Safe and rapid development mining — picking low hanging fruit

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Abstract

Safe and rapid development is fundamentally critical to future production at any mine site. Production rates at all mines are intrinsically related to how effectively the mine can be developed. In the early 1980's electro-hydraulic jumbos were superseded with pneumatic versions with promises of high speed development. Today drifter power has increased from 10–30 kW, however rates of development have not increased. Why?

Development rates are held back by increasing ground support requirements, however, it is often the case that the "efficiency" of these requirements is poorly considered and understood by mine operators. Typical obstacles to high speed development are: cumbersome two pass bolting systems; long waiting times with respect to surface support application especially fibrecrete; poor or non-existent bolt design in development; "blanket"" cable bolting of intersections; the inability of operators to control the size or spans of intersections; and no feed back loops with respect to the support that is installed, i.e. how much load are the bolts "seeing".

1 History

The rate at which a mine is developed parallels the production of the mine. Whilst this may sound as though it is a "motherhood" statement, it is often forgotten or ignored by mine operators. The fundamentals are simple. If an orebody consists of say, 10,000 t per vertical metre, and a development rate of 50 m vertical per year is achieved, then the total production rate of the mine is 500,000 t per annum. If however, a lower development rate is achieved, the mill goes hungry and the business "bleeds".

In the early 1980's, the electro hydraulic jumbo was introduced with great improvements in drill penetration rates compared to pneumatic jumbos. The drifters on the jumbo at that time had an impact power capacity of 10–12 kW. Today, manufactures of drilling equipment are able to supply drifters of 30 kW. Drill penetration rates have increased enormously, however development rates are, in many cases, slower than those that existed many years ago. The reasons for this mainly relate to ground support, both bolting and surface support, and where this support is applied. In the early 1980's, rockbolting was rare, meshing rarer and cable bolting practically non-existent. Today, all drives are bolted and it is extremely rare that a drive does not have a form of surface support and intersections are cable bolted on a regular basis.

The time spent on support certainly provides a safer working environment, however, it is this level of support that undeniably slows down the development cycle. The questions that exist (or should exist) are:

- Does the development drive need the level of support dictated by geotechnical staff?
- Given that a certain level of support is required, what methods are available to expedite the support process?

Without ignoring the importance of the first question, mine operators can certainly make efforts to create efficiencies in the application of varying support techniques as are posed in the second question.

In order to improve the safety and speed of mine development, Mining One has undertaken studies into development efficiency. These studies, sometimes colloquially referred to by Mining One as the "Royal Commission" on decline advance rates, are undertaken to determine the activity and appropriateness of activity at the decline over a set study period (e.g. 72 hours). As a result of the studies, the following issues have been encountered and it is expected that they are common to most development processes.

2 Bolts – installation methodology

The essence of efficient bolting is one pass bolting. Currently available are "drillable" bolts, but at present these seem only to be useful in soft rock/coal "type" applications. This leaves other bolts described as one pass bolts, such as:

- Jumbolt.
- Split Set.
- Swellex.
- Stiff Split Set.
- Resin encapsulated bar bolts.

Without being pedantic, the one that is truly one pass is the Stiff Split Set, hence its favouritism with mine operators in Australia. The other bolts listed require some extra work i.e. inserting grout or resin into drill holes or inserting grout packages into modified Split Sets, or water in to Swellex.

The success of any bolting method with respect to the speed of development lies with the ability by which an operator can reliably install the bolt. It is arguable which of the procedures mentioned is simpler and quicker, however, the "prize" (for want of a better expression) is to achieve the same efficiency of a "normal" Split Set.

A bolt should be simple to install, and relatively inexpensive as a total package. Bolts that require a "return visit" are problematic until the return visit occurs. The author has witnessed jumbo operators "battling" with complicated bolting procedures with a resultant increase in QA/QC verification problems. In many instances designers, whether they are geotechnical or mining engineers, do not seem to understand the complications of bolt systems that they prescribe.

Examples of these problems are the use of 21 mm∅ bolt steel in drill holes that are 32 mm∅, resulting in "gloving" and poor/inefficient connections between the resin/steel/rock.

