitor • Notification to affected workers • Follow up systems including medical testing
Training system	• Refresher training in statutory matters fully addressed • Records kept to demonstrate
Workforce feedback	• System to obtain, record, evaluate, action and follow up workforce feedback • System includes methods to measure effectiveness of actions to address original problem and reduce risk • KPIs to show use and results
Previous audits	• Documentation showing previous audit results have been analysed, actioned and followed up • System includes methods to measure effectiveness of actions to address original problem and reduce risk • KPIs to show use and results

Demonstration of effective communication and consultation processes including the following:

- Understanding the difference between communication and consultation and where each is to be used
- Consultation to be appropriate to the matter at hand, including use of affected workforce
- Consultation is not necessarily consensus
- Understanding the difference between communication and training and where each is used
- Types and levels of communication/training to be derived by a risk based approach
- Where risk levels are high communication should be formal training with assessments
- All communication processes that have a safety or health base should have recorded attendance

- It is essential to record absentees (including management) for follow up communication sessions
- Venues for communication must reflect ability of attendees to receive and absorb data – comfort, seating, acoustics, displays, etc
- Regular auditing of sessions is needed to make sure entire workforce receives the message
- On the job challenge tests are warranted for testing effectiveness of communication

Handling of reports on hazards, especially written reports including:

- The process must break the “tick and flick” sign off cycle that is common for inspection reports, especially where a defect or hazard is identified but no actions listed or recommended
- Deputy / Open Cut Examiner (OCE) reports frequently do not have actions for hazards identified or signed off as completed.
- Consider revision of inspection reports to include actions taken, actions required, assessment of urgency, etc, and a procedure to put uncompleted hazards into the action planning process
- All supervisory personnel should complete inspection reports, whether required by law or not, as a standard to demonstrate duty of care
- All hazards not addressed at the time of report should go onto action planner system to capture, track and close out

HSMS annual planning and implementation process so that

- A formal HSMS planning and implementation process is recommended
- The plan should include a combination of reactive and proactive measures that address the safety and health history of the workplace, not necessarily the industry as a whole.
- Each element on the plan should have a justification that can be demonstrated based on site history or workforce input
- Each element should be costed, evaluated by a management of change method and subject to a benefit analysis.
- HSMS planning and implementation is a responsibility of all management and supervision. It should have input from the whole workforce, via the Safety Committee, Site Safety Representative / Check Inspector. The role of the workers representatives in this matter cannot be overstated; it is possibly the strongest test for effectiveness of a SHMS.

Some key points that a SHMS auditor may consider

The following material details specific considerations for these audits, based on input from the Inspectorates and external auditors in both states and from personal audit experience.

- Is there a system at the mine to reduce risks to an acceptable level?
- How does the system plan and express this target?
- Does the SHMS set measurable acceptable limits (e.g. by use of a risk matrix) for managing risk or does it rely on “as low as reasonably achievable”?
- Is the system complete, controlled and accessible to all workers?
- Are all coal mine workers including contractors, consultants, visitors effectively covered by the system? How is this coverage demonstrated?
- Are concerns, objections and other matters raised by the workforce dealt with by a recorded process to investigate and action?
- Is the consultation process wide ranging, implanted, demonstrated and recorded? Are all groups of affected workers represented in the consultation process? Is a Safety Committee used? Is there consultation with site and industry safety and health representatives?
- Is the communication process in place and is it responsive to level of risk? Is the communication passive, active or a combination? Can the communication process be shown to be effective? Is the attendance of all workers and others captured by the communication system? What is the communication catch up process for absentees?
- Is there a system to capture, track, analyse and implement measures from externally sourced safety and health information such as legislative changes, incident reports, OEM releases?
- Is there an imbedded management of change process that is consistently and appropriately applied at all times by all persons? Can this be demonstrated? Is it based both on business need and risk management processes?

- Is there a documented corrective action process that is fully utilised, followed up, closed out, and reviewed for effectiveness of actions?
- Is there an effective accident, incident, near miss and hazard reporting system (including safety suggestions) that can be shown to have investigation and reporting relevant to risk, root cause analysis, trend analysis, action planning and prioritisation related to risk, close out and reporting processes and internal and external dissemination processes relevant to risk?
- Does this system look at other health matters such as occupational diseases, overuse syndrome, health issues such as stress related illness, etc?
- Is there a documented management system for potentially chronic exposures that may lead to disability / incapacitation? How were the exposures determined? What risk management is used? What monitoring is used?
- Is there a hazard identification system in place that covers the range from broad brush to task detail by multi level tools such as Stop, Look, Assess (SLA) and Manage, Job Safety and Environment Analysis (MJSEA), Workplace Risk Assessment and Control (WRAC), etc, and are these used in the right contexts?
- Is the formal hazard identification, risk analysis and control implementation process to an acceptable standard, are procedures written from and related directly to risk management processes, can it be shown to be fully implemented across the entire workforce, are controls related to the hierarchy of controls, are the controls fully implemented by way of action planning process and measured for effectiveness and sustainability, is the documentation fully completed and accessible?
- Are there risk based triggered action response plans in place for a wide range of events based on site wide risk assessments? Are all personnel fully trained in crisis and emergency response and are there exercises conducted to a plan to demonstrate effectiveness?
- Is the Training Management Plan in place and fully implemented? Does it have a needs analysis that includes hazard identification, risk management and related topics to a suitable standard? Are all legislative matters captured in the plan? Is the refresher training covered adequately? Is the record system up to date?
- Is there an inspection and supervision management system in place that is based on legislation and risks identified for example from a broad brush risk assessment? Is the supervision trained and competent to identify and manage risk to an acceptable level? Can the effectiveness of supervision be demonstrated?
- Is there a system of leading and lagging indicators that can show continuous improvement related to the implementation and effectiveness of the SHMS? How were the indicators developed and justified?
- Is there a formal annual planning process that includes the results of mine Safety and Health performance, audits and reviews of the SHMS and sets targets for the mine planning/budgeting process, including resources? Are these targets shown to have justification e.g. based on risk? Can workforce involvement in the plan be shown?
- Have previous audit results been demonstrably used as triggers for reviews, and have the audit results been assessed for use in the improvement process?
- How is the management responsibility for audit and review of the SHMS handled? What are the triggers for management audit / review? How are the outcomes recorded, handled and communicated?
- Are the system and its implementation driving a culture that looks to proactively reduce risk to acceptable levels and in particular prevent recurrence of incidents?

FUTURE DIRECTIONS

The following strategy is appropriate for the future audit needs:

1. All compliance, implementation and effectiveness audits should be combined and handled internally by personnel who are selected and trained for these roles, and have suitable time allocation to prepare, conduct and report these matters. These audits can be broad brush, in that they address the entire legislative area for the site, or selective to drill down into individual elements that may be of concern for the site or industry as a whole.
2. External auditing can be done to verify the process, but should be limited in scope for each audit and not necessarily involve all sites and all system elements.
3. The audits should use prepared worksheets detailing audit requirements for each element, which are technical compliance, implementation and effectiveness. These would be produced by

- consultative process using suitable external and internal legal, safety and health and compliance personnel, and reviewed by senior management before use.
4. It is essential that sites are given audit templates and guidance instruction well in advance of the actual audit, so that they can apply appropriate resources to preparing the material required.
 5. The same standard of audits should be applied at all sites, whether operated by owners or contractors, and to include related off-lease sites where applicable, such as external train load outs, waste dumps, etc.
 6. Given that the range of material to be audited is too large to handle annually, and that there is no legal reason to do so, it would be sensible to target areas of the systems each year on a risk based approach and on actual site incident histories if applicable. Reviewing results of previous audits is also warranted.
 7. It is important for the auditors to provide guidance and assistance where practical in the implementation of suggested improvements. This provides some consistency in approaches taken but is obviously restricted by available time.

CONCLUSIONS

Industry examples have shown the high cost to the business of not having an effective safety and health management system in place. There is a significant opportunity for organisations to mitigate this risk by careful application of the legislated requirements for review and auditing.

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The views expressed in this paper are those of the author, and do not purport to have any relationship to any coal mining operation, operator or mining company.

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GEOTECHNICAL RISK ASSESSMENT IN KERMAN COAL MINE- CENTRAL IRAN

Mohammad Hossaini¹ and Sommayeh Behraftar¹

ABSTRACT: Mine safety in underground coal mines is normally threatened by the likelihood of accident occurrence. The outcome of such occurrences includes and is not limited to loss of machinery and equipment, loss of life, injury, disability, and mine closures. In this study, the Risk Priority Number (RPN) has been determined for the Kerman Coal Mine and the main causes of uncertainty found through the RPN. The implementation of a decision tree and a risk management plan considering the causes of accidents has been proposed. Data covering a complete range of every accident occurred during the time period of 2003-2008 has been analyzed and the accidents have been classified and sorted by RPN. It has been shown that amongst all types of incidents, the risk of roof failure is the most probable risk of all. It is concluded that the probability of an accident occurring every 24 days is 95%. It has been shown through the decision tree that due to the high number of accidents, the cost of investing in preventative measures is significantly less than costs related to accident consequences and therefore, financially justified.

Key words: Risk Analysis, Risk Priority Number (RPN), Decision Tree.

INTRODUCTION

Underground coal mining is one of the most high risk industries and every year, accidents in mining activities cause fatalities, serious injuries and incurring heavy financial losses. Minimizing existing risks and providing a safe working environment not only requires a well planned management of the risks related to various accident types, but also the creation of suitable solutions for them.

In underground coal mines, roof collapse as a geotechnical risk, is amongst the main causes of accident occurrences. The consequences of such failures are worker's disability, death, injury, equipment damage and financial losses. During 2003-2008, in Kerman Coal Mines, 59% of accidents and 30% of fatalities have been caused by roof collapse (HSE section, Mineral Supply and Production Co., 2008).

In this paper, the risk of accident occurrences has been studied and for this purpose quality data has been quantified through statistical analysis involving the decision tree method has been used for assessment and management of risks.

DEFINITIONS

To avoid any confusion in the interpretation of the terminology used in this paper, the following definitions adopted from literature are presented (Einstein, H.H., 1997 and Duzgun, H.S.B., Einstein, H.H., 2004):

Uncertainty: Implies a condition in which not only the probable happenings are not known, but also the probability of known happenings is not clear. In other words, neither the probable happenings nor the probability of their occurrences is clear.

Danger: Although the potential of rock fall from the roof exists, the characterization of danger does not include any forecasting.

Hazard: Conditions in which the probability of a roof collapse exists in a certain period of time.

Risk: Implies a condition in which not only the probable happenings are known but also the probability of known happenings is almost clear. However, which incident may occur is unknown. Therefore,

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making a decision in a high risk environment is much easier than in an uncertain environment. Since the hazard may lead to different consequences depending on the mine environment, risk can be introduced as;

$\text{Risk} = \text{Hazard} \times \text{Consequences}$

In the case of roof fall;

$\text{Risk} = \text{Probability of roof fall (P)} \times \text{Consequences or } R = P \times C$

Where R= Risk associated with roof fall, P = probability of roof fall occurrence in a certain period of time (Hazard) and C = Consequences which may include fatalities, injuries, disabilities, equipment breakdown, loss of time, etc.

CLASSIFYING THE INCIDENTS IN THE KERMAN COAL MINE

The effect of risk is determined through probability and consequences. A probable incident and its consequences have been classified through a ranking of 1 to 10, depending on its degree of probability and the intensity of its consequences. Risk Priority Number (RPN), which varies from 1-100, is then calculated by the multiplication of the two above rankings. RPN is employed in deciding on necessary modifications for avoiding or reducing the potential errors.

Data collected from all types of incidents occurred in Iranian coal mines during the time period of 2003-2008 has lead us to adopt specific classifications suitable for all nationwide underground coal mines. As shown in Tables 1 to 3, this classification ranks the probability of occurrence, intensity of injuries and number of "out of work" days from 1-10. RPNs shown in Table 4 were then calculated from due numbers in Tables 1-3.

As seen in Table 4 being struck by flying debris, being trapped between heavy objects and destruction have the highest RPN. Geotechnical problems (mostly roof failure) are the main cause of these incidents. The total RPN for the 3 main incidents caused by roof collapses is 690 out of 1000. This indicates that this type of risk (roof failure) requires particular attention.

Table 1 - Classification of "occurrence probability"

Occurrence frequency	Ranking
More than once a day	10
More than once a week	9
Once a week	8
More than once a month	7
Once a month	6
More than once in 6 months	5
More than once a year	4
More than once in 2 years	3
More than once in 2-5 years	2
once in more than 5 years	1

Table 2 - Classification of "injury intensity"

Type of injuries	Ranking
Death	10
Amputation	9
Burns	8
Bone breakage & dislocations	7
Internal cuts and bleeding	6
Electrical shock	5
Bruising, twisting & strain	4
Poisoning	3
Cuts & wounds	2
Load lifting injuries	1

Table 3 - Classification of "out of work" times

Out of work times	Ranking
Permanent	10
2-3 years	9
1-2 years	8
6 - 12months	7
3 - 6 months	6
1- 3 months	5
1 week – 1 month	4
1 day – 1 week	3
1 shift - 1 day	2
No out of work time	1

RISK ESTIMATION

In order to evaluate the risk involved in a roof failure incident, two main components are required. These components are probability/hazard and consequences.

