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Ground support scheme selection and its role in mine economics and risk

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Abstract

Ground support scheme design should be principally driven by the demand from the rock mass of the initial excavation and the response of that excavation with time. The correct evaluation of the demand is the starting point, followed by understanding the performance and sensitivities of the range of potential ground support schemes that may be applied to control displacement.

Technical design, duration, and cost of installation of the ground support scheme are parts of the full development cycle time and advance costs. Focusing strictly on the upfront purchase and installation cost of a ground support scheme creates a narrow financial goal that ignores the larger impact on the overall value of the operation. Ground support scheme capacity should be integrated into the full development cycle and financial model so that the risk of ground support scheme failure and cascading effect on operational failures (ore sterilisation, operational disruption, and safety) are understood.

Evaluation of the total development cost is often opaque as it is generally undertaken by development mining contractors that have a different process of evaluating project risk and consequence compared to a mine owner.

The paper explores the costing of capital development from surface to 2,000 m by considering 5 depths and rock mass demands for multiple ground support schemes. This approach aims to account for differences in evaluating project risk from a contractor perspective of pricing with a focus on equipment and labour utilisation, and short-term cash margin to a mine owner achieving long-term project value, minimising capital expenditure, achieving project payback and resource extraction.

Sites should apply the outcomes in terms of the consequence of the wrong ground support scheme which results in loss of the decline/capital and access to the ore zone. This situation is not rare with the author knowing of multiple sites that have been faced with this outcome.

Keywords: *ground support scheme, demand, capacity, development cost, mine risk*

1 Introduction

The financial criteria used to evaluate underground mines have assumptions with regards to risk. Often, the engineers that undertake the ground support design do not have an understanding of the build-up of the full mining costs (and how little the ground support scheme elements are) and the design engineers do not appreciate the geotechnical risks that are adopted with sub-optimising ground support scheme elements based on cost alone.

Not all development is equal. In a modern Australian mine, the requirement for maintaining productive decline access has resulted in meshing of drive profiles to grade, even in excellent rock mass, so no time is 'lost' on check-scaling. The high heading surface support code (MOSHAB 1999) requirements has contributed to a reduction in fall of ground fatalities in Australian metalliferous mines (Lang & Stubley 2004; Owen 2009) but has also meant that we have a generation of engineers and operators that know how to pattern bolt and

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do a classification scheme response, but are not able to evaluate the rock mass demand and 'read the ground'.

This has resulted in an outcome where rock mass that needs minimal or no reinforcement has the same ground support scheme installed as that which is weak and deforming. From a human perspective, training our workforce to understand that there is a ground support scheme installed that does 'nothing' is difficult. In difficult ground conditions, the uncertainty of excavation and rock mass response often leads to a situation of assessing the rock mass, monitoring the response and adjusting the ground support scheme design, commonly termed the 'observational method' (Stacey 2003, 2009; Dunn & Parrott 2023).

Whilst this is a common and typically effective practice, when capital infrastructure is impacted by significant creep or rockbursting/strainbursting (and kilometres of decline/capital needs either ground support scheme upgrades or replacement), there are significant and sometimes terminal consequences for the mine. This is particularly the case (and challenging) when the rockbursting occurs significantly after development due to mine stress changes rather than the stress changes associated with the initial excavation.

It is proposed that the solution is to default to high capacity, high toughness, and efficient to install ground support schemes for deep mine capital development. The objectives are orebody access and production security, as well as operator safety.

The challenge then becomes to have the knowledge for when a rock mass has the potential for strainbursting/rockbursting to occur and that mine design/scheduling does not make this situation worse. Part of the solution is to improve geotechnical education in metal properties, statistics, dynamic rock mass failure, load transfer and bonding/anchorage for reinforcement and support systems. The rest of the solution is two-fold. First, understand the real cost of development (not just the components and the small percentage that the ground support cycle – and particularly the elements selected – contribute to the total development cost). Second, ensure that operator engagement, training and compensation is paramount as they develop the necessary skills for modern ground support schemes.

