

Selecting an optimal ground support system for rockbursting conditions

V Louchnikov *AMC Consultants Pty Ltd, Australia*

MP Sandy *AMC Consultants Pty Ltd, Australia*

Abstract

A growing number of underground mines experience high seismic activity due to the expansion of the mined out void and the progressive advancement of the operations to greater depths. Seismic events with a local magnitude ranging from 1.5 to 3.0 are routinely recorded at various mines in Western Australia where many such events have resulted in rockbursts damaging underground excavations and posing a high risk of injury to underground workers.

Being the last line of defence in mitigating the risk of a rockburst occurrence, an optimal ground support system must be sufficiently robust to survive the highest magnitude event predicted by seismological analysis, yet still be economically and practicably feasible to ensure sustainability of the mining operation.

This paper presents the process of ground support selection as applied to two Western Australian hard rock mines experiencing rockbursting conditions. The methodology is presented in the format of the capacity meeting the demand. Various commercially available rockbolts and surface liners are reviewed in terms of their energy absorption and deformation capacities. Once all data on the demand and capacity is in hand, the task of selecting an optimal support is then a matter of ranking the systems, shortlisting the better ones for a trial, and finally deciding on the most optimal one or two.

Keywords: *rockburst, dynamic ground support, optimisation*

1 Introduction

Stress often plays a major role in the stability of excavations in underground mines as operations advance to ever greater depths. The depth-related increase in stress magnitude, supplemented by stress concentration at the boundaries of large mined-out voids inevitably results in higher seismic activity and associated rockbursts. Many Australian mines routinely record seismic events which no longer fall into the ‘microseismic’ category, with magnitudes frequently exceeding zero. Large seismic events with a local magnitude ranging from $M_L 1.5$ to $M_L 3.0$ are commonplace at many operations, with the majority of such events resulting in rockbursts, damaging underground excavations and posing high risk of injury to underground personnel.

Understanding of mining-induced seismicity has significantly improved over the past 20 years or so, allowing mine planners to use seismic data in mine design to avoid positioning excavations in seismically active zones. Despite these improvements, rockbursts still occur for various reasons, and mine managers still face these critical questions:

- Are our support systems good enough to guarantee the safety of underground personnel in the event of a rockburst?
- If not, what systems can guarantee that?
- What limitations related to costs, cycle times, logistics, etc. would be imposed on the operation if a routine ‘dynamic support’ system is introduced?

Two underground mines are presented as examples in this study: Mine A – a hard rock gold mine, and Mine B – a hard rock base metals mine, both located in Western Australia. The maximum mining-induced stress modelled at the boundary of development drives on the lowest levels at both mines is in the order of 120 to 150 MPa, which means that all local rock types with unconfined compressive strength (UCS) between 80 and 220 MPa have potential to experience rockbursting ranging from moderate to severe, according to Kaiser et al. (1996).

Both mines routinely record tens to hundreds of seismic events daily. The majority of the events of local magnitude greater than $M_L1.0$ have been interpreted as caused by a fault-slip mechanism. These events may trigger seismic shakedown type of rockbursts, which appear to be the most common type in Western Australian mines.

It is clear that being the last line of defence in mitigating the risk of a rockburst occurrence, an optimal ground support system must be robust enough to survive the largest event predicted by the seismological analysis, yet still be economically and practicably feasible to ensure sustainability of the mining operation.

This paper presents a methodology of selection of a ground support system for dynamic conditions at these two mines.

2 Estimation of energy demand

Dynamic support implies a ground support system capable of surviving dynamic rockburst events where rockbolts with higher load bearing and deformation capacities are used, coupled with an equally strong surface support liner. In order to select a suitable system capable of withstanding a shakedown type of rockburst, three input parameters are required for the given seismic conditions: The design event magnitude, the design event source distance, and the design failure mass.

2.1 Design event

2.1.1 Mine A

The largest seismic event recorded at Mine A was $M_L2.9$. The largest ‘expected’ event, as predicted by the analysis of a five-year seismic dataset, is $M_L3.3$. Despite the fact that the $M_L3.3$ event has a very low probability of annual occurrence (less than 3%), it was selected as the design event, to provide some conservatism.

The most notable damage from the $M_L2.9$ event was observed in a development drive at a distance approximately 50-60 m from the hypocentre of the event, as determined by the seismic monitoring system (Figure 1). A distance of 50 m was selected for design.



Figure 1 Damage in the backs of a development drive from a $M_L2.9$ seismic event at Mine A

2.1.2 Mine B

The largest seismic event experienced at Mine B was $M_L 2.1$. A Frequency-Magnitude analysis indicated that the maximum likely event would have a magnitude between $M_L 2.2$ and $M_L 2.5$ (Figure 2). Due to the relatively large event location error in the deeper parts of the mine where the seismic sensor coverage was incomplete, the $M_L 2.5$ was chosen as the design event magnitude.