The author has been involved with senior geotechnical educators that make statements that have no base in fact, or are made with gross exaggeration:

- A Split Set is a "temporary" bolt. This may be the case in a wet/corrosive environment, but in a dry mine, this comment is not a sensible design.
- Rockbolts can never replace cable bolts. The reason given for this was that rockbolts have not got the corrosion protection that cable bolts have and that rockbolts cannot be the same length as a cable bolt. CT bolts i.e.post-grouted point anchor bolts for example, have two layers of corrosion protection beyond cable bolts and three metre rockbolts are the same length as three metre cable bolts.

Sweeping statements such as those highlighted above remain in the minds of students who apply them to the detriment of intelligent or innovative design.

3 Bolt properties

Bolt properties need to be well understood by manufacturers and those who use the bolts in mine design. Mine designers scour web pages and corporate brochures from bolt manufactures, but cannot find information for example, on the adhesion between the bolt and the rock. Understandably, the manufacturers believe this will depend upon the specific methodology employed at a mine, however, an understanding of a range a values of bond strengths even with appropriate caveats is imperative for the designer of the mine. The adhesion of the bolt to rock whether created by mechanical friction (in the case of Split Sets or Swellex), or created by chemical bonding (as is the case with resin bolts, grouted bar bolts or cable bolts) is fundamental to any design.

This is fundamentally one of the reasons why Split Sets are not commonly used in Canadian mines. The common Split Set's size in Canada is 33 mm∅ compared to 45 mm∅ in Australia. The increase in friction is from approximately 2 t/m of embedment for a 33 mm bolt to 4 t/m for a well installed Split Set. This increases to 15 t/m of embedment when the cement grout is placed inside the Split Set. (Fuller and Dugan, 1992; Villaescusa and Wright, 1997) Once this occurs that bolt behaves more like a grouted bar bolt except that the ease of installation is much better.

When bolt properties are clearly understood by the designer, a more effective/efficient bolt system can be employed.

The fundamental information with respect to steel bolts required for design is:

- Ultimate tensile strength (UTS) – maximum, typical, minimum.
- Yield strength – maximum, typical, minimum.
- Friction or bond strength – typical (with appropriate caveats).
- Plate/bolt interface strength.
- Steel elongation.

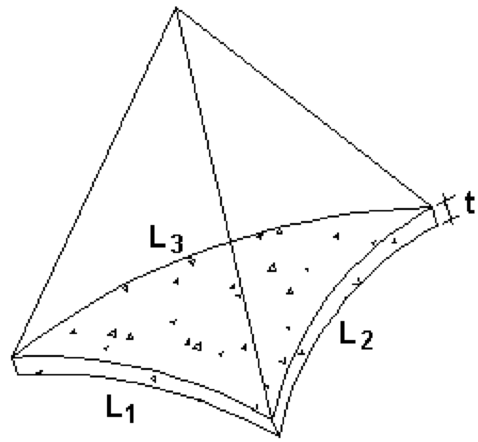
4 Surface support

Surface support, such as mesh and shotcrete, are the most favoured types used in Australia. Shotcrete (fibrecrete) has become popular, however, batching/transport of shotcrete can slow the development process. Opportunities to batch shotcrete efficiently underground do exist. Shotcrete is basically rock (graded down to fine particles (sand) and cement. In most mines rock is abundant underground, but operators insist on batching on surface. Cement handling via silos is difficult but bulka bags provide solutions for underground batching on the proviso that the batch volume is matched to the size of the bulka bag.

Shotcrete strength, including shear strength, is critical to the design and the implementation process. The most common design methods, Rocscience's Unwedge software, uses shear strength in their design algorithm.

Rocscience state:

> *The assumed mode of failure for shotcrete in Unwedge is direct shear. The area of shotcrete the wedge has to shear through is given by the product of the perimeter of the exposed face and the thickness (t) of the shotcrete (see figure below). The shotcrete acts in a direction normal to the excavation wall that it is supporting. For a flat exposed face the total shotcrete capacity is given by:*
>
> $$C = (L_1 + L_2 + L_3)\cdot t\cdot \tau_5$$
>
> *where τ_5 is the shear strength of the shotcrete. For a wedge with a non-flat exposed face (piece-wise linear) the capacities of each segment are added vectorially and the shotcrete acts in the direction of the resultant vector. The resultant shotcrete support force is always included as a passive force in the Unwedge analysis.*