All costs and productivity data in this paper are developed using 2026 Australian Dollars, activity rates, and an Australian mining development contractor using new equipment and a fly-in fly-out workforce. Dollar figures are only shown to 4 significant figures.

1.1 Terminology and definitions

The consistent use of terminology and definitions is critical for technical discussions. Refer to Chapter 2 of Villaescusa et al. (2023) as well as Thompson et al. (2012, 1992) for a full discussion. They are summarised in this paper.

1.1.1 Demand

This is the total energy or displacement applied from the rock mass onto the excavation due to a change in induced stresses, static conditions, and dynamic loading.

1.1.2 Capacity

This is the physical ability of the ground support scheme in response to the demand from the rock mass and excavation. It can be evaluated in terms of force (kN), displacement (mm), pressure (kPa) and energy (kJ/m^2).

1.1.3 Reinforcement and support

Reinforcement is embedded within the rock mass surrounding an excavation and is fixed in place by friction, mechanical anchorage, or bonding agent.

Support is placed on to the rock surface and responds to deformation of the rock mass (it can function without reinforcement due to bonding to the rock, i.e. fibrecrete, or works only with reinforcement systems for attachment points, i.e. mesh).

1.1.4 Systems and schemes

A system is the elements added together, i.e. a resin encapsulated reinforcement system considers the element of the reinforcing bar (its grade and yield properties), how the encapsulation medium is mixed to anchor to the drillhole, the bond from the encapsulation medium to the bar, and the surface fixture connecting the bar to the excavation.

A scheme is the combination of the reinforcement system and support systems to control excavation deformation to the required amount installed according to the mining plan and sequencing.

Characterise the capacity of the systems and schemes by:

- Load: maximum load value and residual load value at maximum displacement.
- Displacement: deformation at maximum load value and maximum displacement.
- Stiffness: slope of the load displacement curve.
- Energy dissipation: area under the load displacement curve.

1.1.5 Rockburst/strainburst

This is the dynamic failure of the rock mass with bulking/failure into an excavation. He et al. (2025) has identified 80 published definitions in the literature from 1946 to 2024, and this speaks to the complexity of the problem. When there is no easy agreement on a definition of the problem, it is not surprising that there is limited agreement on solutions.

Strainburst is a localised sudden ejection of rock into the excavation due to stress conditions from excavation of the heading. It is massive brittle rock – particularly if an intrusive in a host rock mass – failing due to the stress path/strain change from the loss of confinement. The source mechanism is crush when using a Hudson chart.

Rockburst is a large seismic event with significant moment/energy radiated that causes failure of the rock mass about an excavation. Source mechanisms can be crush, slip or combination/complex. The stress/strain change is related to other mine void changes in the mine or a long-term deterioration of the rock mass so that dynamic failure occurs releasing a stored strain energy.

For the 2 cases, a crush source mechanism means the seismic source incorporates the excavation.

1.1.6 Factor of Safety

The Factor of Safety (FoS) is the ratio of capacity divided by the demand. If less than 1, the system is unstable and if greater than 1, the support system is stable. A FoS greater than 1 is required to account for uncertainties due to geological variance in the rock mass, demand, and installation quality of the ground support scheme.

It can be evaluated in terms of loading mass and resistive force for wedges, for excavation displacement and scheme capability to displace for creep, and a combination of energy and displacement for dynamic loading.

1.1.7 Design acceptance criterion

Design acceptance criterion is the standard or capacity that is demand acceptable to manage risk or performance of a system, e.g. the work by Dunn & Graaf (2023).

1.1.8 Abbreviations and definitions

A list of abbreviations and definitions relevant to this paper are included in Table 1.