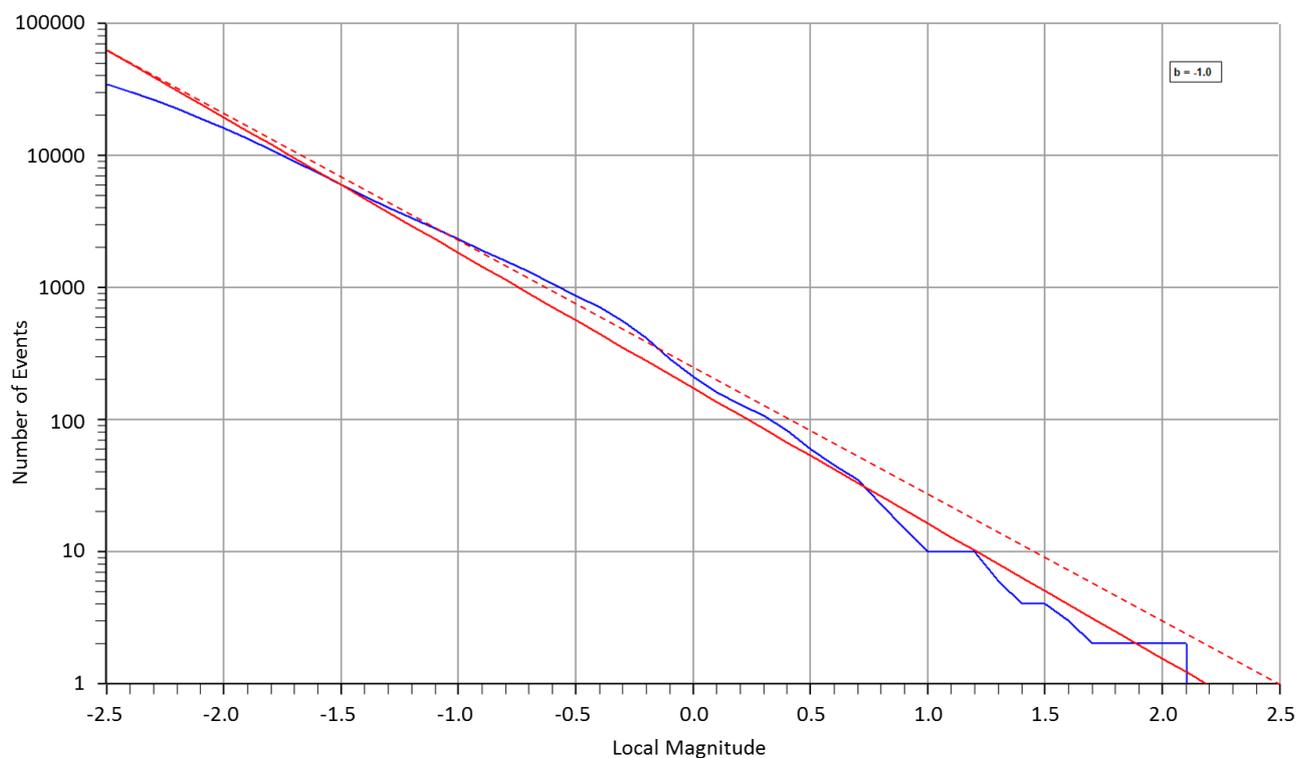


Figure 2 Gutenberg-Richter magnitude-frequency analysis for Mine B

The $M_L 2.1$ event at Mine B did not result in any notable rockbursts, however a separate large event of $M_L 1.5$ resulted in a significant bulking ground failure in the main decline (Figure 3). The calculated distance between the event source and the failure location was 30 to 40 m. A large regional fault is known to be located at a similar distance from the decline. The 30 m distance was selected as a design value.



Figure 3 Ground failure in the main decline of Mine B triggered by a seismic event $M_L 1.5$

2.2 Design mass

The extent of the fractured rock mass, which could potentially fail under a shakedown seismic event, needs to be determined.

The mass of the failed material from a rockburst triggered by the $M_L 2.9$ event at Mine A was estimated at 30 t. A blasthole survey indicated that the depth of stress-induced extensional fracturing in the production drillholes extends up to 1.7 m into the backs, with the most intense damage occurring within the first 0.8 m, as shown in Figure 4(a). This was in agreement with an observation made by the site personnel that most of the rockbolts failed under this rockburst event were sheared at 0.7 to 0.8 m into the hole. Extensional fracturing was also numerically modelled and calibrated against the blasthole survey (Figure 4(b)). The site-specific critical value of extensional fracturing of $400 \mu\epsilon$ can be used to estimate the depth of failure (DOF) indicative of unstable ground.