Study in recent times has provided a simple correlation between shear strength and UCS. Rocscience state:

> *To estimate the value of shotcrete shear strength, there are a variety of different codes that relate concrete compressive strength to shear strength. The Canadian CSA simplified standard is shear strength (MPa) =* $0.2\sqrt{f'_c}$ *where f'_c is the 28 day compressive strength in MPa. For example, given 35 MPa unreinforced shotcrete, the shear strength is approximately 1.2 MPa. ACI and Eurocode are similar.*

UCS itself can be readily determined by the use of needle penetrometers and the like. Using the above information, a designer can determine an acceptable factor of safety, taking all of the above into account. The factor of safety should relate to the size of rock that can fall through the area between rockbolts. This typically is 1.5 x 1.5 m. Whilst waiting for shotcrete to core, other work can be undertaken, e.g. bolting and drilling 80% of the development face. Whilst undertaking these tasks, there is no need for the mine operator to progress beyond the last bolted and shotcreted area. The author has encountered negative comments with respect to bolting through wet/green shotcrete. When discussing these comments, it is important to note that even after much discussion, no reasons, based on facts, could be put forward as to why bolting through wet shotcrete could not be undertaken.

5 Intersection design

As previously discussed, inappropriate bolt design is a significant impediment to fast development. With respect to intersection design, it appears that the standard “design” for intersections has become cable bolts. The word “design” is highlighted because, in the author’s experience, in most cases a design has not been undertaken. Mostly, geotechnical engineers nominate the use of cable volts and mine managers blindly follow their advice. This is not to say that in many instances cable bolts are required, however the point to be made is that a design taking into account rock structure, the size of intersection and the appropriate bolt should be undertaken.

Three methods of design are commonly used:

Grimstad and Barton (Grimstad and Barton, 1993)

- charts.
- Parabolic arch estimation.
- Unwedge analysis.

5.1 Grimstad and Barton

These charts give a design methodology based on rock quality, Q, as seen in Figure 1. (Grimstad and Barton, 1993)

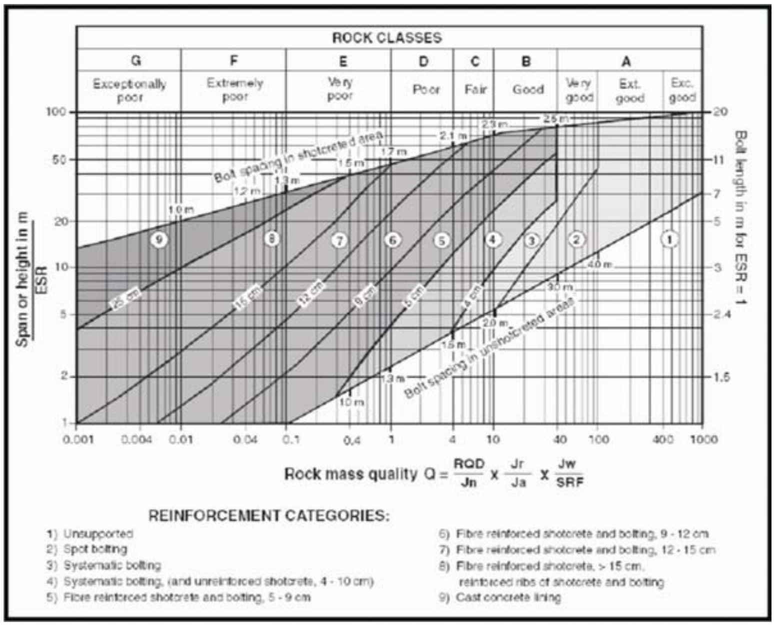


Figure 1 Barton and Grimstad (1993) rock quality versus span

For a permanent mine opening an excavation support ratio of 1.6 is recommended by Barton et al. (1974). Using this figure and a 10 m span, a bolt length of approximately 2.5 m can be determined for any rock mass quality. The use of the charts however, does not differentiate between bolt types and therefore is relatively limited in its application.

5.2 Parabolic arch estimation

This methodology determined the dead weight of a paraboloid of rock based on an inscribed circle in the intersection. The volume (V) of the paraboloid can be determined by:

$V = \pi D^3/24$ (where D is the diameter of the inscribed circle).

This formulae is based on the assumption that the loose rock exists to a height of 1/3 of the span (D) of the intersection.