Table 1 Abbreviations and definition

Terms	Text
Resin	Resin encapsulation medium used on the threaded bar. The speed combinations used are not significant for this topic. Can be either cartridges or pumped
Hybrid	A solid bar with an end anchoring mechanism inside of a split tube friction bolt. It relies on increase in end friction from the anchor and is not a mechanical lock onto the rock mass
Gewie	A steel threaded bar either 20 or 24 mm nominal diameter with a thread. Thread pitch, width and height can vary. Bar steel grade can vary with significant change in load transfer response. 2.4 or 3.0 m in length. 20 mm diameter and 500 MPa yield are the base in this study
Debonded	This is where a sheath is over part of the threaded bar to stop the encapsulation medium bonding to the steel. It should only be the central portion of the bolt and not the collar
CMC	Continuous mechanical coupled
CFC	Continuous friction coupled
DMFC	Discrete mechanical friction coupled
GSS	Ground support scheme/s
SS	Split tube friction bolt that has an oversize C-section driven into a smaller diameter hole
FC	Fibrecrete (sprayed concrete with plastic fibres added to have Round Determinate Panel (RDP) capacity of 300–550 J). Typical thickness in mining application of 50–100 mm
M	Weld mesh 100 × 100 × 5.6 mm wires. Wire >500 MPa yield, weld strength >75% of the wire yield
Minax	Woven wire mesh >1,770 MPa yield stress, G80/4.6 product from Geobrugg

2 Demand/capacity/response

The selection of appropriate ground support scheme (GSS) should start with the evaluation of the rock mass demand. The demand upon the excavation is driven by multiple factors (rock strength and anisotropy, rock modulus, fracture toughness, structure and joint conditions, structural setting, stress and the actual excavation), each of which has many years (often decades) of research undertaken in developing empirical relationships and numerical models.

The capacity of the installed GSS is typically evaluated only in terms of reinforcement system resistive force (collar pull tests of installed rockbolt) and occasionally the displacement is measured. The capacity of reinforcement systems in laboratory testing is quantitatively evaluated with double embedment configuration to obtain the load–transfer relationships (Hutchins et al. 1990; Villaescusa et al. 2013a). The double embedment tests are generally axial and when they do evaluate shear, the most significant aspect which controls reinforcement system performance, dilation into the excavation across the joint or stress induced fracture is unknown. However, it is dilation that will result in reinforcement system survival or guillotining.

Support systems are typically evaluated from element or system tests (e.g. RDP, EFNARC, and the WASM large area panel tests [Player et al. 2008; Morton 2009; Villaescusa et al. 2013b; Roberts et al. 2018]).

Hence, we should not be surprised that the GSS capacity is forecast by numerical models (Lorig & Varona 2013; Ghazvinian et al. 2023; Dehkhoda et al. 2023). The actual capacity of a full scheme is typically only evaluated by installed monitored performance. The challenge in this situation is determining the demand. Whilst the demand can be evaluated post-event by gravity-driven failures, the actual demand from rockburst upon the installed GSS is significantly more challenging. As such, brackets of rock mass demand are related to reground ground support scheme requirements (Villaescusa et al. 2023, Table 5-1).

Figure 1 – whilst comprehensive in summarising the inputs to ground support design – is not complete. It does highlight the complexity of the situation on what is trying to be achieved. What we are trying to achieve is effective ground support schemes (Figure 2; in this case shown for dynamic environment).