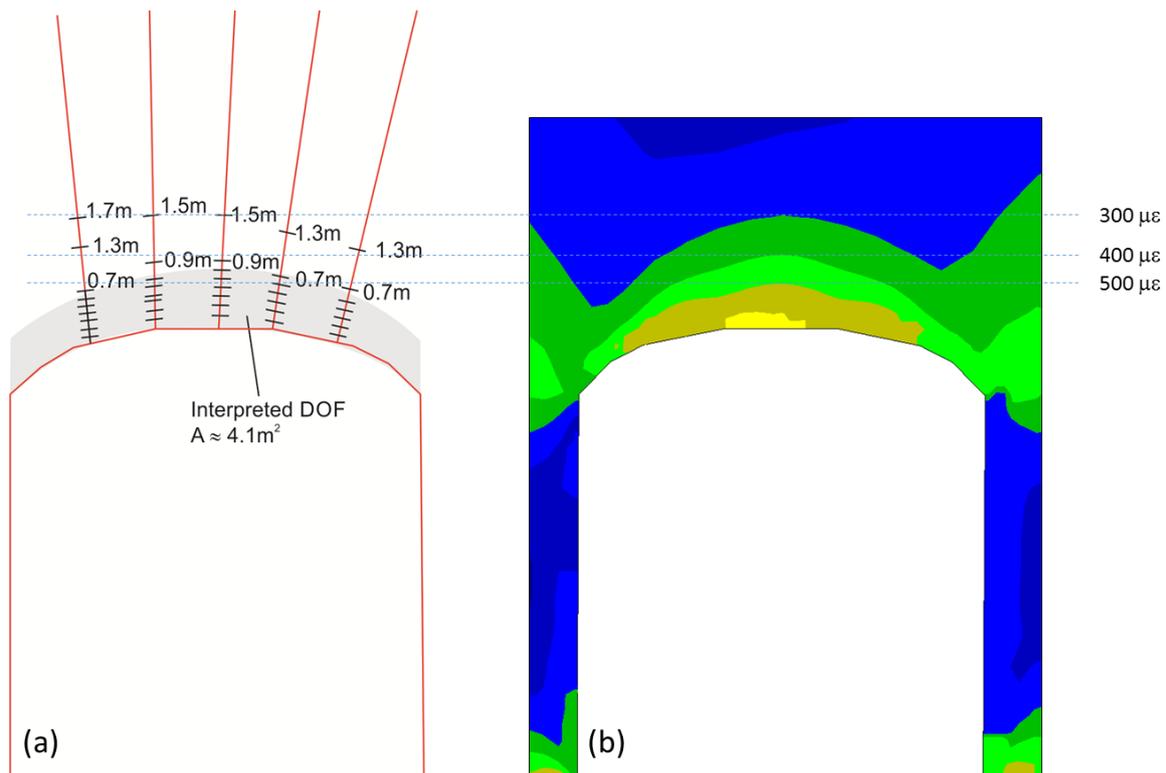


Figure 4 Interpreted DOF in the backs of a drive at Mine A based on: (a) measurement of fracture intensity; and, (b) numerical modelling (extension strain contours shown)

The volume of fractured material in the DOF zone at a 5.5 m drive length (taken as equal to the drive width) and 0.8 m DOF is approximately 25 m^3 , which corresponds to around 70 t of deadweight. This value was adopted as a design rock failure mass at Mine A. The surface area, over which this mass is distributed, is 30 m^2 .

The design mass at Mine B was also estimated from the DOF: the depth of intensive fracturing extended up to 1.0 m as measured in the blastholes and this agreed with the numerical modelling. The volume of the material, which may potentially be shaken down is 30 m^3 , corresponding to a mass of approximately 85 t. The surface area, over which this mass is distributed, is 30 m^2 , similar to Mine A.

2.3 Energy demand

Under static conditions, assuming that the 70 t (Mine B) or 85 t (Mine A) deadweight is completely loose (which of course is a conservative assumption as there is always some degree of arching present in the rock mass whereby some of the load is transferred to the walls of excavation), the ground pressure is around

16 and 28 kN/m² respectively. Under static conditions this pressure can be adequately supported by 3 m long split sets installed on a 1.4 m grid spacing (capacity 80 kN/m²) and weld mesh (capacity 40 kN/m²).

