A Safe Way to Reduce Roof Support Costs and Improve Safety and Productivity

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ABSTRACT

The fundamentals concepts of roof support are similar between coal mining, metal mining and tunnelling and yet there are different design approaches, different hardware, and different levels of site investigation. This should not be the case and this paper discusses the key geotechnical issues that should underlie all roof support design - the need to reinforce defects. A design method to build a beam in laminated rock under high horizontal stresses is presented. The method allows for changing rock strengths, defect properties and varying horizontal stresses. At one level the design can be implemented with design charts and tables. Lessons learnt during the implementation of the designs at 5 different mine sites are discussed.

MINING ISN'T DIFFERENT

If mining is thought to be different - then coal mining must be different again! Why is it that underground metalliferous miners cannot believe the amount of roof support that coal miners install ? Why is coal mine roof support hardware different from that sold into metalliferous mines ? What do civil engineers mean when they talk about dowels and bolts ? The aim of this paper is to tackle some of these questions while, at the same time, putting forward the case for a new (and at the same time a very old) approach to specifying roof support in coal mines.

Coal mine engineers use the same theories of soil and rock mechanics as their colleagues in civil engineering - be they foundation, slope stability, or tunnelling engineers. The materials in which they design are similar, the laws of physics are the same and the demands for safe and cost-efficient outcomes are the same. However there is a difference between the way coal mine engineers and civil engineers practise their professions. The differences do not relate to the different geological materials but appear to be steeped in tradition and history. Civil engineers are taught soil mechanics with its strong focus on elastic theory and the use of failure models and factors of safety (limit equilibrium), mining engineers are taught rock mechanics with a focus on empirical methods and back analysis.

Civil structures are capital intensive, and typically of a scale of tens to several hundreds of metres. The designs lack flexibility (restricted to a specific site) and have extremely tight specifications regarding stability, settlement, and serviceability. Even a domestic dwelling which may cost $100,000 to build requires a geotechnical assessment which costs in the order of $500. A major city development can cost $50-$100 million and require $30,000-$450,000 in

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geotechnical design. In the civil arena, the owners/operators are rarely the builder so they rely on project managers, consultants, and contractors.

As a design exercise, a modern longwall mine is no different. It is certainly capital intensive (say greater than $250 million), and has tight specifications - but this time on productivity and rate of return on capital. It does have a greater amount of flexibility in the design which allows it to develop without the areal density (holes/km^2) of site investigation. The areal density of the site investigations can also be less as a result of the homogenisation that rock diagenesis tends to put over the complexity of sediments and soils. In contrast to the civil venture, the mining company is typically the designer, builder, operator and owner. There is no reason why the same relative level of expenditure on geotechnical design should not apply to longwall mines - how many new mining ventures spent $150,000-$1 million specifically on geotechnical issues ? And if they did, did they get good value ?

One of the considerations that appear to be lacking in coal mine geomechanics is the appreciation of the role of defects in the behaviour of rocks and rock masses. Defects are the natural weaknesses in rocks; in coal measures the defects are bedding surfaces, joints, coal cleat, greasy-backs etc. Given the relatively high horizontal stresses that characterise mining excavations in Australia and elsewhere, the most important defects are bedding. For example, mudstones, sandstones, and conglomerates all have a similar range of unconfined compressive strengths but it is known that sandstones give better roof than mudstones, and conglomerates can span longwall panels and delay caving - the difference is in the frequency of bedding defects.

In most cases, rock failure near an underground opening or even on a highwall is expressed as movement on existing defects. Until recently, coal mine geotechnical engineers did not have access to design tools to assess the role of bedding defects in controlling roof reinforcement. The previous design tools based on finite element and finite difference computer codes (such as FLAC) assumed the rock is a continuum (no open defects and the measured rock strengths reduced to account for most of the closed defects) and as a result the focus was on estimating rock strengths from laboratory testing and geophysical logs such as the sonic tool. Rock defects are modelled explicitly in discrete elements codes such as UDEC and these codes allow the defects to open and close. Discrete element codes are extremely numerically intensive and have not yet become commonplace in the industry. Barton (1996) gives a very good example of how UDEC and FLAC differ in the results of analysis of jointed rock masses. If defects are to be included in a design method, the key parameter is their location in the rock mass and their shear strength.

Since 1993 Coffey Partners International Pty Ltd has approached the design of roof reinforcement in bedded strata from an almost traditional civil engineering viewpoint (Seedsman and Logan, 1996). Building on the results of a number of excavations in the Hawkesbury Sandstone (Pells, Poulos & Best, 1991), a design approach based on beam building has been developed. The method recognises the role of bedding defects and seeks to install roof bolts so as to prevent the onset of movements along the defects. In this way the impact of the defects on the rock mass performance is negated. Analytical techniques, instead of numerical techniques, have been used so that the design focus can remain on the critical role of variability of the rock strata - the analytical techniques allow rapid redesigns and sensitivity studies. It is these design tools that have allowed new insights into coal mine roof reinforcement and underwritten the safety, productivity and cost improvements that are being achieved.