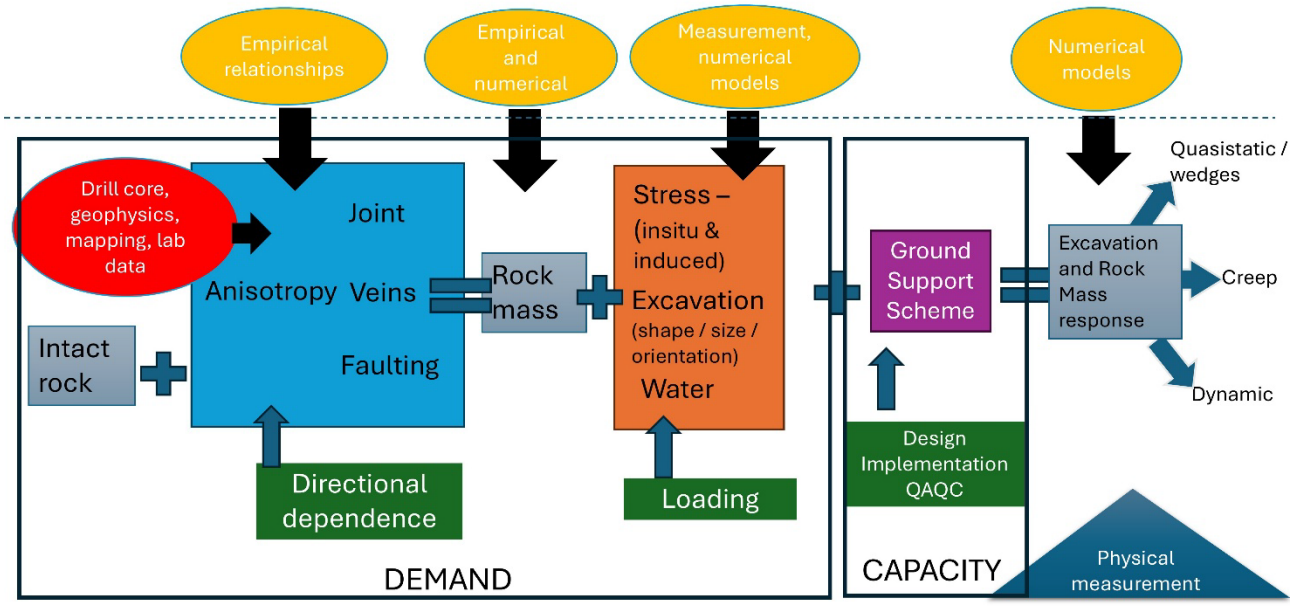


Figure 1 Inputs to ground support scheme design

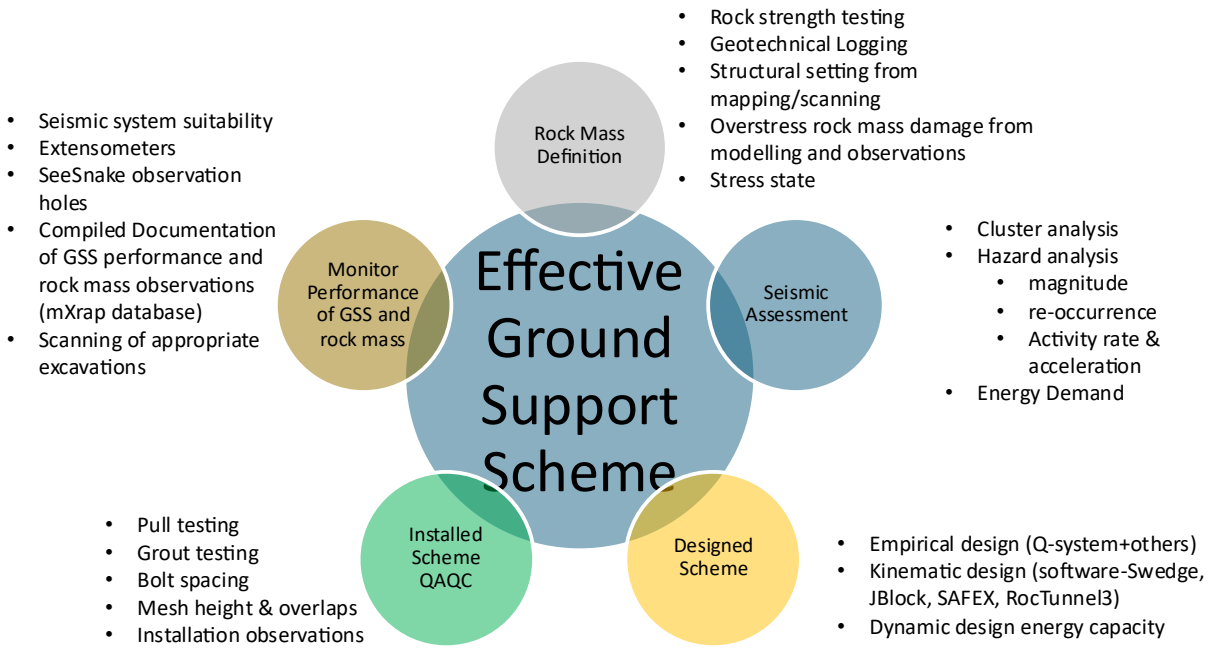


Figure 2 The inputs to an effective ground support scheme for a dynamic environment

The final part of the design of the GSS and its response to the environment is the design acceptance criterion.

A clear statement of design acceptance criterion by company boards (that matches their company risk profile) against the value of lost access to deposits would enhance how the mine design and GSS are undertaken during studies, and by the site engineers rather than the design drawing in individual experience on a level of appropriateness.

A common outcome at operations that have catastrophic capital development events is that such a rock mass response was not expected or forecasted. Hence the applied mine design concentrated the stress (made the situation worse) or the GSS did not have the required capacity or coverage to control the damage.

This should then provide a pathway for company boards to understand that the economics and risks for protection of the asset, by the use of apparent higher capacity schemes than forecast at design stage for dynamic rock mass response as mines and organisation, underestimate how damaging the seismic events will be. This is due to the uncertainty in relationship between size of seismic and damage to excavations and when rock mass response changes from controllable yield and minor to moderate seismicity to major damaging events.

3 Our study

The study considers 5 zones in a mine with one decline and capital development that is 5.5 (W) × 5.8 m (H) with a full arch profile (larger and flatter profile declines would be expected to require additional GSS capacity) and determines the productivity of a jumbo and the cost per heading by mining practices. The 5 zones are defined in Table 2. The assessed GSS are defined in Table 3 with figures of the GSS in Appendix A. The GSS are selected based on current industry practices but also what