PROBABILITY OF ACCIDENT OCCURRENCE

There are two methods of estimating the probability of accident occurrence:

- 1 Observational method
2. Statistical analysis of previous incidents (Einstein, H.H., 1997).

In the first method an experienced miner/engineer can estimate the probability of an accident based on the condition of a mine and certain indicators around the mine. In the second method the probability of accidents can be estimated by analyzing the specifications of previous accidents and their intervals. This method can be more precise provided that information related to the type, date and consequences of the accidents are available.

In this study, because all data pertinent to the previous accidents of Kerman Coal Mine is available, the second method has been used for accident estimation. The number of roof failures in each method (NOF) and time intervals between failures (TBF) from 2003 to 2008 has been analyzed. Statistical outlines of these factors are detailed in Table 5. Considering the

nature of the data, proper distribution is selected and using data distribution function their probability is obtained. As the NOF data is distinctive, the Poisson distribution seems to be suitable. The K^2 fitting test is performed to evaluate this way of distribution.

Table 4 – RPN for incidents happened in Kerman Coal Mine during 2003-2008

Number	Incident type	Degree of probability	Av. degree of injury intensity	Degree of disability	RPN
1	Being struck by flying debris	7	6	9	378
2	Being trapped between heavy objects	4	5	10	200
3	Destruction	2	7	8	112
4	Falling from a higher surface	4	7	7	196
5	Tripping on a flat surface	3	5	7	105
6	Exposure to electrical circuits	2	5	10	100
7	Gas poisoning	1	10	10	100
8	Explosion	3	6	5	90
9	Colliding with moving objects	7	5	1	35
10	Exposure to severe heat	2	1	5	10

Table 5 – Statistical outlines of NOF and TBF for Kerman Coal Mine

	Range	Minimum	Maximum	Mean	Standard Deviation
NOF	10	0	10	6.22	3.95
TBF	101	0	101	8.1	10.89

Table 6 represents the results of K^2 fitting test.

Table 6 – K^2 Test results with 95% of confidence for NOF in Kerman Coal Mine

	Test	Degree of freedom	Critical amount	Fitting quality
NOF	78.3	59	79.1	good

Therefore, TBF is well represented by exponential distribution. Probability Density Function (PDF) and Probability Mass Function (PMF) are represented by Exponential and Poisson distribution, given by Equations 1 and 2, respectively.

$$f(x) = \frac{1}{\theta} e^{-x/\theta} \quad 0 \leq x < \infty \quad (1)$$

$$p(x) = \frac{e^{-\lambda} \lambda^x}{x!} \quad x = 0, 1, 2, \dots \quad (2)$$

Where θ and λ are the parameters of exponential and Poisson distributions respectively.

As both Poisson and exponential distributions are single parameter distributions, it is only necessary to estimate the mean values of TBF and NOF (Einstein, H.H., 1997). Assuming that the individual average of the data for TBF and NOF are best estimates of the distribution parameters, θ and λ would equal to average amounts of NOF and TBF respectively.

Therefore, the probability of a roof fall in t days could be obtained from Equation 3 (Einstein, H.H., 1997).

$$P = \int_0^t \frac{1}{\theta} e^{-x/\theta} dx \Rightarrow P = 1 - e^{-t/\theta} \quad (3)$$

CONSEQUENCES OF ACCIDENTS IN KERMAN COAL MINE

The rates of losses by roof falls have been determined by the safety group of Kerman Coal Mine as follows:

Fatalities; 25%

Injuries; 30%

Equipment damage; 30%

Delay in operations; 15%

As seen in these ratings, 555 of the losses are due to fatalities and injuries. Figure 1 depicts the rate of each type of injury the mine workers suffered as a result of roof failure.

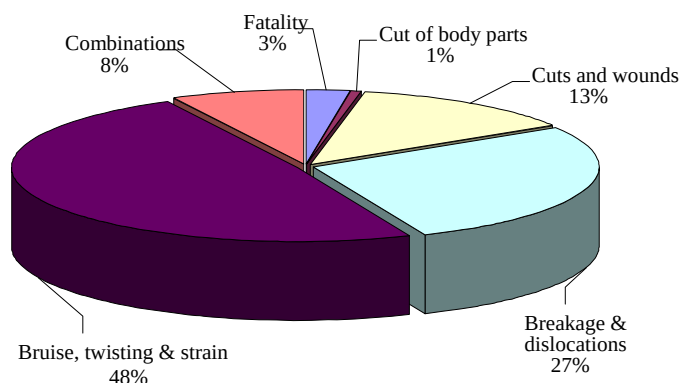


Figure1 - Rates of various types of injuries due to roof falls in Kerman Coal Mine

RISK IN KERMAN COAL MINE

As stated before, risk equals the probability of roof fall multiplied by consequences. On the other hand, the probability of rock fall can be estimated from time intervals of accidents and its distribution (Equation 1). Thus the roof fall risk in t days can be calculated from Equation 4 (Duzgun, H.S.B., Einstein, H.H., 2004).

$$R(t) = C_T (1 - e^{-t/\theta}) \quad (4)$$

The probability of roof failure can be obtained from Equation 3. Figure 2 depicts the probability of accident occurrence due to roof failures in Kerman Coal Mine. As this figure illustrates, the time intervals between 2 accidents is in 95% probability to be less than 24 days.

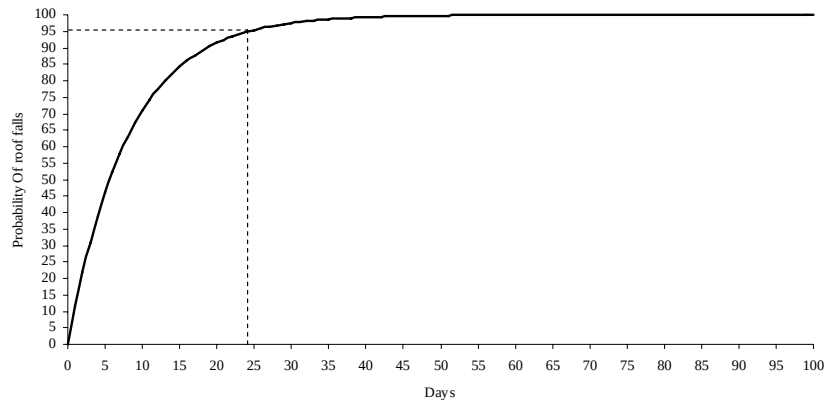


Figure 2 – The probability of accident occurrence in t days

DECISION MAKING ASSESMENT

To propose a solution for the safety problems in the Kerman Coal Mine, the number of accidents from 2003 to 2008 were used as a basis for evaluation. An assessment of the financial justification of risk reduction was also done through the number of accidents and by use of decision tree. Decision tree led us to 2 ways from which we will need to select one. The two options are as follows:

a_1 ; No action required; maintaining the status quo and

a_2 ; reaching a solution in order to reduce/prevent accidents by utilizing experiences from previous occurrences.

As shown in Figure 3 each branch results in two sub branches; one related to the case where failure occurs and the other is for a “no failure” case and therefore bears no consequences. If failure happens, the related sub- branch would be divided into k sub-branches where $k = 1, 2, 3$ is the number of roof failures. Each roof fall would have its own specific consequences, depending on its circumstance. The cost of losses in a_1 main branch depends on the number of falls and their type of consequence would equal C_1 and the cost of losses and prevention measures in a_2 main branch equals C_2 . Therefore, k denotes the number of roof failures in a year and $P(k)$ is the probability of k being equal to 1, 2, 3.

Assuming that after the improvement of conditions, a_2 branch leads to a $Q\%$ saving in total roof fall costs the amounts of C_1 and C_2 can be calculated from Equations 5 and 6, respectively.

$$C_1 = C_T k, \quad k = 0, 1, 2, \dots \quad (5)$$

$$C_2 = \left(1 - \frac{Q}{100}\right) C_T k + C_a \quad k = 0, 1, 2, \dots \quad (6)$$

Where;

C_1 = imposed losses in a_1 branch,

K = number of falls,

C_2 = imposed losses in a_2 branch,

C_T = imposed losses due to a single fall, and

C_a = cost required for improvement in a_2 branch.

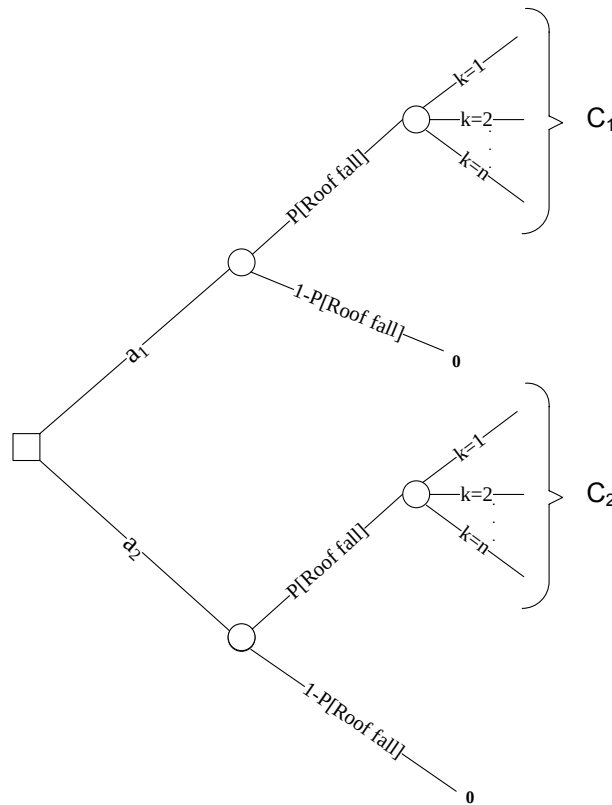


Figure 3 - Decision tree for accident occurrences in Kerman Coal Mine

Equation 7 can be employed in order to compare the costs imposed by any branches, through the

$$E[a_i] = \sum_{k=0}^{\infty} C_i P(k) \quad (7)$$

number of annual accidents.

Where, $P(k)$ is obtained from Equation 2.

The expected amounts for a_1 and a_2 branches can eventually be calculated from Equations 8 and 9, respectively (Duzgun, H.S.B., Einstein, H.H., 2004).

$$E[a_1] = C_T \lambda \quad (8)$$

$$E[a_2] = \left(1 - \frac{Q}{100}\right) C_T \lambda + C_a \quad (9)$$

Depending on the cause of accidents and the reduction rate in losses, the values of Q and C_a vary on a case by case basis.

Finally, by following the sub-branches and employing related equations, a branch which bears lower costs would be selected.

DECISION ANALYSIS OF ACCIDENTS IN KERMAN COAL MINE

Figure 4 illustrates a histogram of the number of monthly accidents.

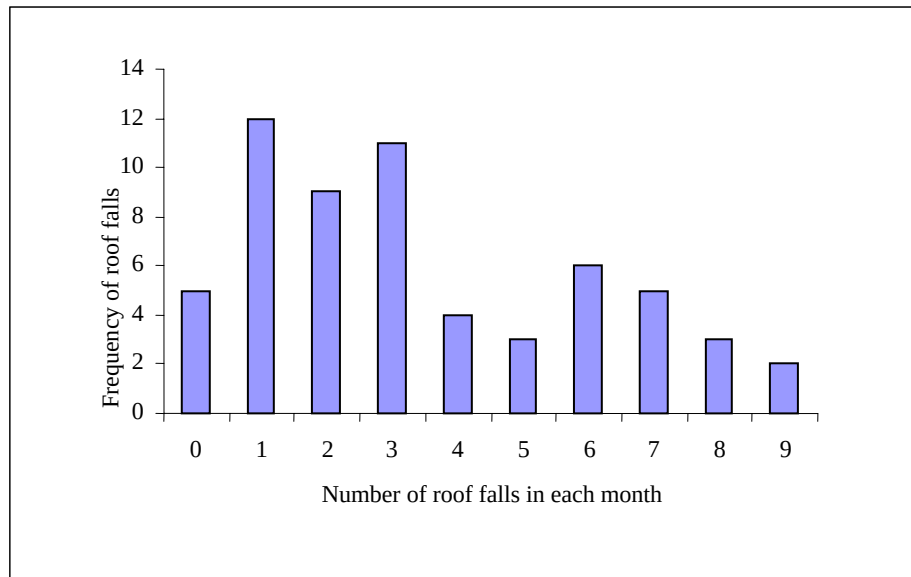


Figure 4 - Histogram of monthly NOF in Kerman Coal Mine

The relative costs in a_2 branch (i.e. C_2) can be calculated with the following assumptions proposed by the safety group of the main company:

- 1- The adoption of a well planned training program, thorough safety inspections and improving roof conditions will lead to a 40% reduction in losses due to accidents.
- 2- The costs of training, safety inspections and improving roof conditions is 1.3 times of losses due to doing no action which is represented by a_1 branch (i.e. C_1).