Under dynamic conditions the support system must be designed such that it is capable of absorbing the kinetic energy radiated from a seismic event and the potential energy associated with the bulked rock mass moving into the excavation under gravity. Thus the total energy required to be absorbed by the support system (energy demand) is estimated as:

$$E = \frac{1}{2}m \times (ppv \times SE)^2 + qmgd \quad (1)$$

where:

- m = the design mass (kg).
- ppv = body wave peak particle velocity at the boundary of excavation (m/s).
- SE = site effect (amplification) factor.
- q = a constant equal to 1 for a rockburst from the backs, 0 from the wall and -1 from the floor.
- g = gravitational acceleration (m/s²).
- d = distance the failed material travels (m), typically assumed 0.2 m, which is an allowable displacement of the excavation walls during a rockburst.

The site effect factor can vary widely from site to site and is difficult to quantify. Kaiser et al. (1996) proposed a range between 1 and 4, while in Western Australia this factor is generally accepted as 2 (Potvin et al. 2010). A value of 2 can be suggested for both mines to account for the elevated stress concentration around excavations on the deepest levels.

The ppv can be calculated from the design event source parameters using the following relationship (after Potvin & Wesseloo 2013 who adapted it from Kaiser et al. 1996 to include saturation in the near field):

$$ppv = \frac{C^* \times 10^{0.5(M_L+1.5)}}{R+R_0} \quad (2)$$

$$R_0 = \alpha 10^{\frac{1}{3}(M_L+1.5)} \quad (3)$$

where:

- C* = scaling factor equal to 0.25 for typical mining conditions.
- ML = local event magnitude.
- R = distance from the event source (m).
- R0 = source radius accounting for the attenuation of the seismic wave near-field (m).

Assuming a conservative approach and accepting the lowest value of 0.53 for the constant α (which has a range between 0.53 and 1.14), the ppv at the excavation boundary in the case of a M_L3.3 design event at 50 m distance from the source is 0.9 m/s. This is in line with a seismological analysis undertaken at Mine A, where it was shown that at 50 m the 'expected' ppv from a 3.3 M_L event would be between 0.7 and 1.2 m/s. A ppv of 0.9 m/s is suggested as a design value for Mine A.

At Mine B the M_L2.5 design event at a 30 m distance from the source would result in a ppv of 0.6 m/s.

The total energy demand calculated from Equation (1) is in the order of 250 kJ for Mine A and 230 kJ for Mine B. The surface area, over which the energy is distributed, is equal to the footprint of the loosened rock mass, i.e. 30 m².

From the drop-weight test work undertaken by the AGH University (Bucher et al. 2013) it has been observed that approximately 75% of the energy demand is absorbed by rockbolts and 25% by surface support for simulated 'stiff' rock conditions (i.e. those in a typical hard rock mine). In the AGH test setup, the support

system comprised mesh pinned by rockbolts on a 1.2 × 1.2 m grid pattern. It can be assumed that with a wider rockbolt pattern the energy distribution will be more towards the surface support; however no further studies on this subject have been undertaken to date. In the absence of such data, a proportional relationship can be proposed, e.g. for the 1.4 × 1.4 m rockbolt pattern, which is quite typical in Western Australian mines, the energy split can be assumed as 65% for rockbolts and 35% for surface support.

For weaker rock masses the energy split ratio would be expected to move towards the surface support, with a distribution of 30% on the rockbolts and 70% on the surface support for very poor rock masses (Bucher et al. 2013). Due to the blocky nature of some domains at Mine A, and distinctively foliated ground at Mine B a split of 60/40 (rockbolts/ surface support) is suggested for both mines.

As the results show, the energy demand for both mines is similar, although the design event parameters were different. It can be postulated that the energy demand estimated for these two mines might be applicable to many other mines in Western Australia experiencing or expecting seismic events of magnitude between $M_L2.5$ and $M_L3.3$.

Table 1 Design parameters for dynamic support system selection

Design parameter	Unit	Mine A	Mine B
Design event magnitude	–	3.3	2.5
Distance from event source	m	50	30
Maximum ppv at excavation boundary	m/s	0.9	0.6
Depth of failure (loosened rock mass in backs)	m	0.8	1.0
Design bulked rock mass	t	70	85
Energy demand from design event	kJ	250	230
Energy demand normalised (per unit area)	kJ/m ²	8.3	7.7
Energy distribution ratio (bolts/surface support)	%	60/40	60/40
Energy demand on rockbolts	kJ/m ²	4.9	4.6
Energy demand on surface support	kJ/m ²	3.4	3.1

3 Capacity of ground support

There are a number of commercially available yieldable rockbolts suitable for dynamic conditions; however the weakest link in a ground support system is often the surface support, including the connections with the rockbolts. Therefore the system should be designed based on the surface support capacity matching the demand, along with robust connections to the rockbolts.

Surface rock support serves two main purposes in underground installations: it retains loose material and distributes loads between rockbolts. In static conditions the load transfer from the surface support to the rockbolts is not critical, whereas in a dynamic environment this is the primary requirement for the surface support to ensure that the system works as a whole.