Given a 10 m span, all bolts must be in excess of 3 m using this method.

The method does not account for rock structure, cohesion, friction angle and possibly clamping stress. For a 10 m span, with a rock density of 3 t/m^3, the following may be concluded:

Volume = 131 m^3

Weight of rock = 131 x 3 = approximately 400 t

Using single strand cables, i.e. 25 t UTS, and a bond strength of 25 t/m, 16 cable bolts would be required. Using a factor of safety of 1.5, this would increase to 24.

All the bolts must penetrate approximately one metre beyond the boundary of the paraboloid, therefore the longest bolt needs to be at least 4.3 m long. In practice, 5 m would be used through the intersection.

This demonstrates the methodology used to determine support requirements and highlights the conservative nature of the approach, i.e. the paraboloid is considered a dead weight with no rock adhesion, no friction between rock and no clamping from depth induced stress.

5.3 Unwedge analysis

Of these, the Unwedge analysis is the more sophisticated and not necessarily that complicated to perform. The key elements to this analysis are:

- Structural mapping.
- Support for intersection “design” size.
- Support for intersection “as built”.

The support selection must be undertaken cognisant of expected life of the intersection, including deleterious effects of water etc. during the expected life of the intersection.

5.3.1 Mapping

Mapping of the intersection should be undertaken prior to and during the excavation. This is a relatively simple process, but unfortunately seemingly strongly resisted by geotechnical engineers. Reasons for this inability to undertake such mapping abound but when it is understood that decline intersections are cut, say once per 50 m, the excuses are quite weak. It is good to note that progressive mine managers facilitate this mapping process via the education of other professional staff such as geologists, mine engineers and on some occasions, surveyors.

Mapping is done with a “Clar” compass (Figure 2) and with experience of undertaking this task, mine staff become extremely aware of the common structures and those that can create large and potentially unstable wedges.

Figure 2 Clar compass

5.3.2 Intersection size and design

The size of the intersection design has the greatest effect on the type of bolt to be used in the intersection. As the size/span of the intersection increases, the greater bolt length is required. In a typical 5 x 5 m development scenario using Unwedge has shown that it is likely that a 3 m “high friction” bolt will be able to support most wedges.

The Unwedge Analysis allows a much more sophisticated examination of potential wedges and appropriate support than the parabolic arch methodology. It can be used as an important design method as all information, e.g. bolt strength, rock cohesion and friction etc., is readily recorded for future reference. The deployment of this software is also extremely useful for the interrogation of support capabilities including bolts and shotcrete.

A design of a typical intersection is shown in Figure 3.

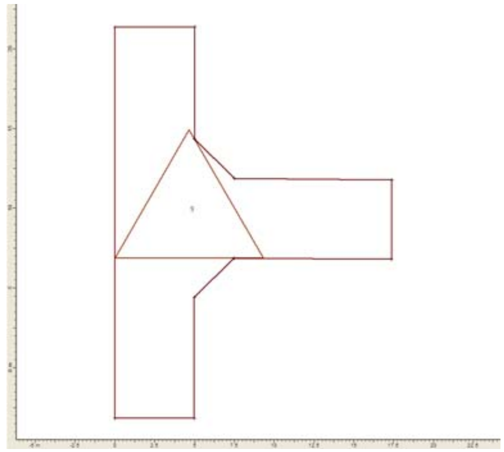


Figure 3 A design of a typical intersection

The drive is 5 m wide and the span of the intersection is approximately 9.5 m. The wedge weighs 91 t. For comparative purposes, a second intersection, the "as built" intersection, is shown in Figure 4.

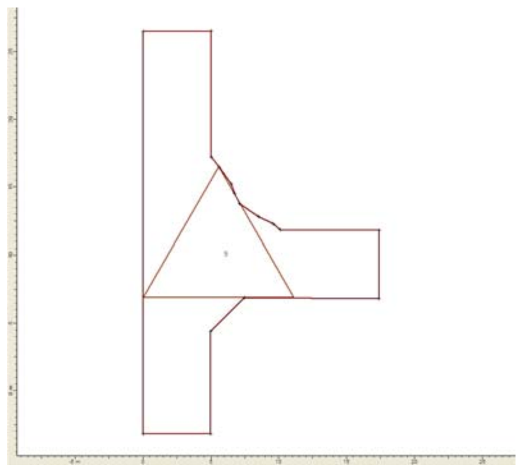


Figure 4 "As built" intersection

In this larger intersection, the wedge has a much larger trace length, in excess of 11 m, and weighs approximately 154 t.