C_2 could then be calculated from the Equation 10.

$$C_2 = 0.6C_T k + 1.3C_T \quad (10)$$

The average number of accidents per year can be obtained from monthly accidents and Table 5 using Equation 11 as follows:

$$\begin{aligned} t &= 12 \\ \lambda_m &= 6.22 \Rightarrow \lambda_y = t\lambda_m \Rightarrow \lambda_y = 74.64 \end{aligned} \quad (11)$$

Using the amounts of λ_y obtained by Equation 11 in the Kerman Coal Mine, the amounts related to each branch are obtained by Equation 12 as:

$$\begin{aligned} E[a_1] &= C_T \lambda \Rightarrow E[a_1] = 74.64C_T \\ E[a_2] &= 0.6C_T \lambda + 1.3C_T \Rightarrow E[a_2] = 46.08C_T \end{aligned} \quad (12)$$

As seen in equation 12, the amount related to a_2 branch is significantly lower than that of a_1 branch. In other words, due to the high number of accidents in the mine, investing in prevention measures is financially justified.

CONCLUSIONS

This investigation of accidents has lead to the followings conclusions in Kerman Coal Mine:

- 88% of accidents are caused by roof failure.
- The Risk Priority Number (RPN) is relatively high and equates to 690 out of 1000.
- With the current situation it is 95% probable that an accident will happen every 24 days.
- Costs related to prevention measures are considerably less than those of accident consequences.

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CHANGES IN ACOUSTIC EMISSIONS WHEN CUTTING DIFFERENT ROCK TYPES

David Crosland¹, Rudra Mitra¹ and Paul Hagan¹

ABSTRACT: machine cutting of rock is widely used in the coal mining industry. Developing an acoustic emission monitoring system that is capable of detecting changes in rock cutting conditions has the potential to enable the development of a real-time control system which can optimise the operation of rock cutting machines such as longwall shearers and roadheaders. A laboratory-based study was undertaken to record acoustic emissions during rock cutting using a linear rock cutting machine. The study considered a range of variables that might impact design of a control system including rock type, cutting depth, cutting speed and location of an acoustic emission sensor. The study found significant differences in the frequency domain of acoustic emissions with changes in cutting configuration including rock type.

INTRODUCTION

Mechanical cutting is extensively utilised in the mining and tunnelling industries as the primary means of rock excavation. In most instances, control of cutting machines is dependent on auditory and visual observations of a skilled operator. There is an increasing amount of excavation required in harsh and extreme underground environments. When combined with the desire to ensure high levels of machine utilisation and productivity, the development of systems that will enable semi-autonomous machine operation is attractive to OEM and operators alike.

Mechanical rock excavation is a costly process that can require large amounts of capital and consumes large amounts of energy. The efficiency of the rock cutting process which can be degraded with wear of the cutting tools, has a direct impact on a machine's productivity. Quite often monitoring of the state of wear is based on visual inspection of the cutting tools which is not only hazardous but also results in downtime of the machine. An on-line monitoring system would improve a number of operational aspects including reducing the frequency of unnecessary machine downtime for inspections; eliminating unnecessary cutter tool change-out and consequently decreasing overall cutting tool costs; reducing energy consumption; and, increased safety by eliminating the need for physical inspections of the working face.

Research to develop sensors has been undertaken sporadically since the early 1960s particularly in the area of horizon control of longwall shearers in the coal industry but it has been met with limited success (NASA, 1982). It was found that the effectiveness of many of these investigations was constrained by the local geology. The most successful system was based on natural gamma radiation and deployed on longwall shearers in the United Kingdom (Marshall, 1989). In the 1990s, efforts continued with a focus on more recently developed technologies that might allow more universal application in a greater range of rock conditions and machine types including x-ray fluorescence, synthetic Doppler radar, infrared thermography and vibration (Pazuchanics and Mowrey, 1991).

It is known that solids emit low-level acoustic emissions or seismic signals when subjected to changes in induced stress or deformation (Kaiser cited in Hardy, 1981). Acoustic emissions (AE) are mechanical vibrations that propagate through solid, liquid or gaseous materials (Hardy, 2003). While much of the focus of measuring acoustic emissions during cutting has been limited to the metal machining industry, there is scope for application to rock cutting.

PROJECT OBJECTIVES

Williams and Hagan (2006) reported variations in the nature of the acoustic signal with changes in rock cutting conditions. Improvements were subsequently made to the instrumentation and test procedure that principally provided a capability to measure AE signals at high sampling rates.

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A follow-on project was undertaken to assess whether differences in rock cutting configuration would be manifested in the higher frequency spectrum of the AE signal. In particular, to determine whether a unique characteristic or AE "signature" could be ascribed to a cutting configuration and that any changes to the cutting configuration would result in a different signature. Monitoring the AE signal could then form the basis of a real-time monitoring system.

The test program involved changes in cutting depth, cutting speed, positioning of the sensor and material being cut. Fast Fourier Transforms were used to determine the frequency content in the AE signal.

TEST APPARATUS

The linear rock cutting machine used in the experiments was a modified Invicta 6M Shaping machine as shown in Figure 1 located in the *Machine Cuttability Research Facility* at UNSW.



Figure 1 - Linear rock cutting machine used in cutting tests

The data acquisition system incorporated a Physical Acoustics Corporation A E transducer model number R15 α as shown in Figure 2. The transducer has an operating frequency range of 50 – 200 kHz.



Figure 2 - The AE transducer sensor, model R15 α

The sensor was connected to a PAC 2/4/6-C preamplifier. The values 2/4/6 represent the three selectable gain settings of 20 dB, 40 dB and 60 dB respectively and bandwidth of 10 kHz – 900 kHz. The gain was adjusted to maximise the acoustic signal.

The signal conditioning unit was a National Instruments (NI) SCXI – 1100 analogue input module. This unit is suited to high performance signal conditioning. The analogue signal was then fed to a NI 6229M analogue to digital card having a 16 bit analogue input resolution and a maximum sampling rate of 250,000 samples per second (250 kS/s).

DASYLab was used as the data acquisition software to control the hardware and store the data from each test. The arrangement of the data acquisition system is shown in Figure 3.

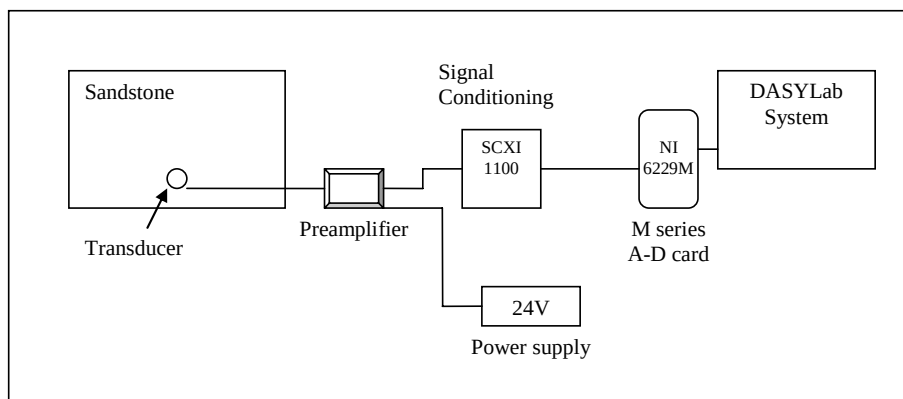


Figure 3 - A schematic of the data acquisition system used in the study

TEST PROCEDURE

Three material types were used in the study – sandstone, coal and a gypsum-based casting plaster. The sandstone block had dimensions of 395 mm long and 265 mm wide with a UCS of approx 40 MPa. The coal sample was encased with a protective plaster casing that provided confinement during cutting. During a cutting test, the sample was fixed to the table of the shaping machine.

Preliminary cutting tests were undertaken to:

- ensure compatibility of the various elements in the data acquisition system,
- confirm the transducer had sufficient sensitivity to detect an AE signal during rock cutting,
- adjust the gain on the preamplifier for maximum sensitivity,
- determine an appropriate data sampling rate,
- configure the software and hardware settings, and
- determine the level of background noise.

RESULTS

Initial recorded data

The magnitude of the acoustic signal against time was plotted for the first series of tests. These graphs indicated the level of energy release was not consistent during rock cutting and mirrored the changes in force on the cutting tool with time. An example of an acoustic/time graph when cutting in sandstone is shown in Figure 4.

The acoustic/time graph was useful to identify changes in cutting conditions as these were reflected in the nature of the acoustic signal. Figure 5, for example, illustrates an arrangement where in one test the cutting tool was made to pass through coal and plaster. In between cutting the coal and plaster, there was a small air gap where the cutting

tool was temporarily disengaged from cutting material albeit for a small nick of coal. The corresponding acoustic/time graph shown in Figure 6 shows changes in the pattern of the acoustic signal at the different stages of cutting.

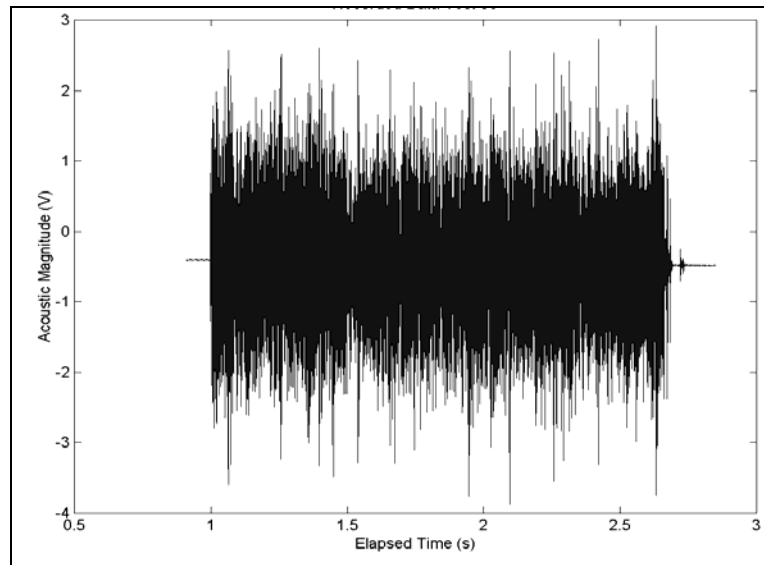


Figure 4 - A typical plot of the acoustic signal during cutting in sandstone

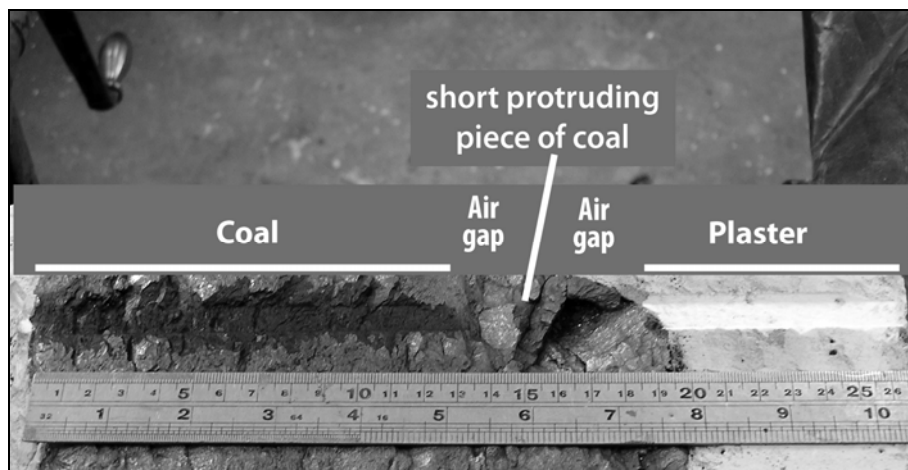


Figure 5 - The test arrangement involving cutting in coal and plaster

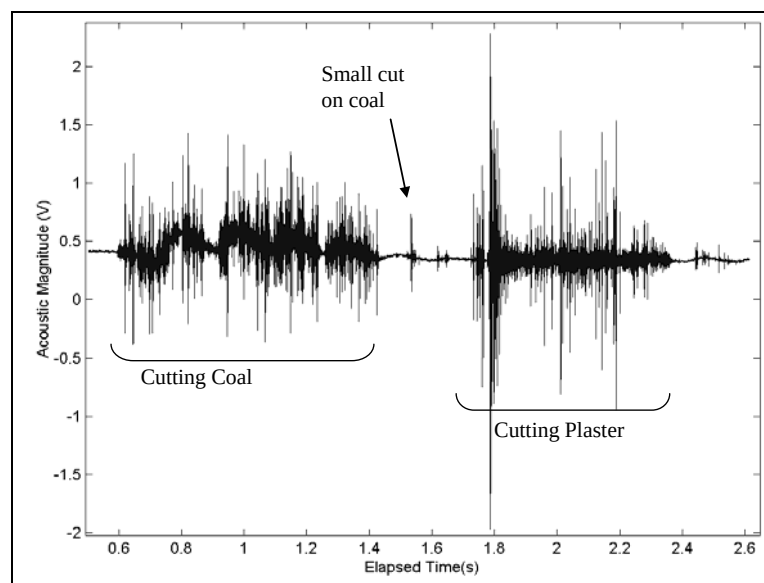


Figure 6 - Changes in the measured acoustic signal when cutting in coal and plaster

SIGNAL ANALYSIS

While the acoustic/time graph can provide a visual contrast with changes in rock cutting condition, a better method was needed that could quantify the changes in acoustic signal.