Capacities of various surface support systems are given in Figure 5 and Table 2. All data presented are based on the test results published by the Western Australian School of Mines (Morton et al. 2008; Morton et al. 2009; Player et al. 2008; Balg & Roduner 2013).

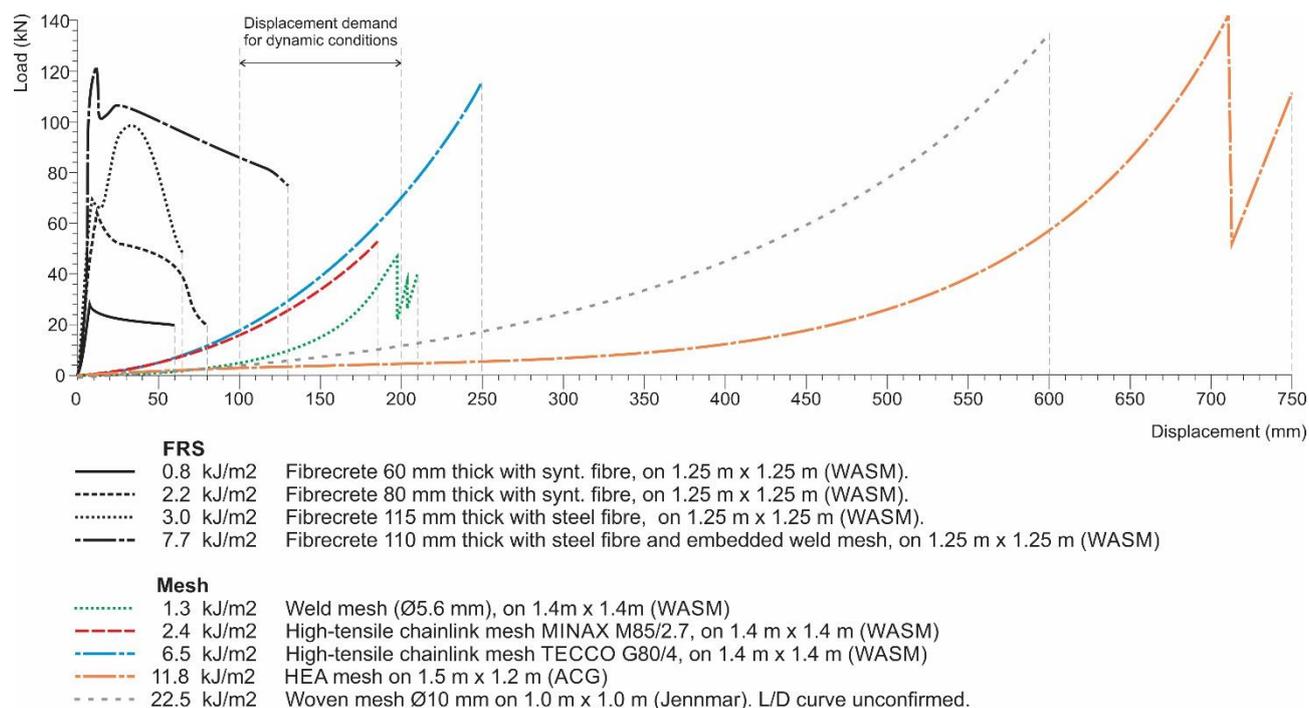


Figure 5 Energy absorption capacities of various types of surface support (from Western Australian School of Mines (WASM) static tests)

Table 2 Energy absorption capacities of various types of surface support (from WASM static tests)

Surface support	Energy absorption per unit area (kJ/m ²)	Maximum displacement at failure (mm)
FRS 60 mm, synthetic fibre	0.8	60
FRS 80 mm, synthetic fibre	2.2	80
FRS 110 mm, steel fibre and weld mesh embedded	7.0	120
Weld mesh 100 × 100 mm (5.6 mm wire)	1.3	210
FRS 60 mm + weld mesh over	2.1	210
FRS 80 mm + weld mesh over	3.5	210
M85/2.7 mesh (Minax high-tensile chain-link)	2.4	200
G80/4 mesh (Tecco high-tensile chain-link)	6.5	300
FRS 60 mm + M85/2.7	3.2	200
FRS 60 mm + G80/4	7.3	300
FRS 80 mm + M85/2.7	4.6	200
FRS 80 mm + G80/4	8.7	300
Woven mesh (6 mm wire) with welded double-wire on perimeter	2.0	300
HEA mesh	11.8	800
Woven mesh (10 mm wire)	22.5	600

It can be seen that fibre reinforced shotcrete (FRS) and weld mesh on their own have capacities lower than required for the design event at both mines. A 110 mm thick application of FRS reinforced with mesh has sufficient energy absorption capacity, however its deformation capacity is limited to about 150 mm (for rock bursting conditions a minimum 200 mm deformation capacity is required). Two high energy absorption types of mesh – HEA mesh and 10 mm diameter woven mesh – have energy and deformation capacities well in excess of the demand. However these mesh systems are too ‘soft’: their high strength is engaged too late, when the ground has displaced considerably beyond the rockbolt deformation capacity, thus compromising the whole system.