we consider to be appropriate. Mines should be able to compare their current scheme to those listed and understand the similarities and differences with a knowledge of how force–displacement relationship work and the dependence on bond strength from encapsulation medium to the bar. A knowledge of all costing inputs and cycle time are required to generate the costing models.

Brand names for systems within the schemes are not used unless necessary due to no other equivalent products being available.

The key aspects of the certainty of demand and what makes the GSS succeed or fail are discussed for each of the zones. The zone elevations are typical to provide a basis for costing but can vary from mine site to mine site depending on the local rock mass, stress state and mine design/sequence. The variation in demand driven by rock mass quality and stress is not a new concept and well expressed by Figure 1-1 in Kaiser et al. (2000).

Table 2 Study zones

Zone	Depth from	Depth to	RMR89	Demand	Weathering	Cut advance	Ground condition factor [1]	Drill and blast factor [1]
1	20	100	30–50	Medium–high	Weathered	3 m	Poor	Fair
2	100	400	+75	Low	Fresh	4 m	Fair	Fair
3	500	900	+75	Medium	Fresh	4 m	Fair	Fair
4	900	1,600	+75	High	Fresh	4 m	Poor	Fair
5	1,200	2000	+75	Very high	Fresh	4 m	Poor	Fair

1 - Ground condition factor and drill and blast factor from Figure 1 (Wood 1999)

Table 3 Ground support schemes

Zone	Pattern	Description	Wall coverage	Seismicity	Encapsulation
1	GS1	FC + Gewie + SS + M + FC	Floor	Aseismic	Cartridge resin
2	GS2.0	SS + M	3.5 m	Aseismic	Nil
2	GS2.1	FC + Hybrid	Grade	Aseismic	Nil
3	GS3.0	FC + Hybrid + M	Grade	Medium	Nil
3	GS3.1	FC + Gewie + M	Grade	Medium	Cartridge Resin
4	GS4.0	FC + Debonded Gewie + Minax + FC	Floor	High	Pumped Resin
4	GS4.1	FC + Debonded Gewie + Minax + FC	Floor	High	Cartridge Resin
4	GS4.2	FC Debonded Gewie + M + FC + Osro	Floor	High	Cartridge
4	GS3.0D	FC + Hybrid + M	Grade	High	Nil
5	GS4.x	GS4.x + 10 cables	Floor	Very high	Lagging and installed with cement grout