To this end the frequency domain within the acoustic signal was examined along the lines of Shen and Hardy (1996). They reported on the presence of peak frequencies, or *Major Dominant Frequencies* (MDFs) in an acoustic signal. The objective in this project was to determine whether a unique set of frequencies, the AE signature, could be defined which would characterise a given cutting configuration.

Analysis of the frequency content in the acoustic signal was based on Fast Fourier Transforms (FFT). MATLAB, a commercially available software package, was used for the FFT analysis.

Sampling rate

One important factor that can constrain the usefulness of frequency analysis is the range of frequencies in the signal that can be considered. The Nyquist criterion states that the maximum useful frequency that can be discerned within an analogue signal, termed the folding frequency, is equal to half the sampling frequency.

Various research undertaken in the 1980s and 1990s such as that by Pazuchanics and Mowrey (1991) at the USBM reported acoustic signals were recorded at sampling rates of up to 1000 samples/second (S/s) in real-time and up to 10 kS/s when recorded to tape and played back later at slow speed. This levels equate to a folding frequency of between 500 Hz and 5 000 Hz. This was a limit imposed by the technology available at the time but given current data acquisition systems are capable of achieving real-time sampling rates at least ten times this level then frequencies of 100 kHz or more can now be considered.

A series of tests were conducted at sampling rates ranging between 100 S/s and 200 kS/s. Graphs of the frequency spectrum at sampling rates of 10 kS/s, 100 kS/s, 150 kS/s and 200 kS/s are shown in Figures 7 to 10 respectively. These sampling rates correspond to folding frequencies of 5 kHz, 50 kHz, 75 kHz and 100 kHz respectively.

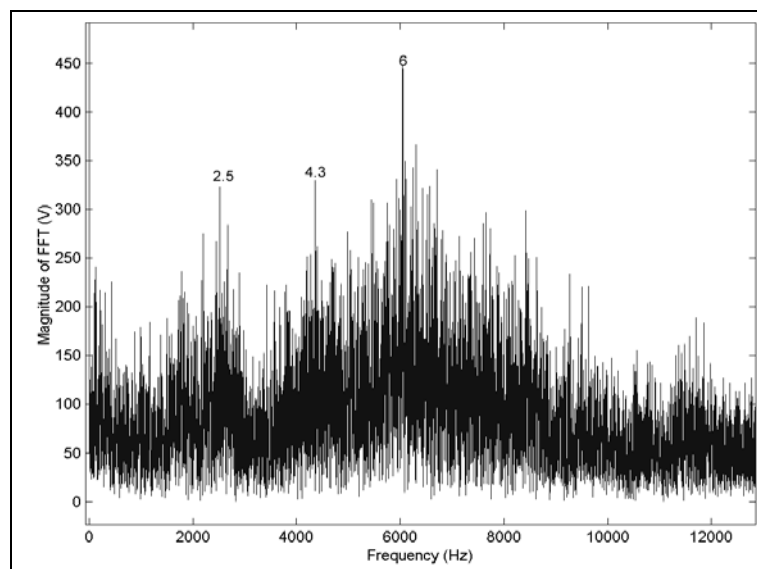


Figure 7 - Frequency spectrum graph indicating three Major Dominant Frequencies in the recorded data with a sampling rate of 10 kS/s

At a sampling rate of 10 kS/s, three MDFs were discernible of 2.5, 4.3 and 6 kHz. At 100 kS/s, seven MDFs are evident within the range of 16 kHz to 48 kHz. With a further increase to 150 kS/s, the same number of MDFs was evident though the range of MDF frequencies increased to 61 kHz. At 200 kHz, the MDFs were very similar to those found at 150 kS/s. It was noted that as the sampling rate increased, the strength of the MDFs compared to the background level was more discernible.

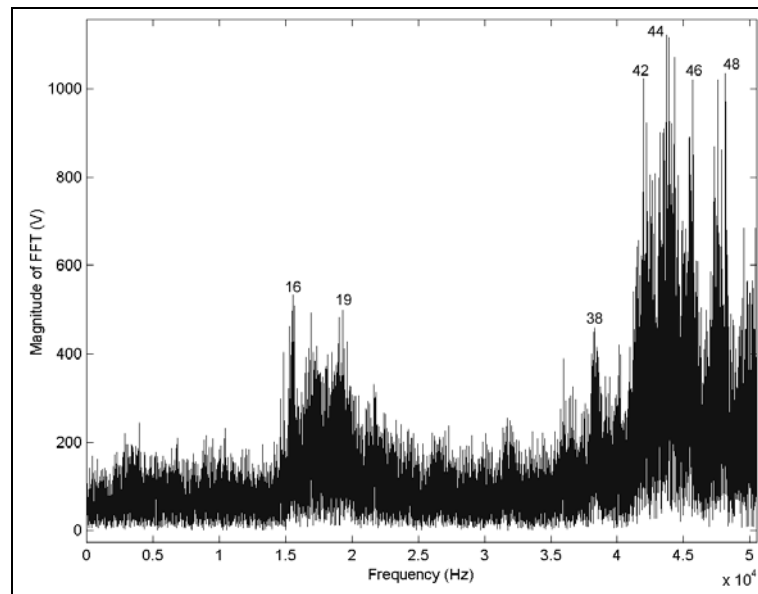


Figure 8 - With an increase in sampling rate to 100 kS/s, seven Major Dominant Frequencies were evident

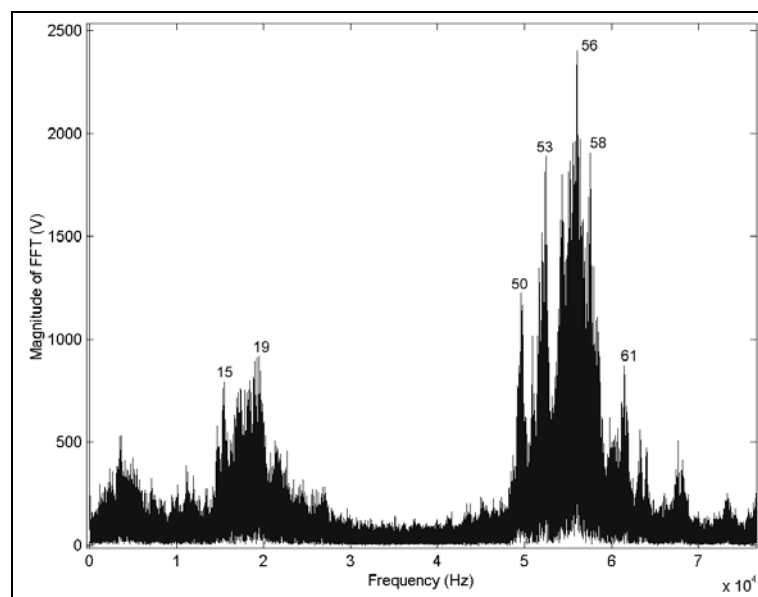


Figure 9 - At 10 kS/s, seven Major Dominant Frequencies were again evident

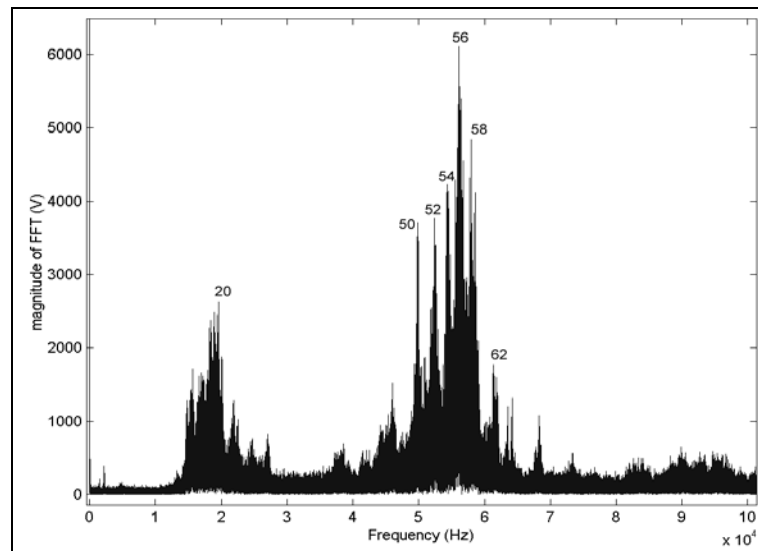


Figure 10 - At a sampling rate of 200 kS/s, seven Major Dominant Frequencies were evident that were similar to those found at 150 kS/s

It is also important to note that at 200 kS/s the highest MDF was 62 kHz which is roughly 60% of the folding frequency indicating that this sampling rate provides sufficient leeway for higher frequencies to be identified. It was decided to conduct all the experiments at a sampling rate of 200 kS/s corresponding to a folding frequency of 100 kHz.

Cutting speed

The effect of a change in cutting speed on the nature of the acoustic signal was examined in sandstone, coal and plaster. Two cutting speeds were selected of 0.150 m/s and 0.846 m/s representing a five-fold difference in speed. The frequency analysis found little significant difference between the two speeds for all three materials with the same MDFs being identified.

Depth of cut

Two levels of depth of cut were examined in sandstone (3 mm and 6 mm) and three levels in coal (3 mm, 6 mm and 9 mm). The MDFs in each set of results were compared. Again there was very little change in the MDFs with cutting depth for each material. Figures 11 and 12 show the frequency spectrum for 3 mm and 6 mm in sandstone respectively.

Transducer location and attachment

The objective of this series of tests was to examine the effect on the acoustic signal of changes in the method of attachment of the acoustic transducer and its location with respect to the cutting tool.

In terms of attachment, beeswax and "super glue" were used. The frequency analysis indicated little difference in the MDFs between the two methods of attachment. There was though an appreciable difference in the amplitude of the acoustic signal indicating better coupling of the transducer and less attenuation of the signal when glue was used. This was reinforced by the need to use a higher gain setting on the pre-amp in the tests when using beeswax.

Location of the transducer was found to have a substantial impact on the frequency response in the acoustic signal. As with the method of coupling, a change in location affected the level of attenuation of the acoustic signal. Also there was a change in the level of noise in the signal, making it more difficult to define the MDFs in certain configurations.

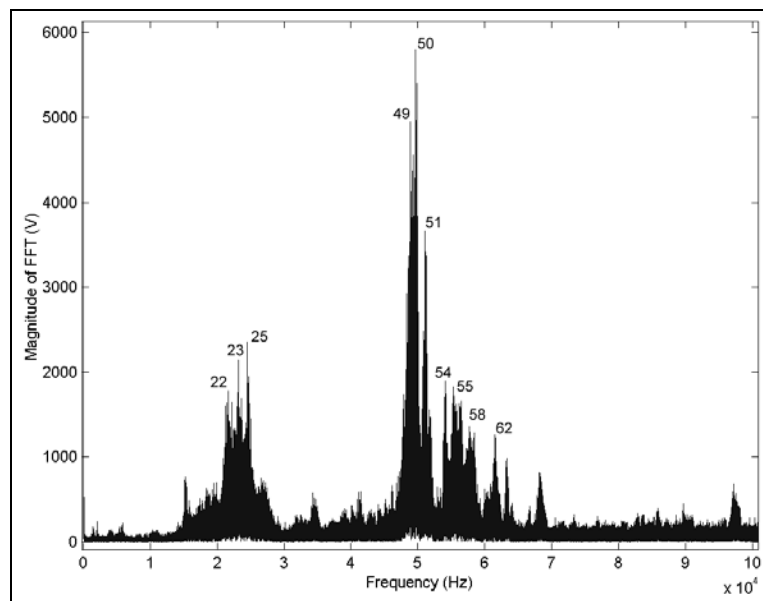


Figure 11 - Frequency spectrum graph for a 3 mm depth of cut in sandstone with the transducer glued on the cutting tool

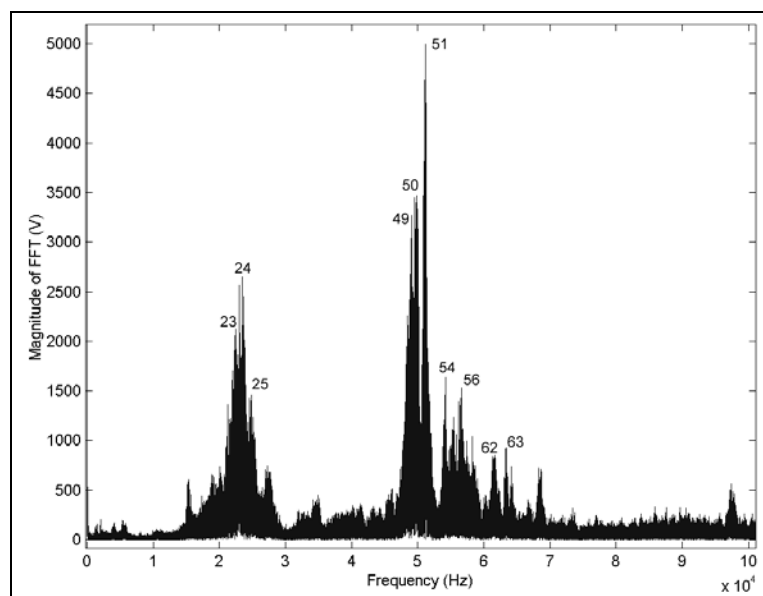


Figure 12 - Frequency spectrum graph for a 6 m depth of cut in sandstone indicating a similar set of MDFs as that found for 3 mm. Transducer glued on the cutting tool

The frequency spectrum graph shown in Figure 11 was for the case when the acoustic transducer was glued directly to the cutting tool post holder. Figure 13 shows the frequency spectrum graph for the same cutting configuration except the transducer was glued to the surface of the sandstone block. When comparing these two configurations it can be seen that:

- there was less attenuation in the acoustic signal when the transducer was mounted directly on the cutting tool;
- there was a clearer definition of the MDFs when the transducer was mounted on the cutting tool;
- there was a shift in the MDFs at the lower frequencies around 20 kHz between the two configurations with a slightly lesser change at the higher frequencies.