An ideal type of surface support is one offering high stiffness at the commencement of wall deformation and then engaging its yielding capability as deformation progresses. The initial stiffness is required to minimise the extent of rock fracturing and hence rock bulking. Such capability is well displayed by FRS under static conditions in moderate to high-stress environments. In dynamic conditions (squeezing or rockburst-prone ground) FRS can neither sufficiently deform nor distribute the load throughout the rockbolt array. Therefore an optimal surface support in these conditions should comprise a combination of FRS and mesh.

Energy absorption capacities of various commercially available rockbolts are presented in Table 3 and Figure 6. Note, this is a guide only as there are many variables affecting the rockbolt behaviour under the dynamic loading.

Table 3 Energy absorption capacities of various commercially available rockbolts determined by dynamic testing (weight drop). Compiled by AMC from various open sources

Rockbolt	Deformation mechanism	Maximum deformation capacity (mm)	Energy absorption at 100 mm slip (kJ)	Energy absorption at 150 mm slip (kJ)
D-Bolt (Ø22 mm)	Bar stretch	220	25	38
Posimix bolt (Ø20 mm), point anchored	Bar stretch	210	23	35
Garock Hybrid Bolt (Ø47 mm)	Slip/bar stretch	200	12	22
Garford Dynamic Bolt (Ø22 mm)	Bar stretch	200	13	20
Cone bolt (Ø17 mm)	Ploughing	300	12	18
Yield-Lok 19 bolt (Ø19 mm)	Ploughing	300	10	14
Swellex bolt (Ø54 mm)	Slip	300	10	14
MD-Bolt (Ø47 mm)	Slip/ bar stretch	300	10	13
Kinloc Bolt (Ø47 mm)	Slip/bar stretch	300	8	10
Frictional stabiliser/split set (Ø47 mm)	Slip	300	4.5	6
Rebar (Ø20 mm), full column grouted	Bar stretch	35	3	–

NB. Deformation of slip and ploughing type of bolts is capped at 300 mm, although they can slip further.

A number of support systems can be selected for given energy demand. It has to be noted that a ground support system for dynamic conditions must be able to absorb the seismic energy with a sufficient Factor of Safety (FS) to be able to survive multiple rockbursts (the main event followed by aftershocks) as well as dealing with the inherent uncertainties associated with seismicity and rockbursts. A minimum FS of 2 is suggested for design purposes.