In Table 3, the second layer of FC is a partial layer to prevent equipment damage to mesh. This second layer of FC should only be 50 mm as a thicker layer does not allow the mesh below to still function as a mesh.

3.1 Zone 1

The rock mass is still within the regolith profile with low (and variable) Rock Quality Designation, poor joint condition, very weak to weak intact rock strength, and additional major structures likely within the weathering profile.

High variability and rapid change within the weathering profile is not unusual, including changes from weathered to fresh and back to weathered again associated with major structure and lithological changes. It is critical to maintain control of the excavation profile and FC is the best option to achieve this. The resistive force from the FC from very small displacement aids weak rock mass to stabilise rather than deform and unravel (Mbenza et al. 2024; Bernand & Thomas 2020) and integrating the weld mesh provides additional deformation capacity and toughness for the support system.

Within the weathered zone, the following actions are undertaken depending on the variance of rock mass conditions (these variations are not included in the cost model):

- Difficult to maintain drive profile
  - Blasting practices with uncharged holes or uncharged final row/under boring heading size and scale to dimension
  - Use mechanical advance (rattling, or mini excavator).
- Round length is shortened to 2 m advance with mesh being run across rather than along the drive profile. Mesh across the drive is more resistive to being pull apart then mesh along the drive.
- Spray quickly: clean up fired rock sufficiently so that FC is applied to stop rock mass unravelling and then load out the rest of the fired rock.
- First layer thickness of FC is increased to an upper limit of 100–125 mm (limit for the extra thickness can be sprayed without falling off before curing).

- FC arches with rebar or Osro strap reinforcing are added to improve toughness and capacity.
- The type or quantity of fibres maybe changed (if equipment can manage the additional fibre or longer fibre).

Due to rapid changes in the rock mass conditions (higher variability that likely to be described during study due to drillhole spacing), having a GSS that is adaptable (increase FC thickness, add arches) and can response to the changes (every cut inspected is critical). As the ground conditions improve the one layer the outer layer of FC is removed, and then lower mesh raised.

This is in the upper portion of the mine and it will be there for the longest period of time. The uncertainty in ground conditions in weathered rock mass means engineers should be not striving for exact compliance to the design but rather focus on controlling the ground from deforming/unravelling with the appropriate round advance and quick installation of the FC shell.

For this case study, the depth is shallow and stress-driven creep is unlikely with the appropriate GSS used.

3.2 Zone 2

The rock mass is fresh and competent with demand driven by the structural setting. Rock mass classification such as Q-system (NGI 2025) is not designed for targeting discrete wedges for kinematic instability. For many operations in steep dipping shear hosted orebody, the structural setting is 2 steep sets and one shallow set and any wedges that do form are shallow in depth and extent and hence, the use of friction stabilisers and weld mesh provides a function fast installation that removes hazards associated with small blocks. Reardon et al. (2022) and Varden et al. (2023) provide comprehensive discussions on the challenges with use of split tube friction stabilisers.

When there are clear moderately dipping sets in the structural data that forms a clear wedge in the backs of the development then friction stabilisers (CFC) are typically replaced by end anchored (DMFC) or fully encapsulated (CMC) reinforcement systems.

For wedges in shallow civil applications, Hoek (2006, Chapter 2, Table 3) recommends a FoS greater than 1.5 for slide and greater than 2 for falling wedges and blocks.