When the test was repeated using the coal sample, similar results were observed.

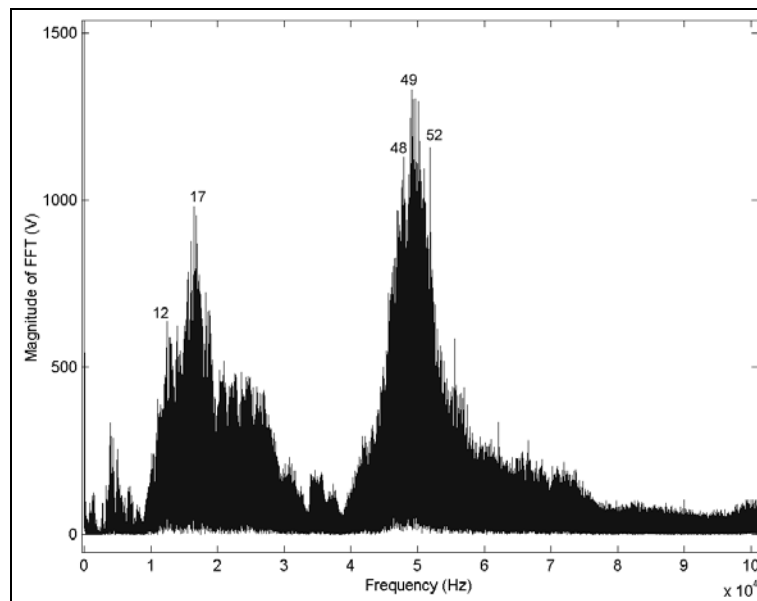


Figure 13 - Frequency spectrum graph for a 3 mm depth of cut in sandstone with the transducer glued on the sandstone block.

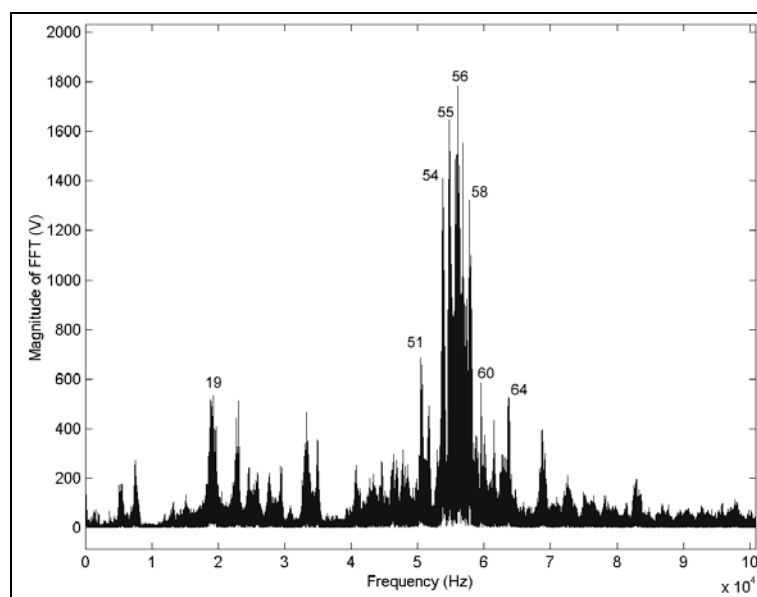


Figure 14 - Frequency spectrum graph for a 6 mm depth of cut in coal with the transducer glued on the cutting tool.

Rock type

When comparing the test results for different material types, there were significant differences in the amplitude of the signal as well as the MDFs.

Figures 14 and 15 show the frequency spectrum when cutting in coal and in plaster. The difference in amplitude would be expected as they correspond to the differences in the energy of cutting between coal and plaster.

There was also a distinct difference in the set of MDFs especially when compared to that found when cutting in sandstone shown earlier in Figure 12.

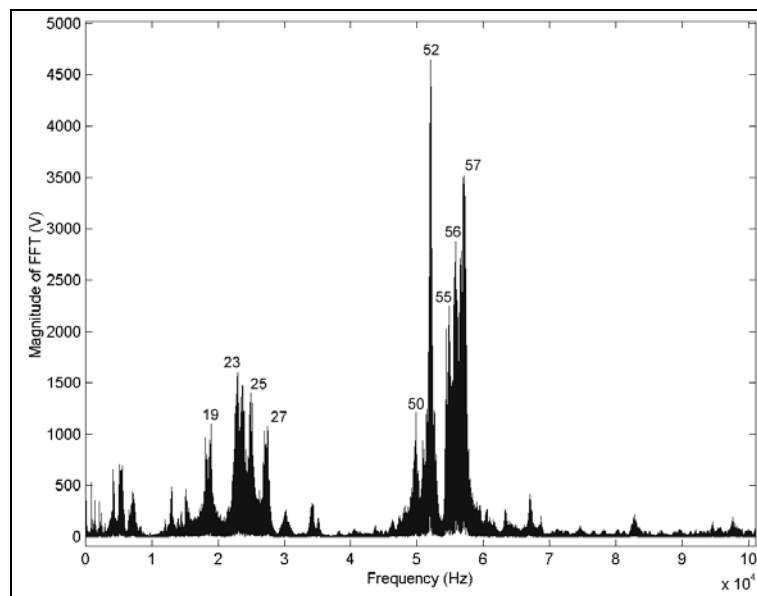


Figure 15 - Frequency spectrum graph for a 6 mm depth of cut in plaster with the transducer glued on the cutting tool

Following on from this, a distribution of the commonly occurring MDFs was compiled based on all the tests for each material. Figure 16 shows a graph of the frequency of occurrence of the MDFs for the different cutting configurations when cutting in sandstone.

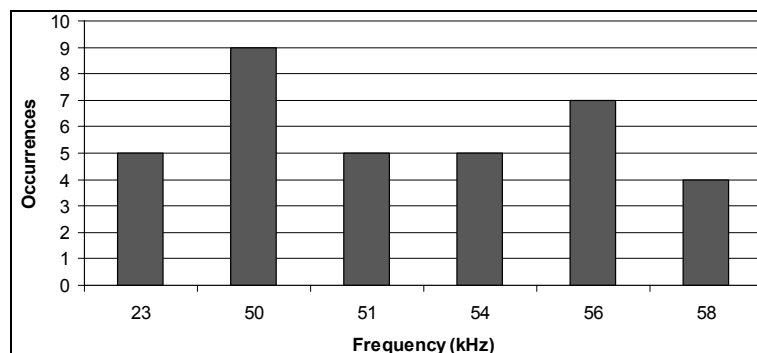


Figure 16 - Distribution of the most common MDFs for the different tests when cutting in sandstone

The MDFs were added, graphed and compared for each of the three materials. This was undertaken in order to determine the characteristic frequencies common to a particular cutting parameter.

The two most common MDFs were 50 kHz and 56 kHz for sandstone; 54 kHz and 55 kHz for coal; and, 56 kHz and 57 kHz for plaster. The number of tests used to determine each of the common MDFs for sandstone, plaster and coal were ten, seven and seven respectively.

CONCLUSIONS

At a sampling rate of 200 kS/s, a set of Major Dominant Frequencies (MDFs) could be defined in an acoustic signal that can be used to characterise the Acoustic Emissions (AE) emitted under differing rock cutting configurations.

The MDFs only became evident at sampling rates in excess of 100 kS/s which is well above that which could be achieved with commercially available equipment in the 1980s and 1990s.

This sampling rate of 200 kS/s allowed for MDFs of up to 100 kHz to be identified. At this sampling rate the highest MDF was approximately 65 kHz.

Several different rock cutting configurations were examined of which some had more of pronounced effect than others.

In the case of the cutting depth and cutting speed there was very little significant effect on MDFs. Significant differences were observed though with respect to transducer location and method of attachment. The best signal was found with the AE transducer attached to the cutting tool using glue as opposed to being attached to the rock using beeswax.

In the case of cutting in three different material types (sandstone, coal and plaster), there were significant differences in the MDFs. These MDFs defined the signature for a cutting configuration and indicate that a monitoring system could be developed to maintain cutting within a certain rock horizon.

The objective of this project was to determine whether measuring AE during cutting could be used as a basis for monitoring machine performance. It was found that AE were more sensitive to changes in some rock cutting configurations than others. Further work is necessary to confirm the usefulness of this as for real-time application in machine control and when using multiple cutting tools on a cutting head.

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A STUDY ON THE EFFECT OF MOISTURE CONTENT ON ROCK CUTTING PERFORMANCE

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ABSTRACT: Road headers and other cutting machines are often used for excavation in the mining and tunnelling industries. During excavation, changes in the properties of the rock mass can alter which adversely impact machine performance. Also it has also been found that an estimation of machine performance differs from that actually achieved in the field. It has been postulated that one reason for this variation in performance could be due to a rock's moisture content.

This paper outlines the results of a study that examined the impact on rock cutting performance of changes in the moisture content of sandstone. The study found both cutting and normal forces decreased with moisture content by up to 40 and 49% respectively when cutting a saturated sample compared to the dry rock sample. Reductions were also found in specific energy, cutter pick wear and some other rock properties.

INTRODUCTION

Research into machine mining as a means of rock excavation has been on-going particularly since the 1950's when there was intense interest in mechanisation to improve productivity. From that time a number of theories have been developed to estimate the magnitude of cutting force and other cutting parameters in the machine excavation of rock (Nishimatsu, 1993). In addition many empirical studies have quantified the effect of changes in tool and operational variables on cutting performance (Hood and Roxborough, 1992). Roxborough (1987) noted that in many cases the force required to cut rock is primarily governed by its strength. In some instances consideration has also been given to other properties of the rock mass that may influence cutting performance. One aspect that has been given little attention is the moisture content of a rock. It has been postulated that the presence of water can alter a rock's behaviour particularly with respect to the energy necessary to fracture the rock and crack propagation though little has been published on its effect on rock cutting performance.

A study was undertaken to determine what changes if any might occur in rock cutting performance with moisture content. The study involved testing sandstone in the dry and saturated states and at various levels of moisture content in between, measuring changes in cutting forces, energy and material properties.

PREVIOUS STUDIES

Rock properties

Various studies have noted the presence of water in rock can alter the properties and behaviour of a rock mass. For example some rock can be reactive to water especially if clay is present as it will soften the material loosening its structure and increasing its deformability leading to a reduction in the overall strength of the rock mass (Brady and Brown, 2004). The presence of water in a rock mass can also reduce the shear resistance of joint sets dependant on the level of pore water pressure (Budhu, 2000).

In terms of its mechanical strength, Roxborough (1987) stated that "the mechanical strength of a rock commonly reduces as its moisture content increases." Vasarhelyi and Van (2006) found "water content is one of the most important factors influencing rock strength...the strength decrease is remarkable after only 1% water saturation."

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Cutting mechanisms

Most pick-type cutting tools fracture rock by inducing a tensile failure. As the tensile strength of rock is approximately one-tenth of its compressive strength, picks are relatively effective in mechanical rock breakage albeit they limited to use in soft rock types such as those normally associated with coal formations including coals, shales and sandstones. Picks and other cutting tools are used as a means of funnelling energy from the cutting machine and focusing it to fracture rock (Fowell, 1993). As the power of cutting machines is limited, any change in the amount of energy necessary to fracture rock will translate to changes in cutting rates.

The objective of this study was to measure changes in cutting performance that might be brought about by changes in the moisture content of rock. In particular, sandstone samples at various levels of moisture content were tested with respect to changes in cutting force, normal force, specific energy, yield, pick wear rate, Cerchar Abrasivity Index (CAI) value, compressive strength and Young's Modulus.

TESTING METHOD

Rock sample preparation

In this study, rock test samples were diamond cored from a single block of Mt White Sandstone, the characteristics of which as contained in Table 1. Sandstone was chosen due to it being highly porous and permeable and hence being able to absorb water more readily than other rock types. It is also a rock frequently excavated using pick-type rock cutting machines.

Test samples with a diameter of 57 mm (large diameter test core sample) were obtained for rock cutting tests and a diameter of 43 mm (small diameter test core sample) for the rock strength tests.