BUILDING A BEAM IN COAL MEASURE ROCKS

Introduction

There are specific steps in the application of beam theory to specify rock reinforcement in bedded strata. The general concept is not new but the ability to include the consideration of horizontal stresses has not been readily available in the past.

Field of application

The method is applicable to the building of a structural beam that can span a given opening. It is stressed that in thinly bedded roof there may be failure of scats between the roof bolts - this may mean that a strap or mesh is needed. The method does not specify strap or mesh requirements - this is done through a qualitative assessment of the immediate roof skin.

Note that horizontal stresses in the immediate roof may approach or exceed the compressive strength of the roof beam. In such a circumstance, beam building theory does not have application. The possible onset of failure can be recognised by comparing the measured unconfined compressive strength with the presumed or measured horizontal stress acting across the roadway. The formation of a development roadway can increase the horizontal stress acting in the immediate roof by 20% to 30%. There is an additional concentration of stress on retreat of a longwall face (Mathews, Nemcik, & Gale, 1992). If compressive failure is not indicated, then the design approach is to maintain the pre-failure (ie elastic) behaviour of the roof as long as possible by preventing delamination.

Key steps

There are 4 key steps.

1. In Step 1, the required thickness of the beam to be formed is determined. Well established linear arch or jointed rock beam theory (Brady and Brown, 1983) is used to determine the required thickness - input parameters include rock strength, rock modulus, span, and horizontal stress. Note that the horizontal stress can be the development stress or the abutment stress. Note that for most Australian situations, rock beams of about 0.5m to 0.7m thickness are indicated.

2. In Step 2, the forces that drive the delamination of bedded strata are considered. The assumption is that a bedded unit will delaminate if shear movements are allowed to develop along the bedding defects. Three dimensional elastic theory is used to calculate the shear forces and in particular the excess horizontal shear stresses along any defect after a roof bolt has been installed. Input parameters include the span, friction angle of the defect, and the horizontal stress. It has been shown that the shear stresses increase with height into the roof for about 2m - as a result the shear forces at a height into the roof equivalent to the required beam thickness become the design stresses and the design proceeds based on the assumption that a defect exists at this location (the design defect). The shear stresses are always less than the shear strength of intact rock as such a condition is in fact identified in step 1 and the design does not proceed to this step.

3. In Step 3 the shear resistance of bolts installed across defects is estimated. It is here that the significant difference between bolts and dowels is highlighted. Bolts are tensioned - they must be point anchored prior to tensioning and then they may be column grouted. The maximum shear resistance of a corectly tensioned bolt is the tensile strength of the bar multiplied by the tangent of the friction angle of the defect (for example 17 tonnes for X bar in sandstone). Dowels are untensioned and hence need to be full column grouted. If the defects are closed, dowels can provide the same maximum shear resistance as bolts. However the installation of a dowel cannot ensure that the defects are closed - in stressed, thinly bedded ground the defects can be open before the tendons are installed - in such a case the shear resistance can fall to as low as 4 tonne for the same X bar in sandstone of 20 MPa compressive strength. The difference is that, in the case of the open defect, the shear resistance of the dowel is limited by the bearing failure of the rock just ahead of the bolt.

4. The bolting pattern is optimised if the defects are closed. This can be achieved if the bolts are anchored in strata above the design defect. In step 4, ground anchorage concepts (Littlejohn and Bruce, 1975) are used to determine the required grouting length for the design bolt loads.

Limit state concepts are used to factor in uncertainties for the various input parameters. Typical factors are given in Table 1 but they vary on a site by site basis depending on the confidence in the available data. The equivalent factors of safety are typically in excess of 1.5.

Table 1 - Typical design factors to allow for uncertainties in input parameters

Parameter	Design Factor
Unconfined compressive strength	Typically 0.85
Horizontal stress (modified as in text)	1.05
Shear demand	1.2
Shear resistance	1.1
Anchorage	2.0

IMPLEMENTATION

Implementing change is difficult but perhaps it is even more so in an industry steeped in tradition with a reluctance to change roof support strategies for fear of failure. There is an expectation that there is only one solution to a ground support problem - how often has been heard the complaint "All you geotechnical engineers disagree with each other - which one is right ?". It is possible that in the context that the request for advice is made, each solution is correct ? No one solution will work in all conditions.