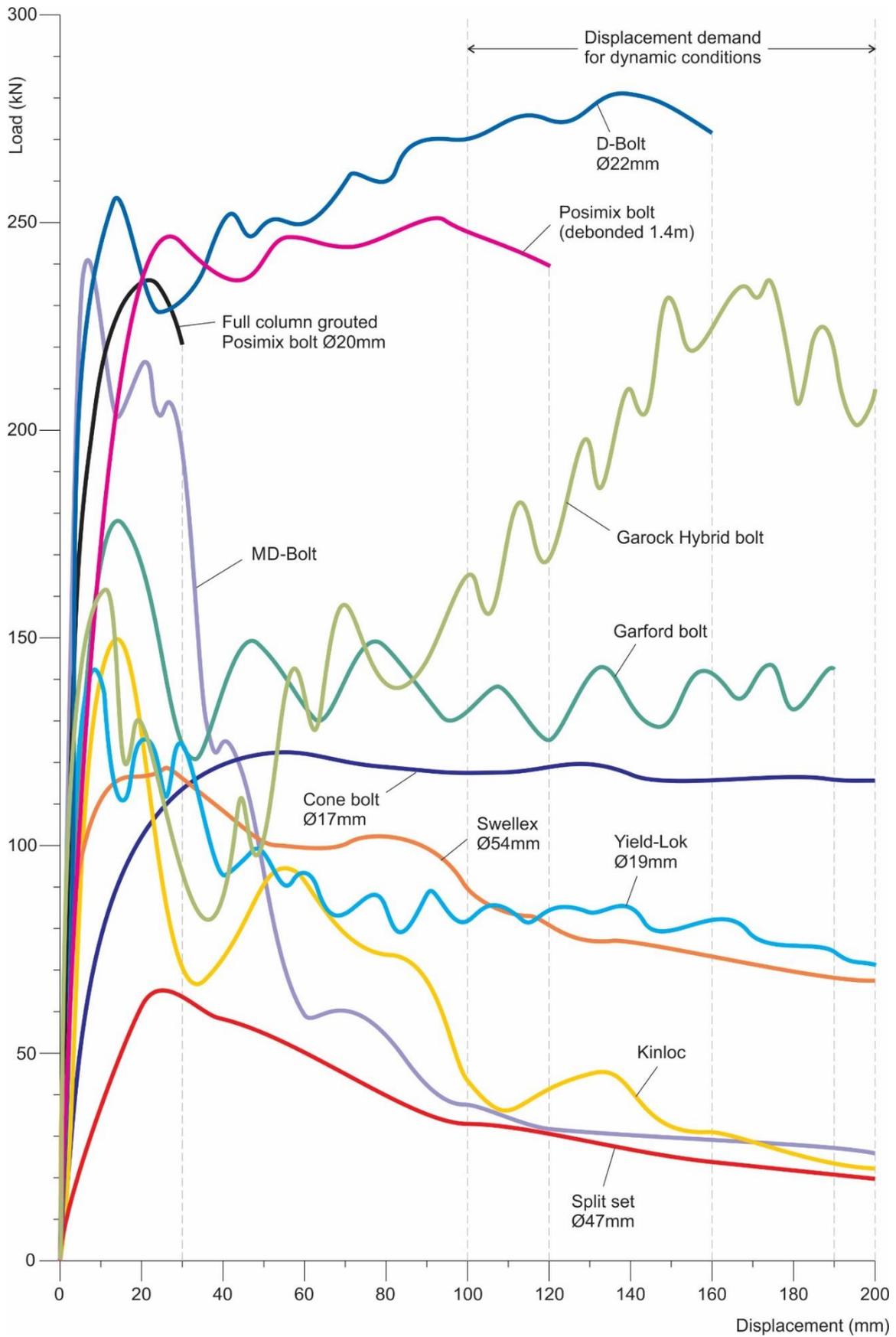


Figure 6 Typical load-displacement characteristics of commercially available rockbolts determined by dynamic testing (weight drop). Compiled from various open sources

Once a number of support systems have been shortlisted, installation trials should be organised. Each system trial should entail the following:

- Duration: minimum four weeks.
- Time and motion study during each development cycle.
- Written feedback obtained from each operator involved in the trial.
- Managed and supervised by geotechnical engineers.
- Documenting the process: notes, photographs, video.
- Estimating costs with jumbo utilisation factored in.
- Reporting on each trial summarising the data collected, feedback from the operators, advantages and disadvantages noted.
- Presenting findings to mine management for making a decision on the preferred system.

It is likely that after completion of the trials there still will be a number of candidate systems. To assist in decision making on the preferred dynamic support option a quantitative rating of the systems can be completed. The overall score will be the sum of a number of rated criteria, such as support capacity, productivity, material handling, sensitivity to quality of installation, and costs.

Ground support systems, shortlisted at Mine A and Mine B for practical evaluation, comprised combinations of point anchored Posimix bolts, MD bolts, a 50 mm thick layer of FRS, 6 mm woven mesh and G80/4 high-tensile chain-link mesh placed over FRS. One of the proposed profiles selected for a trial at both mines is presented in Figure 7.