Table 1 - Characteristics of Mt White Sandstone

Geological Name	Argillaceous quartz sandstone
Geological Age	Triassic
Petrologic Description	Medium-grained quartz sandstone with a predominantly argillaceous matrix.
Bulk Density	2.27 tonnes/m ³
Absorption by Weight	4.4%
Modulus of Rupture	8.9 MPa (dry) and 2.5 MPa (wet)
UCS	57 MPa (dry) and 22 MPa (wet)

Moisture content

As the purpose of the study was to measure changes in cutting performance with moisture content, it was important to minimise the impact of changes between different test samples. Hence the moisture content was altered in a single rock type. Two techniques were investigated to achieve a desired level of moisture content in a test sample. Moisture content was determined in accordance with the International Society for Rock Mechanics (ISRM, 1981).

The first technique involved measuring the mass of cores after they had been placed in a drying oven for fourteen days. This established the mass of the "dry" core samples. Ten cores in all were weighted; five each of the small and large test core samples. At one minute intervals a sample was placed in water. After a period of time, a sample was removed and weighted. Soaking times varied between 5 minutes and 30 minutes at 5 minute intervals then at 40, 60, 90, 120, 150, 180, 660, 960 and 1440 minutes.

Results of the soaking testing are summarised in Figure 1. The graph shows measurements for each of the individual test samples and the average of the two sample sizes. The graph indicates different

rates of water uptake between the two sample sizes as would be expected due to differences in volume and hence length of the migration path of water. Both sample sizes achieved the equivalent of nearly 50% of the saturated water level within the first minutes of soaking. Thereafter the rate decreased with time with nearly 90% of the saturated water content being achieved within 120 minutes for larger sample and 40 minutes for the smaller sample.

The second technique involved placing fully saturated samples in a drying oven and weighting the mass at predetermined intervals. Results of drying test are shown in Figure 2.

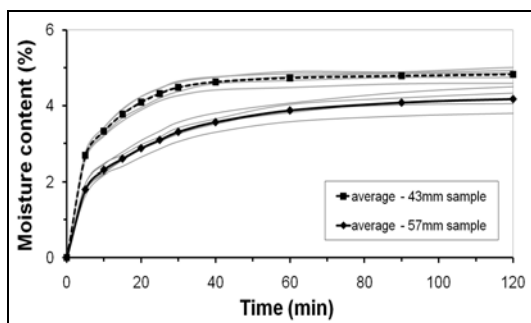


Figure 1 - Variation in moisture content with time in soaking two size core samples

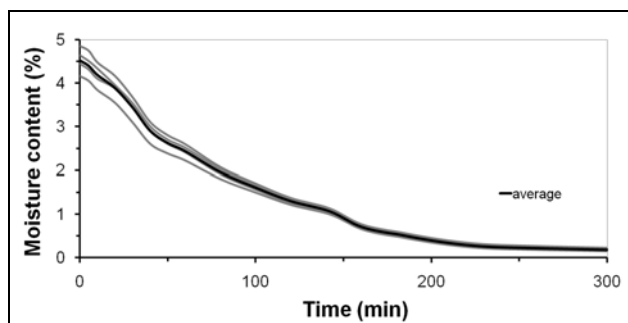


Figure 2 - Variation in moisture content with time on drying of a 57 mm core sample

The first technique proved the most practical as the soaking times to achieve the different levels of water content were of short duration and hence more controllable. The various levels of moisture content selected for the study are indicated in Figure 3 and are summarised in Table 2.

Table 2 - Targeted and measured moisture content for test samples

	moisture content		soak time
	target	as measured	
	0.5%	1.15%	1.5
	1.5%	1.71%	4 min
	2.5%	2.51%	13 min
	3.5%	3.28%	35 min

Measurement of the five large samples found after being left for several weeks at room conditions in the laboratory they retained some moisture. The average moisture content of the samples in the "air-dried" condition was 0.22%.

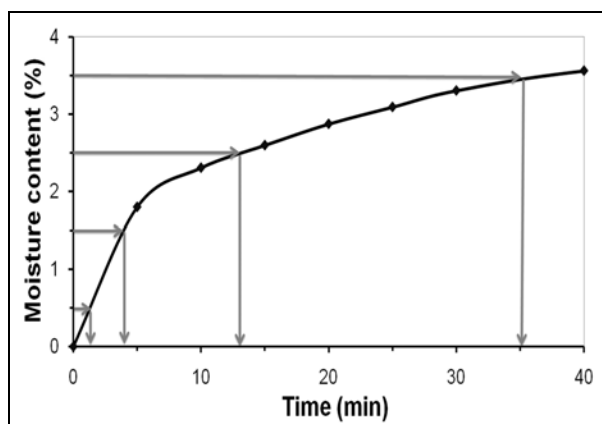


Figure 3 - Selected moisture content for testing and corresponding required soaking time

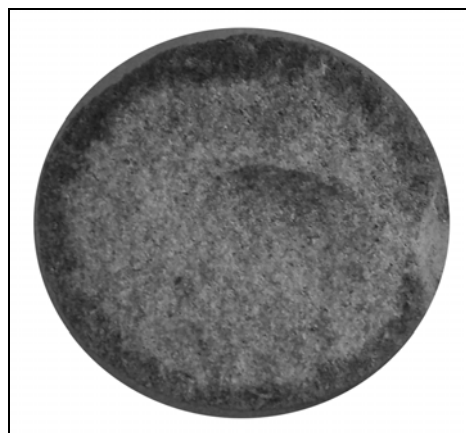


Figure 4 - Extent of water absorption indicated by a discoloured ring around the edge of rock test sample

Uneven diffusion

It was observed during these tests that the samples appeared to be drying unevenly and that they become hard and brittle. This raised the issue about the nature of the moisture distribution within the samples.

A further test was undertaken to ascertain the nature of the moisture profile across the cross-section of the samples. This was done by submerging samples in water containing a coloured dye. After being submerged for several minutes the sample was removed. On breaking the sample, the cross-section revealed water had migrated inwards from the outer surface with the dye forming a ring pattern as illustrated in Figure 4. This indicated that while the outer part of the core was fully saturated, the inner core remained dry. Hence the moisture content in the rock sample was uneven.



Figure 5 - Linear rock cutting machine



Figure 6 - Cerchar test apparatus

In order to promote more even distribution of the water absorbed by a sample, each sample to be used in the cutting tests would after being removed from water at the pre-determined time indicated in Table 2 be wrapped in plastic and left for at least 72 hours prior to testing. This would allow time for the moisture to diffuse evenly throughout the sample.

Test apparatus

The assessment of changes in cutting performance was undertaken using equipment in the Machine Cuttability Test Facility at UNSW. A modified linear shaping machine with a tri-axial dynamometer as shown in Figure 5 was used to measure the forces and energy of cutting. An Instron test machine with a 500 kN capacity was used for the compression strength tests.

The Cerchar Abrasivity Index test was performed with standard Cerchar testing apparatus as shown in Figure 6.

RESULTS

Material properties changes in the strength of the samples at various levels of moisture content were undertaken with the small samples. Three levels of moisture content were examined, these being:

- oven dried condition, 0% moisture content;
- "air-dried" condition, 0.2% moisture content; and
- saturated condition, 5.2% moisture content.

Results of testing at the three levels are summarised in Table 3.

The results indicate a significant reduction in the compressive strength of the rock with the saturated sample of 68%. It was found a significant reduction in strength of 63% occurred even at the lowest

level of moisture content compared to oven dried test rock sample. This finding is in line with that observed by Vasarhelyi and Van (2006).

Table 3 - Summary of effect of moisture content on material properties

<i>moisture content</i>	<i>UCS (MPa)</i>	<i>strength reduction</i>	<i>Young's Modulus (GPa)</i>
0.0%	60.3	-	6.3
0.2%	22.1	63%	4.1
5.2%	19.4	68%	3.2

Cutting performance

A summary of the effect of changes in moisture content on rock cutting performance is summarised in Table 4.

For each of the "dry" results (0.0%) and the "saturated" results (4.6%), the values stated in Table 4 represent an average of four replications of linear cutting tests undertaken in each of two test samples; that is eight replications in all for each of the dry and saturated samples. This large number of replications was undertaken so as to increase confidence in the results at both extremes in moisture content. In the case of all other results, the values represent the average of four linear cutting tests made with each sample.

In terms of cutting force and normal force, there were significant reductions between the dry and saturated samples of approximately 40% and 49% respectively for the cutting and normal forces.

Table 4 - Effect of moisture content on cutting performance

<i>test sample condition</i>	<i>moisture content</i>	<i>Force (kN)</i>		<i>Yield (m³/km)</i>	<i>SE¹ (MJ/m³)</i>	<i>CAI²</i>	<i>Wear (mg/m)</i>
		<i>Cutting</i>	<i>Normal</i>				
<i>dry</i>	0.0%	1.24	0.89	0.0713	17.3	2.7	7.1
	0.2%	0.88	0.57	0.0463	18.1	-	1.5
	1.2%	0.91	0.50	0.0656	13.6	2.6	-
	1.7%	0.77	0.41	0.0671	11.3	2.5	0.2
	2.5%	0.89	0.66	0.0695	12.9	2.6	4.5
	3.3%	0.74	0.45	0.0719	9.0	1.5	0.8
<i>saturated</i>	4.6%	0.74	0.45	0.0691	10.8	2.4	1.8

Note: 1. Specific Energy 2. Cerchar Abrasivity Index value

As indicated in Figure 7, the greatest reductions in force were registered at the smallest level of moisture content of 0.2% level which was when the rock was in the air-dried or ambient conditions. At this level of moisture content the reductions were 29% and 36% respectively for cutting and normal forces which represent nearly 72% of the total force reductions. Consequentially by the time the sample contains even the smallest amount of moisture, there is a step change with most of the gains in terms of a reduction in force having been achieved. Any further increase in moisture content did not result in any further significant perceived changes given the scatter in test results though on average there were further reductions of 11% and 14% respectively in cutting and normal force.

The results indicate no further step changes as the sample becomes fully saturated once the sample has absorbed even the smallest amount of water. In summary, given the range of moisture content examined in this study it would seem that it is not the quantum of water that is not important but whether the rock contains any water at all.

In terms of the yield of material from rock cutting, the results indicate moisture content had little impact within the bounds of experimental error. There was little significant difference in yield between the dry and saturated samples.

In terms of Specific Energy, which is a function of cutting force and yield, since yield was found to have no effect then any variation in Specific Energy with moisture content should mimic that of cutting force. This was found to be the case as shown in Figure 8. The difference in Specific Energy between the dry and saturated rocks was 38% which is similar to that determined for cutting force.

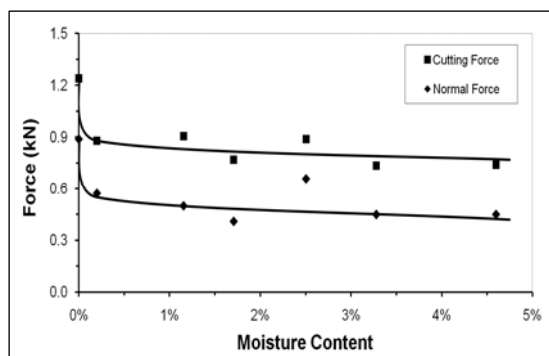


Figure 7 - Effect of moisture content on forces in rock cutting

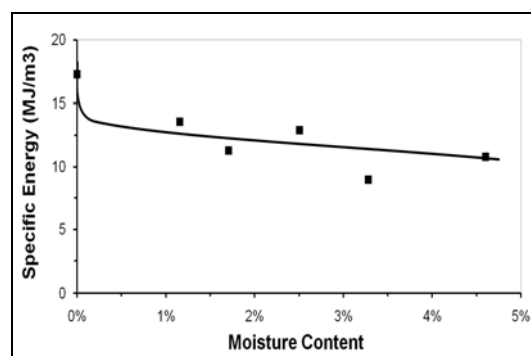


Figure 8 - Effect of moisture content on Specific Energy of rock cutting

In terms of CAI, there was a slight reduction of 13% between the dry and saturated samples. However unlike the nature of reduction in forces, there was no evidence of any major step change, CAI seemed to decrease steadily with moisture content.

Finally with respect to impact wear as a result of rock cutting, there was again a step change although on a much greater scale. Wear of the tool in the saturated sample was about 20% of that found in cutting the dry sample. As shown in Figure 9, similar reductions were observed in all cases of the sample containing water. It should be cautioned though that in the case of wear, only one measurement was determined for each sample as it represents the cumulative wear from all four replications of cutting tests.

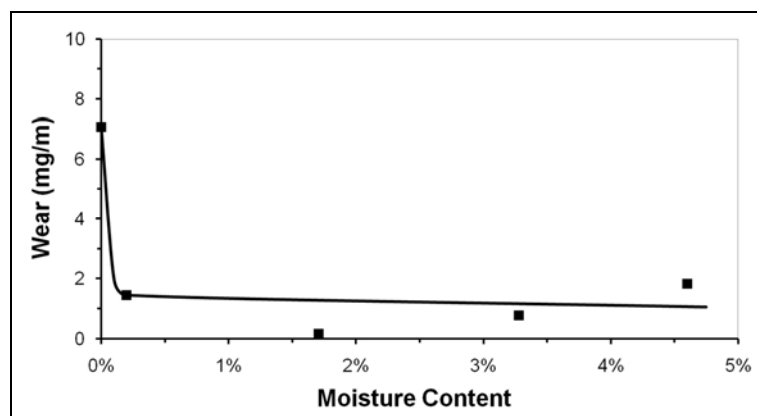


Figure 9 - Effect of moisture content on wear of the cutting tool

CONCLUSIONS

In conclusion, it was observed that there was a significant decrease in most of the rock cutting performance parameters with moisture content. The difference in performance was greatest when comparing cutting a dry sample to a saturated sample. The test samples used in the study were 57 mm sandstone cores.