It is these over-excavated intersections that generally create the need for cable bolting and the inherent delays that this entails. These intersections occur because of poor mining practices, and anecdotally, because the mine operators believe that they will have to cable bolt the intersection anyway, so – why not make it as large, as is reasonably practicable.

Interestingly, the geotechnicians are generally of the opinion that the operators will create a large opening no matter what design is provided, so it is best to provide a design that can cope with the over-excavation. This disconnection of thought between mine operators and geotechnical engineers/professionals is endemic in Australia and can be relatively easily resolved by adopting a two part procedure:

- Design the intersection, including support, appropriately.
- Confirm the design and issue an "as built" design ratification.

If the "as built" intersection is larger, then the support will require re-designing most likely resulting in the installation of more bolts, e.g. cable bolts.

If the above procedure is adopted, very soon the mine operators and the mine manager will “feel” the consequences of over excavating the size of intersections.

Using the intersections shown in Figures 1 and 2, designs have been undertaken. In the design a variety of bolts have been used as follows:

- 2.4 m Split Sets.
- 2.4 m grouted Split Sets.
- 3 m Split Sets.
- 3 m grouted Split Sets.
- 6 m single strand “bulb” bolts.

The bolts were placed on 1.5 m x 1.5 m spacing whilst the cables were placed, as is common practice, on a 2 x 2 m spacing. All other criteria remained the same, e.g. cohesion, friction angle, stress induced clamping conditions of zero.

The results, i.e. the factors of safety, are shown in Table 1.

Table 1 Support factor of safety

Bolt Type and Length / Wedge	SS 2.4 m 1.5 x 1.5 m spacing	SS 2.4 m Grouted 1.5 x 1.5 m spacing	SS 3 m 1.5 x 1.5 m spacing	SS 3 m Grouted 1.5 x 1.5 m spacing	6 m single strand cable 2 x 2 m spacing
Normal	1.09	2.84	1.54	3.19	2.40
Oversize	0.72	2.03	1.22	2.33	2.06

SS = Split Set

Table 1 results – factor of safety

It is notable that three long Split Set bolts in a normal intersection provide a factor of safety in excess of 1.5. In larger intersections, grouted friction bolts are to achieve factors of safety in excess of 1.5.

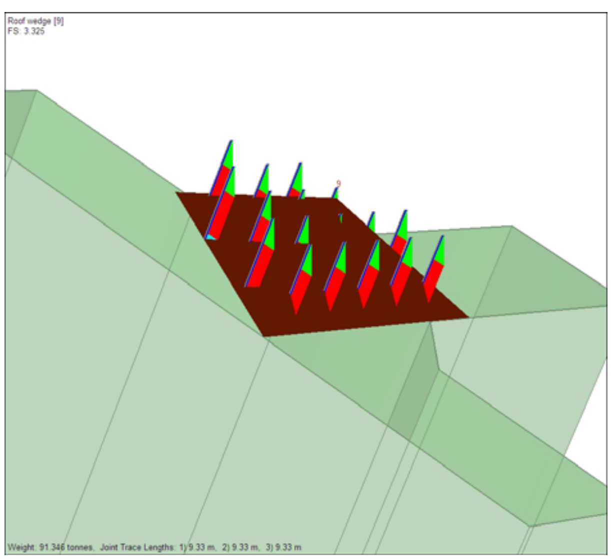


Figure 5 Output from one 2.4 m grouted bolt design in a normal intersection

Figure 5 shows an output from one 2.4 m grouted bolt design in a normal intersection. The lower shading on the bolt shows the propensity for tensile failure whist the upper shading shows the "pullout" potential of the bolt. The high pullout strength of the grouted bolt enables all of the steel strength to be utilised, something that cannot generally occur with non grouted Split Sets, hence their propensity to fail in high demand situations such as intersections. This is shown graphically in the bolt shading in Figure 6 where 2.4 m ungrouted bolts have been used in an oversize intersection. The shading shows pullout of every bolt and no propensity to utilise the tensile capabilities of the bolt.