Li (2017) recommends; wedge in the backs FoS 1.5–3, wedge in the wall FoS 1.5–2. For both Hoek (2006) and Li (2017), it is critical to be able to define the structural setting, how the wedge is to be exposed, its size, and the timing of the reinforcement of the wedge. The difficulty in displacement monitoring of free falling wedges is also noted. This uncertainty in being able to define the wedges is a contributing factor to the higher FoS. Hudson & Harrison (1997, Chapter 19) provides one of the better examples of how to assess wedges for free falling, sliding and stable blocks on the backs and walls of excavations. Wedge can also be ‘stable’ by having small amount of cohesion or clamping stress; however, relying on these to maintain stability is not recommended particularly due to stress changes and ground deformation that occurs in mines due to orebody extraction.

However, in rock masses where the structural setting is less clear, with moderately dipping sets that are less dense and more disperse, the implication becomes that 3 and more likely 4 release plane defined wedges do form on an occasional basis. The frequency of occurrence is determined by the geological and structural setting as exposed by the mine excavations, but the knowledge of the occurrence is dependent on the amount of data collection and appropriate interpretation.

Additionally, when analysis that only tests 3 release plane wedges is undertaken, then the weight of the wedge is likely to be incorrect. The basic software options for the assessment of 3 release plans are no longer appropriate and engineers will need to use JBlock (Esterhuizen 2026), RocTunnel3 (Rocscience 2026), or Safex (Villaescusa et al. 2018).

In these situations, the mine either needs a strict process for defining occurrence or to replace the CFC with a reinforcement system with one that is more suitable (CMC or DMFC).

An alternative strategy to GS2.0 SS + M is GS2.1 FC + Hybrid. The cost benefit and cycle time benefit of FC as a mesh replacement was well discussed in Davidson (2005). The paper also notes that SS are used for holding the mesh up rather than providing ground control.

3.3 Zone 3

The rock mass is fresh and competent with demand driven by the in situ and induced mining stresses. Toughness of the GSS is a consideration. Typically, strainbursts/rockbursts are limited and damage, if it occurs, is minor, with bulking of the excavation and minor damage beneath the height of the installed GSS. Rock noise is common and seismic systems are typically installed to monitor seismicity. Not all areas of the mine will have geotechnical challenges in Zone 3 with issues usually starting due to stress concentration as a result of mine design, intrusives (stiffness contrast), adverse structures (displacement, rock noise), and orientation of drive perpendicular to principal stress in anisotropic rock mass.

The main issue is progressive overstressing and rock mass yielding/bulking. The GSS uses FC to provide initial confinement, with an outer layer of weld mesh for deformation capacity. The reinforcement systems are based on solid bars – either hybrid or gewie bars – as the bulking of the rock mass results in increased damage zones and pull out or failing of the collar ring of CFC reinforcement.

Hybrid reinforcement systems rely on significant increase of friction at the toe, and they do not have the mechanical interlock of cement or resin, and generally do not have the mechanical interlock of now 'old-fashioned' mechanical shell anchor bolts. Whilst the hybrid bolts achieve rapid installation and high capacity, they are not a universal bolt to all situations.

Variations in the GSS in Zone 3 are significantly impacted by the degree of rock mass anisotropy or reduction in rock quality. There are 5 typical variations considered, and they are often based on monitoring or observation performance of the GSS:

- Change from fully bonded gewie bars to centrally debonded gewie bars to increase the deformation capacity and toughness, which is particularly useful where guillotining of the reinforcement system occurs.
- Increase the length of reinforcement systems.
- Charge reinforcement system density (with consideration of mesh overlap locations).
- Increase surface support coverage to within 0.5 m of the floor to control 'guttering' up the walls.
- In a transition between Zones 2 and 3 not using the FC but still using the selected reinforcement system.

In anisotropic rock mass, the amount of seismicity may decrease and the amount of bulking shear and deformation can increase. It is important to realise that anisotropic rock masses can also suffer rockbursts and the orientation of the drive relative to the anisotropy (parallel to strike is significantly less favourable than perpendicularly) and the dip of the anisotropy (45–70˚, is more prone to deeper failures than shallower or steeper dipping anisotropy).