Reductions of up to 40% and 49% were found for cutting and normal forces, 38% for Specific Energy, 80% for impact wear of the cutting tool and, 68% in compressive strength. There was little evidence of any significant change in rock abrasiveness as measured by CAI and, in rock yield.

Significantly in most cases the magnitude of the reduction in performance parameters was greatest with only the slightest addition of water to the rock. The inherent moisture content of the sandstone test samples if left at ambient conditions was found to be 0.22%; as opposed to a saturated sample containing 4.6 to 5.2% water. Even at the low moisture content of 0.22%, the reduction in both cutting and normal forces was nearly 72% of the total potential reduction in forces.

Whether there is a causal relation between the reduction observed in compressive strength and in the rock cutting performance parameters was determined in this study. A common mechanism that may account for these reductions may be found in fracture mechanics and the part that water plays in modifying the characteristics of crack propagation.

The results indicate that it is not so much that the rock needs to be saturated but that any minor amount of water in the rock is sufficient to reduce the levels of the various cutting performance parameters.

Interestingly this would suggest that as rock in its natural state is likely to contain some moisture then there is unlikely to be any significant change in performance as a cutting machine excavates rock. Importantly also there is unlikely to be any benefit from introducing water to the cutting face for the purpose of reducing performance parameters.

An exception to this general rule might be if rock excavation takes place in or adjacent to high temperature rock resulting in the *in situ* moisture content of the rock being negligible. In this case there would benefit from the addition of water at the face that would lead to a reduction in cutting forces and the other performances parameters.

Amongst some of the many questions that arise from this study, the following merit follow-up work.

- Can the results be duplicated in different sandstones and other rock types? What effect does permeability and porosity have on the magnitude of reduction in performance parameters?
- Can the performance benefits be repeated in the same rock after more test replications?
- What is the effect on the performance parameters with rock at very low moisture content?
- Are there any differences between free water on the rock surface and water within the rock matrix? This would impact on whether water sprays at the face would be sufficient to gain any benefit or whether water has to be injected into the rock mass.
- Are the test results sensitive to the method of achieving a dried test sample?
- How sensitive are the test results to the testing environment and what effect do environmental factors such as temperature and pore water pressure have on the results?

ACKNOWLEDGEMENTS

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AN ASSESSMENT OF THE IMPACT OF STYLUS METALLURGY ON CERCHAR ABRASIVENESS INDEX

Julian Stanford¹ and Paul Hagan¹

ABSTRACT: The Cerchar test is increasingly being used as a method to assess the abrasivity of rock in drilling and machine mining applications. A study was undertaken to determine the effect of changes in the metallurgy of the steel styli that is used in the test procedure on the Cerchar Abrasiveness Index (CAI) value. The study involved testing seven different metal types heat treated to the same hardness level and one steel type at nine different hardness levels.

The study found there was little change in the CAI value with different steel types however hardness of the steel styli was found to affect the CAI value. CAI varied inversely with hardness of the steel styli.

INTRODUCTION

From the time of its earliest development in the mid 1960s, the Cerchar Abrasiveness Index (CAI) test has gained increasing popularity as a means of assessing the abrasivity of rock. This is in part due to it being a simple, fast and effective method of measuring and comparing the abrasivity of different rock samples (Michalakopoulos *et al*, 2006). The test has found common use within the mining and tunnelling industries to estimate wear rates and cost of equipment replacement. Indeed, the Cerchar test is now considered by some as one of the 'standard' parameters for hardrock classification (Plinninger, Kasling and Thuro, 2004).

Over the years, the Cerchar test has been subject to significant study especially with respect to what effect test conditions and the geotechnical properties of rock might influence test results (Suana and Peters, 1982; Al-Ameen and Waller, 1994). One test parameter that has been subject to some debate is the metallurgy of the steel stylus (sometimes referred to as the 'pin' or 'needle') used in the Cerchar test, particularly with respect to its hardness. Currently there is no one standard that has been unanimously adopted and variants to the test continue to be used making comparison of results somewhat tenuous. Indeed, classifying results according to CAI might be misleading without knowing the precise specifications of the stylus used in the test.

This paper presents the results of a study that examined the effect of material properties and composition of the steel stylus on CAI test results. In particular, the study examined metal grade of the steel and its hardness.

THE CERCHAR TEST

The Cerchar test, and associated CAI, was developed at the Laboratoire du Centre d'Études et Recherches des Charbonnages de France (CERCHAR). The test was developed at a time of increasing mechanisation in the coal mining and tunnelling industries and with it a need to estimate likely production rates and operating cost in different rock conditions with different scales and type of equipment. A method of determining the abrasivity of rock is one important parameter needed in this estimation.

The importance of abrasivity is that it is directly related to the degree of wear that mining, tunnelling and drilling equipment such as roadheaders, longwall shearers and continuous miners is subjected (West, 1989; Atkinson, 1993).

The Cerchar test, for which a schematic of the apparatus is shown in Figure 1, involves scratching a steel stylus (annotated as Item 5 in Figure 1) under an applied static load of 70 N (Item 6), 10 millimetres across a rock surface that is held in place by a clamp (Item 1). Before each test, the tip of the stylus is sharpened to achieve a conical tip angle of 90°. Usually the test on each rock sample is

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repeated between three and six times, each time with a sharpened stylus. The length of the resulting wear flat on the stylus is measured under a microscope. The length is converted on the basis that a 0.1 mm wear flat equates to 1 CAI unit and the average of the replications is calculated and reported. The CAI value ranges in magnitude between zero and seven.

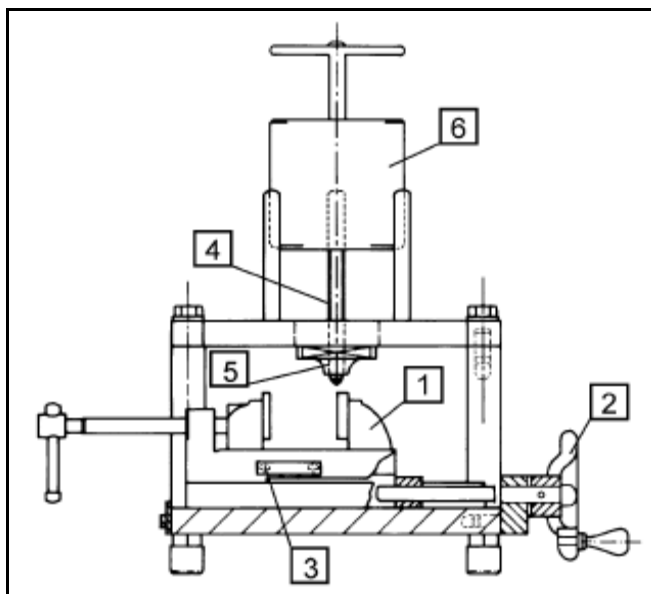


Figure 1 - Cerchar apparatus (after West, 1989)

The meaning attached to the value of CAI in terms of the degree of severity of abrasivity is summarised in Table 1.

Table 1 - Criteria for the Cerchar Abrasiveness Index

CAI value	Category (after CERCHAR, 1986)	Category (after Michalakopoulos et al, 2006)
0.3 – 0.5	Not Very Abrasive	Very Low Abrasiveness
0.5 – 1.0	Slightly Abrasive	Low Abrasiveness
1.0 – 2.0	Medium Abrasiveness to Abrasive	Medium Abrasiveness
2.0 – 4.0	Very Abrasive	High Abrasiveness
4.0 – 6.0	Extremely Abrasive	Extreme Abrasiveness
6.0 – 7.0	Quartzitic	-

STEEL STYLI METALLURGY

While the geometry of the steel stylus and the testing procedure are well documented and accepted, specifications of the steel stylus are not. CERCHAR (1986) specified a steel strength equivalent to an Ultimate Tensile Strength (UTS) of 2000 MPa. It is assumed that the criteria presented in Table 1 reflect this property of the steel stylus. West (1989) claimed, however, that steel treated to a Rockwell C Hardness (HRC) number of 40 gave the most representative results with respect to CAI, despite 2000 MPa reflecting a much higher steel hardness of HRC 57. Differences in some of the material properties of steel between these two standards are highlighted in Table 2. Given the extent of the difference in material properties of steel between these two specifications, it is likely that this would translate to a significant difference in wear of the stylus and hence magnitude of the CAI value.

Table 2 - Differences in material properties between two specified steel types

after	UTS (MPa)	Rockwell Scale (HRC)	Vickers (DPH HV/10)	Brinell (BHN 3000kg)	Scleroscope
CERCHAR (1986)	2000	57	633	595	76
West (1989)	1255	40	392	371	54

As the Cerchar test is in effect a measure of the difference in the relative hardness between steel and rock, the level of hardness of the steel stylus would be crucial to the amount of wear on the stylus and resulting CAI value. So long as the material properties of the stylus remain undefined, questions will remain about the significance of test results. Indeed it has been acknowledged that there is a problem with different steel qualities being used around the world (Plinninger, Kasling and Thuro, 2004; Verhoef, 1997).

Research Objectives

The aim of the study was to improve the usefulness, accuracy and knowledge of the Cerchar test by examining what effect changes in steel type and hardness have on the CAI value.

Selected Materials

Styli Metals

A total of seven different steel types were selected for the study. These were chosen to represent a cross-section of the steel types likely to be used for Cerchar testing in different laboratories around the world. They were selected in consultation with M. F. Dippert Pty Ltd and the steel sourced from Bohler-Uddeholm Australia.

In addition, a further nine styli were machined from Silver Steel heat treated to hardness levels of HRC 15 (untreated), 24, 29, 35, 40, 45, 50, 55 and 60 respectively. The properties of the various steel styli used in the study are summarised in Table 3 and a sample of the machined styli is shown in Figure 2.

Table 3 - Properties and composition of the different steel used as styli in the study

type	stylus hardness (HRC)	use	typical analysis (%)						
			C	Si	Mn	Cr	Mo	V	W
Silver Steel	50	dimensionally stable steel used in cutting tools	0.95	0.25	1.1	0.55	-	0.55	0.1
H13	51	hot work tool steel	0.39	1.0	0.4	5.2	1.4	0.9	-
M340	52	plastic mould tool steel	0.54	0.45	0.4	17.3	1.1	0.1	-
CALMAX	52	plastic mould and cold work steel	0.6	0.35	0.8	4.5	0.5	0.2	-
SVERKER 3	52	cold work tool steel	2.05	0.3	0.8	12.7	-	-	1.1
Rigor	52	cold work tool steel	1.0	0.3	0.6	5.3	1.1	0.2	-
S600	55	high speed steel	0.9	0.25	0.3	4.1	5.0	1.8	6.4



Figure 2 - Sample of some of the steel styli used in the study

Rock Sample

Mt White Sandstone sourced from Gosford Quarries Pty Ltd was used as the test rock in the study. It is argillaceous quartz sandstone of the Triassic period having a density of 2.3 t/m^3 and a UCS (dry) of 57 MPa. The silica grains in the sandstone were irregular in shape and varied in size between 0.13 and 0.52 mm. Samples of the rock were cut into cubes using a diamond blade saw that provided a flat, uniform surface for testing. The samples were air-dried prior to testing.

EXPERIMENTAL PROGRAM

The study consisted of two parts using the test apparatus as shown in Figure 3.

- Part A examined the effect of steel type (grade, composition, etc) using seven different metal types at a constant hardness.
- Part B examined the effect of styli hardness at nine different levels with the one steel type, Silver Steel.

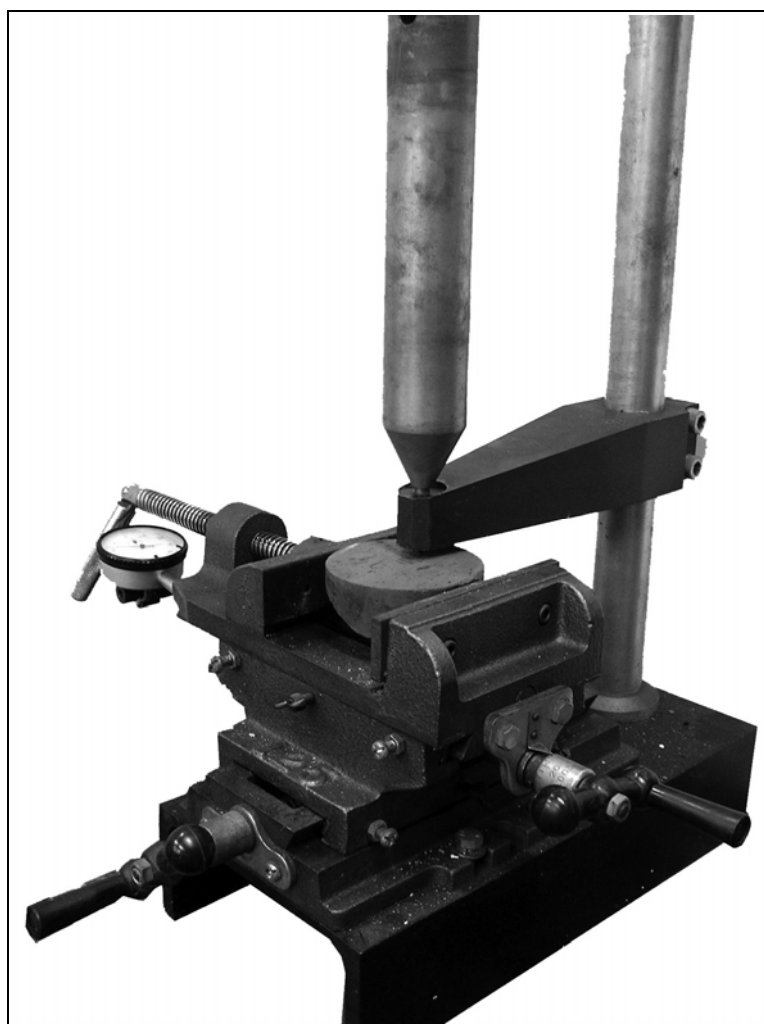


Figure 3 - The Cerchar test apparatus as used in the study

In this way the effect of steel type and hardness were isolated as the testing variables with a total of 16 different steel styli used in the study.