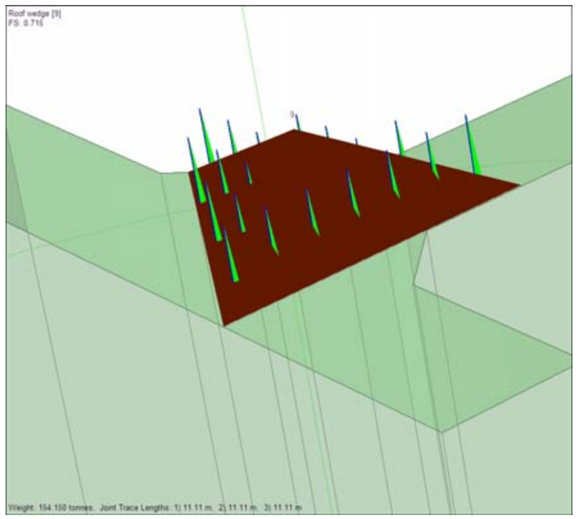


Figure 6 Output from 2.4 m ungrouted bolts used in an oversize intersection

All of these analyses could be re-run using other high friction bolts such as resin bolts or grouted bar bolts. The performance of these bolts should be in excess of that shown for the grouted Split Sets.

2.4 m "high friction" bolts can offer appropriate support in normal sized intersections and 3 m "high friction" bolts can do the same in lager intersections.

Table 2 Probability of failure and factor of safety

Probability of Failure	
1.0 FOS –	50%
1.2 FOS -	10%
1.5 FOS -	1%
2.0 FOS -	0.3%

As a matter of interest Table 2 shows a correlation between factor of safety and probability of failure. A factor of safety equal to, or in excess of, 1.5 resultant in a probability of failure equal to or less than one per cent.

From the above, it is clear that long "high friction" bolts can achieve the same, or on some occasions, better factors of safety than cable bolts. The important part of this examination is that with a rigorous design, the mine manager can not only be presented with a safe and documented design, but one that can be implemented without necessarily resorting to the cumbersome cable bolting and the slowing of the development rate that resultants from cable bolting.

6 Conclusions

Improving the speed of mine development in the most practical and safe manner is not necessarily just about having faster drills, bigger loaders and bigger trucks. It is also about understanding the solutions that have been applied to ground control in the past and ensuring that those solutions are providing value in today’s practice, i.e. optimising those solutions. It is very easy to throw more steel, concrete and time at mining problems. It is more challenging to determine what is the correct amount of steel, concrete and time that a project should take. The perception that a mine manager must ensure that geotechnical aspects are considered in relation to the design, operation and abandonment of a mine does mean that the manager passes all geotechnical decisions to a geotechnical engineer. It also means that the manager should understand the geotechnical environment, get advice and make decisions with respect to that environment. Geotechnical consideration, for a variety of reasons, has created obstacles to safe and speedy development in the past. Practical geotechnical engineers and mine managers can remove these obstacles only when they understand the geotechnical environment, they can question the basis of the decisions and be prepared to be innovative in the manner by which geotechnical considerations are applied.

References

Fuller, P.G. and Dugan, K.J. (1992) Factors affecting ground support performance in mining. Fifth Underground Operator’s Conference, Ballarat, Australia, 14–16 July, Australasian Institute of Mining and Metallurgy, pp. 139–144.

Villaescusa, E. and Wright, J. (1997) Permanent excavation support using cement grouted Split Set bolts. Proceedings of the International Symposium on Rock Support, Lillehammer, Norwegian Society of Chartered Engineers, Norway, pp. 660–670.

Barton, N.R., Lien, R. and Lunde, J. (1974) “Engineering Classification of Rock Masses for the Design of Tunnel Support”, Rock Mechanics, Springer Verlag, Vol. 6(4), pp. 189–239.

Grimstad, E. and Barton, N. (1993) “Updating the Q-System for NTM”. In Kompen, Opsahl and Berg (eds) Proceedings of the International Symposium on Sprayed Concrete – Modern use of wet mix sprayed concrete for underground support. Fagernes, Oslo: Norwegian Concrete Association.

Rocscience…website www.rocscience.com/producuts, Excavation Design, Unwedge, Underground Wedge Stability Analysis