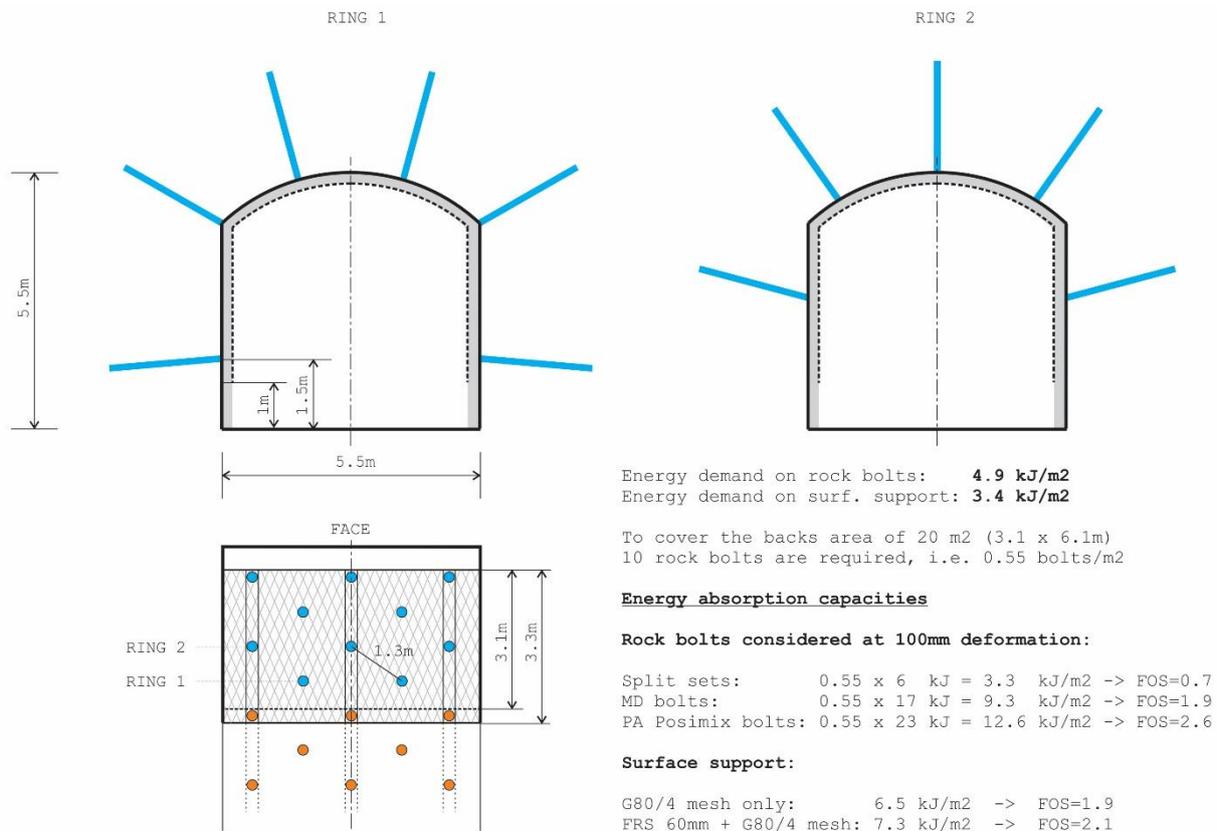


Figure 7 One of the support systems for dynamic conditions shortlisted for a trial at Mine A

4 Concluding remarks

The process of ground support selection for dynamic conditions presented here is based on estimating the energy demand following the well-established methodology popularised by Kaiser et al. (1996). Various commercially available rockbolts and surface liners are reviewed in terms of energy absorption capacity and deformation capacity.

Once all data is in hand, the task of selecting an optimal support system is a matter of ranking the options, shortlisting the better candidates for trials, and finally deciding on the most optimal one or two.

Acknowledgement

The energy absorption values for surface support presented in this paper are largely based on review of the extensive testing of ground support elements carried out by the staff of the Western Australian School of Mines. The authors acknowledge and greatly appreciate access to the WASM work.

References

- Balg, C & Roduner, A 2013, 'Geobruigg AG: ground support applications', *presentation at the Geobruigg's International Ground Support Conference*, Walenstadt, Switzerland, 11–13 September 2013.
- Bucher, R, Cala, M, Zimmermann, A, Balg, C & Roth, A 2013, 'Large scale field tests of high-tensile steel wire mesh in combination with dynamic rock bolts subjected to rockburst loading', in Y Potvin & B Brady (eds), *Proceedings of the Seventh International Symposium on Ground Support in Mining and Underground Construction*, Australian Centre for Geomechanics, Perth, pp. 221–232.
- Kaiser, PK, McCreath, DR & Tannant, DD 1996, *Canadian Rock Burst Support Handbook*, Canadian Mining Industry Research Organization, Sudbury, Ontario.
- Morton, EC, Thompson, AG & Villaescusa, E 2008, 'Static testing of shotcrete and membranes for mining applications', in TR Stacey & D Malan (eds), *Proceedings of the Sixth International Symposium on Ground Support in Mining and Civil Engineering Construction*, Southern African Institute of Mining and Metallurgy, Johannesburg.
- Morton, EC, Villaescusa, E & Thompson, AG 2009, 'Determination of energy absorption capabilities of large scale shotcrete panels', *Proceedings of the Eleventh International Conference on Shotcrete for Underground Support*, Davos, Switzerland.
- Player, JR, Morton, EC, Thompson, AG & Villaescusa, E 2008, 'Static and dynamic testing of steel wire mesh for mining applications of rock surface support', in TR Stacey & D Malan (eds), *Proceedings of the Sixth International Symposium on Ground Support in Mining and Civil Engineering Construction*, Southern African Institute of Mining and Metallurgy, Johannesburg.
- Potvin, Y & Wesseloo, J 2013, 'Towards an understanding of dynamic demand on ground support', *Journal of the Southern African Institute of Mining and Metallurgy*, vol. 113, no. 12, pp. 913–922.
- Potvin, Y, Wesseloo, J & Heal, D 2010, 'An interpretation of ground support capacity submitted to dynamic loading', *Mining Technology*, vol. 119, no. 4, pp. 233–245.