Each test followed the usual Cerchar procedure as discussed earlier and as depicted in Figure 4. In order to ensure a high level of confidence in the test results, the test with each stylus was repeated seven times. The mean CAI value was calculated on the basis of only five replications with the highest and lowest outlier measurements excluded from each calculation.

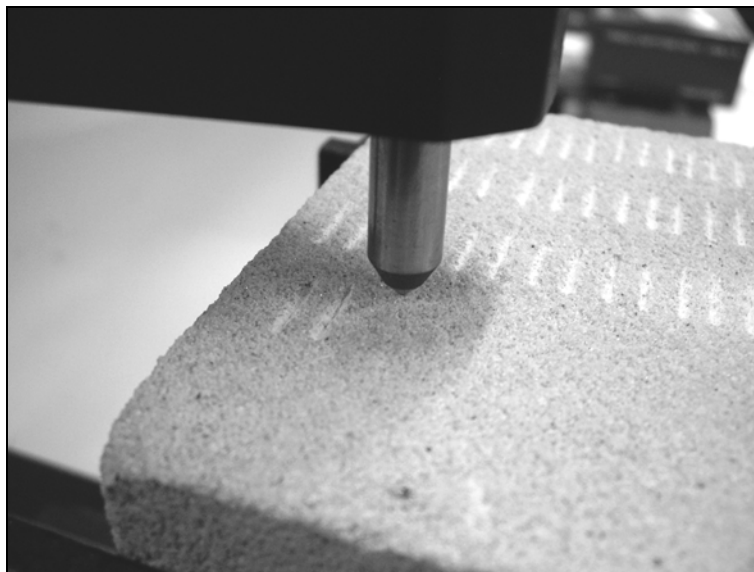


Figure 4 - Test arrangement with pin ready to be scribed across rock surface

RESULTS

Variable Steel Type / Constant Hardness

Results of the test work involving seven different steel types are summarised in Table 4 and the results are graphed in Figure 5

Table 4 - Summary of results of different steel types

steel type	steel hardness (HRC)	CAI		coefficient of variation	deviation from mean CAI
		mean	s.d.		
Silver Steel	50	1.89	0.12	6.2%	-4.2%
H13	51	2.03	0.26	12.7%	+2.7%
M340	52	1.89	0.15	7.9%	-4.0%
CALMAX	52	1.92	0.15	7.8%	-2.7%
SVERKER 3	52	2.08	0.11	5.3%	+5.3%
Rigor	52	2.08	0.11	5.3%	+5.3%
S600	55	1.84	0.23	12.4%	-6.7%
mean		1.97	0.17	8.6%	

Note: Coefficient of Variation (CV) is a normalised measure of the dispersion of a probability distribution and is defined as the ratio of standard deviation to the mean of a sample often expressed as a percentage.

The mean CAI for the seven steel styli was 1.97. The amount of deviation from this CAI value for each individual stylus was comparatively small being at most only 6.7% and in two instances only 2.7%. The smallest CAI was with the S600 stylus which was by far the hardest of the styli tested at HRC 55, however, as the other styli were within ± 1 HRC no other meaningful conclusion can be made concerning steel hardness from this part of the study. The magnitude of these minor levels of deviations becomes significant when cognisance is taken of the heterogeneity of rock and the variability normally exhibited in its properties. For example to reflect the heterogeneity of rock albeit of a different though allied property, Roxborough (1987) reported the variability in compressive strength as measured in terms of the coefficient of variation for sandstone to be 19.8% and for many sedimentary rocks to be slightly higher at 21.7%. In this study the coefficient of variation in CAI was much lower and ranged between 5.3% and 12.7% with an average of 8.6%.

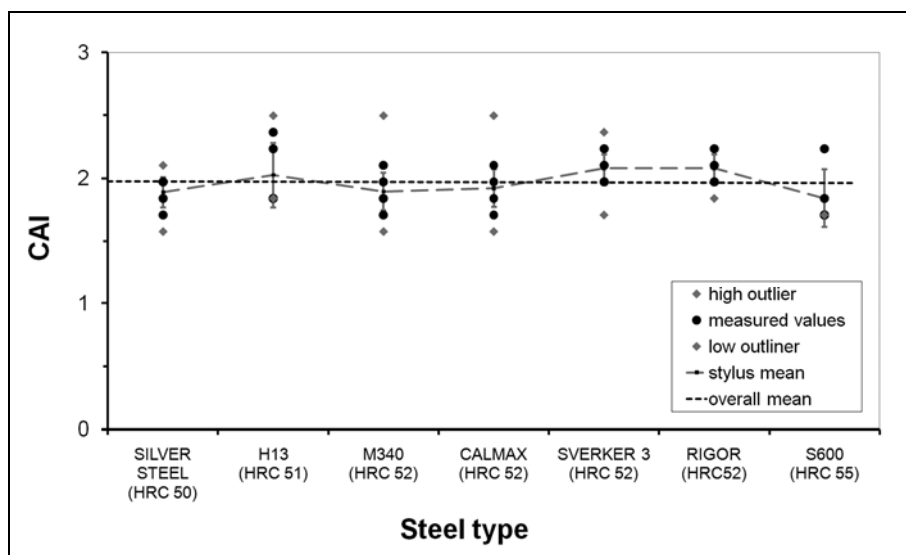


Figure 5 - Effect of steel type on CAI value at constant stylus hardness

Hence CAI does not appear to be significantly affected by changes in steel type of the stylus. It could also be concluded that considering the number of tests that were undertaken with different stylus there does appear to be a reasonable level of repeatability in the test results.

While a constant nominal hardness of HRC 52 was targeted, the actual hardness of the seven different steel styli varied between HRC 50 and 55. This small variation in hardness may contribute in small part to the small variations observed in measured CAI.

Variable Hardness / Constant Steel Type

In terms of the variation in CAI with hardness of the stylus, it was found that CAI decreased with hardness. Moreover considering the range of hardness values investigated it appears that CAI decreases in a linear manner with hardness as is shown in Figure 6. The equation for the line of best fit taking into account all of the readings but excluding the highest and lowest outliers was found to be:

$$\text{CAI} = -0.0766 \text{ HRC} + 5.80$$

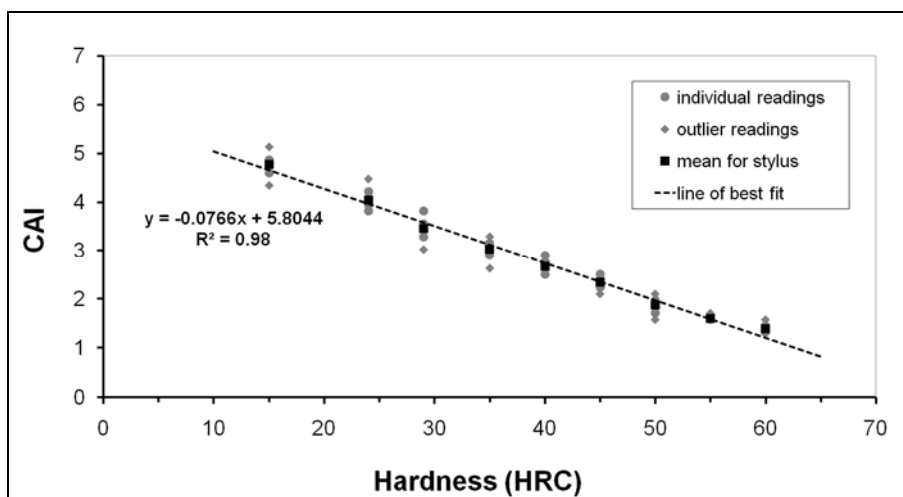


Figure 6 - Effect of steel hardness on CAI with a constant steel type

The correlation co-efficient (R^2) for the data set was 0.98 indicating a good correlation between steel hardness and CAI value. This is also reflected in the small differences between the measured CAI and predicted CAI of at most 0.18 at each level of hardness as listed in Table 5.

Table 5 - Summary of results of changes in steel hardness on CAI

<i>Hardness (HRC)</i>	<i>CAI</i>		<i>coefficient of variation</i>	<i>deviation from trend line</i>
	<i>mean</i>	<i>s.d.</i>		
15	4.77	0.30	5.2%	0.13
24	4.04	0.24	5.9%	0.08
29	3.46	0.29	8.4%	-0.12
35	3.03	0.23	7.5%	-0.09
40	2.67	0.18	6.8%	-0.07
45	2.35	0.16	6.8%	-0.01
50	1.88	0.18	9.7%	-0.10
55	1.60	0.05	3.1%	0.00
60	1.39	0.10	7.4%	0.18

Interestingly, although the variance in the data set for each stylus as measured in absolute terms by the standard deviation tended to decrease with hardness, in relative terms there was little significant change reflected in the coefficient of variation.

ANALYSIS

In testing the effect of metal type it was found that the hypothesis of equal means holds. In other words considering the different steel types tested, No significant effect on Cerchar test results could be attributed to changes in steel type of the stylus.

The highly linear relation observed between CAI values and styli hardness is significant as it allows a simple mathematical model to be determined linking the two variables. In this way, an accurate estimation of CAI as a function of styli hardness may be possible. The significance of this is that it may enable a result to be 'normalised' to a standard stylus hardness for reporting and comparison purposes, for example standardised to either/or specification of HRC 57 (2000 MPa), HRC 40 or some other hardness.

Results of the study suggest that it might be feasible to vary the hardness of the stylus according to the rock being tested. For example, to use a lower hardness stylus when testing softer rocks and a higher hardness stylus when testing harder rocks. This could prove important in several ways. First it could improve the testing accuracy by restricting the length of the wear flat within predefined limits. Secondly, by adjusting stylus hardness the range of rock types over which the CAI test can be usefully applied could be extended. For example softer rock types tend to result in very little if any wear flat when using a very hard stylus.

CONCLUSIONS

The Cerchar test is increasingly being used as a means of assessing the abrasivity of rock samples. There has been some concern expressed about the reliability of the test results especially between different testing laboratories due to the lack of a precise specification of the steel stylus used in the test. The objective of this study was to investigate the impact of changes in some material properties of the steel stylus on the Cerchar test results.

The study found over the range of steel types used as a stylus in the Cerchar test that there was little significant variation in the mean calculated value of Cerchar Abrasiveness Index (CAI). While steel type of the stylus was varied, it was endeavoured to hold hardness constant in the first part of this study at a level somewhere between that specified by CERCHAR (1986) of 2000 MPa and West (1989) of Rockwell Hardness C (HRC) 40. The actual hardness of the seven steel styli varied slightly between HRC 50 and 55 being equivalent to a UTS ranging between 1606 and 1889 MPa. This indicates that selection of stylus for Cerchar testing based on steel type alone is unlikely to have any significant effect of the level of calculated CAI.

In terms of varying the hardness of the stylus, the study found the value of CAI decreased with steel hardness. Over the range of hardness tested from HRC 15 to 60, the value of CAI varied inversely with steel hardness. In all, nine levels of hardness were tested. Consequently hardness of the stylus is

a critical parameter that affects the CAI value for a rock. In light of this it would be prudent when reporting results that hardness of the stylus used in the test work also be reported with the test results.

Based on these findings, the following comments are made with respect to the Cerchar test.

- At least three styli with different hardness levels should be used in a Cerchar test, preferably with as large a difference in hardness levels as is practical. This would allow the construction of a model indicating the variation in CAI with hardness.
- The material properties of the stylus should be reported together with the CAI results.
- A minimum of five (5) replications of the scratch test provides a reliable estimate of the CAI value of a rock sample though it is preferably to undertake seven replications and eliminate the high and low outliers from the calculation of the mean CAI.
- Although it was found that steel type had little or no effect on CAI, it is suggested that the stylus be made from a tool steel or similar composition that is resistant to any heat effects generated during the grinding process of the stylus tip.
- The steel chosen for styli manufacture should be amenable to heat treatment to a wide range of hardness levels.

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