There is also the situation that one can go to a mine and suggest the use of 500mm UC steel sets to hold up a 5m wide roadway. There will be no arguments except for the practicality of carrying out such an installation - there is no request for the design calculations. But if creating a 500mm thick beam cheaply out of the roof strata is suggested, there are all kinds of arguments and requests for justifications and design reviews on a subject that most mining engineers have not been trained or educated to understand. And yet the rock beam is in many cases is more stable than steel sets.

The science of rock mechanics and its application of geotechnical engineering has a relatively low appreciation in coal mines compared to metalliferous mining and tunnelling. Working with a number of mining companies aims to manage a change in their roadway support by the application of a significantly different package of design and in-mine services. The discipline that results is challenging for both the support designers and the mining companies but rewards are being seen in reduced costs, improved development rates, and more stable roof.

The first steps in implementation relate to the confirmation of some of the design assumptions. Roof rock strengths are confirmed by the conduct of short-encapsulation pull out tests and a back analysis based on ground anchorage concepts (Littlejohn and Bruce, 1975). An audit of current installation techniques reveals any short-comings in drill rigs and bolting hardware. Load cells are used to determine the level of pre-tension that can be achieved. Geological mapping seeks to define any changes in rock strength or structure that may require a revision of the geological model.

The support design is presented to the company management and the workforce for discussion and risk analysis. Experience is that the face workers are usually more prepared to accept the changes than management.

A trial driveage is then undertaken with geotechnical engineers confirming that the pattern is installed to design and monitoring the performance of the excavation. Poor workmanship will render the design invalid. Monitoring includes the following in addition to the usual extensometry:

1. Bolt installation techniques;
2. Standards of grouting;
3. Correct location of bolts;
4. Correct pre-tension on installation; and
5. Loss of pre-tension.

A back analysis of the results is conducted to check the design assumptions and to identify how the pattern can be further improved

CASE STUDIES

The first application of the design method was conducted in December 1996 at Cumnock Colliery and the support and encouragement of all involved (management, underground workforce, and the Inspectorate) is acknowledged for what was seen by many at the time as a radical departure from standard practice. The objective was to eliminate roof failures in roadway development while maintaining development targets and controlling costs as dictated by the business plan. This was achieved in a trial area of 100m of roadway development by use of short bolts in place of long dowels and at a significantly lower support density. A major contributory factor to improved advance rates was the implementation of single pass drilling with miner mounted rigs where clearance is restricted by low seam height. Further trials are continuing prior to full implementation throughout the pit.

There were two requests from mines to reduce bolting intensity as a way of increasing development rates. In both mines it was demonstrated that a 50% reduction in the amount of steel in the roof was possible by the use of shorter pretensioned bolts in place of dowels. Development rates improved during one of the trials even with the disruption of Coffey engineers to monitor the operation. In the deeper mine it still remains to reduce the pattern to 4 bolts/strap compared to the previous 8 bolts/strap - recommendations on how to achieve this have been made to the client.

At another mine the request was to assist in the redesign of support after a fall of ground associated with a fault zone. The recommendation made was to convert the 6 dowels/m to pre-tensioned bolts and to increase their length by 300mm to achieve better anchorage in the low strength ground. The trial driveage and 3 subsequent drives through the fault zone have been successful - the latter drives indicating that the first fall did not create a stress shadow.

In Mine E a stable coal beam is being formed in 1.2m of top coal. The geology is such that above the coal is a layer of very weak material that washes out during drilling and prevents anchorage of any longer bolts. Above this weak layer is a very strong unit that cannot be drilled with the available drill rigs, and even if it could be, a long-tendon based support system could not deliver the advance rates required.

The salient points of these 5 case studies are summarised in Table 2

Table 2 - Aspects of the five case studies

MINE	PREVIOUS PATTERN	COFFEY PATTERN	REMARKS
A	4 x 2.1m dowels	4 x 1.2m bolts	allowed single pass drilling and hence faster development, reduced consumable costs
B	6 x 2.1m dowels	6 x 2.4m bolts	recovered from fall, longer bolts needed for anchorage in weak ground
C	6 x 2.1m dowels	4 x 1.6m bolts	faster development rate during trial with less steel, crews reported significantly easier work
D	8 x 2.4m dowels	6 x 1.8m bolts	reduced costs, now aiming for a 4 bolt pattern on an ABM20 to get advance rate improvement.
E	new mine	4 x 1.2m bolts	soft puggy band above seam then very hard sill

CONCLUSIONS

By using design approaches more typical of civil engineering rock mechanics so that specific attention is paid to the bedding defects in the rock mass, the efficiency of roof bolting in coal mines can be improved. This can lead to improved advance rates and reduced costs. In the five projects discussed, the benefits are quite evident.

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