The significance of drive orientation on excavation performance (particularly with adverse stress orientation) is due to undercutting of the foliation, with shearing in the backs plus hanging wall shoulder and footwall toe, and buckling along the hanging wall. This is explored by Sandy & Player (1999, shown graphically in Figure 3 of that paper), Player (2004), and more extensively in Sandy et al. (2007) and Vakili et al. (2013) which also discuss the transition from Zones 3 to 4 for anisotropic rock masses.

In adverse cases (stiff, very high strength rock mass), the response can appear to bypass Zone 3 type deformation response of bulking and shearing, instead exhibiting as Zone 4 with strainburst/rockburst.

Detailed discussion of GSS design, performance and assessment in Zones 3 to Zone 5 are presented in Villaescusa et al. (2023).

3.4 Zone 4

A high strength and competent rock mass, with stress damage exhibiting as spalling and strainburst/rockburst, becomes the primary concerns in this rock mass and stress state. The change failure mode is described graphically by Kaiser et al. (2000) in the commonly used chart relating Geological Strength Index, strength, and stress to failure characteristic.

The challenge is the ability to forecast a stainburst rock mass response prior to selection of the ground support scheme. Diederichs (2018) provides a reasonable process for taking material properties and induced stresses through to a burst hazard potential (BHP). The local calibration factors though have a significant impact for the forecast result (BHP). This is expanded to a Finite Discrete Element Modelling solution by Vazaios et al. (2019) which works to understand how strain energy can be stored in the rock mass and how it can be released following excavation. Key relations of unconfined compressive strength, k-ratio, stiffness contrast, change in confinement, presence of fabric/structure, excavation scale and profile are explored in the role of transfer of strain energy to kinetic energy. The role in structure and anisotropy in increasing the depth of failure from the strainburst are identified by the test cases and the complexity of mechanisms at play.

Askaripour et al. (2022) grouped prediction methods into 2 broad primary fields of stress methods and energy methods. With over 30 tests recognised, no singular method was identified and more research work is recommended for combining material properties and stress state. However, the work does not consider the role of loading stiffness from the mine volume or stiffness contrast between exposed rock masses in an excavation, and the majority of the test only considered a limited number of parameters, whereas Vazaios et al. (2019) has 6 parameters plus multiple material properties available to adjust for model calibration.

Hence, the spatial and temporal aspects that controlled why a strainburst or rockburst occurred maybe recognisable retrospectively involving a forensic review of the failure of the rock mass and the ground support scheme. Whilst the work by Vazaios et al. (2019) is encouraging many researchers to follow their approach.

Seismic risk is partially forecastable (stiffens and strength of rock mass, degree of jointing, in situ stress and drive orientation) but a knowledge of the seismic hazard, how big will the event be, plus when and where it will occur is needed to define the seismic risk to the excavation. The greatest uncertainty that remains is why source initiation occurs at a particular location as small differences in location on a mine scale can make very large differences in excavation performance.

To mitigate this risk of dynamic failing rock mass loading the installed ground support schemes, pre-emptive measures of improving ground support scheme capacity and toughness should be undertaken to ensure worker safety and orebody access.

Three different GSS have been compared with regards to productivity and cost. The differences of the GS4 series to the earlier GSS, and between the GS4 series, are:

- Increased productivity with the use of emerging technology and practices achieved by pumped resin rather than cartridge resin. The Geobrugg system is preferred. The approach has the resin pumped through the drill steel to the back of the hole and continued to pump and fill the hole as drill steel retracted. The mine then installs their selected reinforcement system. Pumped resin is more effective and faster than cartridge resin at encapsulating the bolt particularly in high stress conditions with borehole stability problems.
- Centrally debonded gewie bar bolts with resin anchoring to the rock mass at the toe and the collar. The collar bonding reduces loading force to the surface fixture, the central debonding allow energy dissipation of the strainburst/rockburst by yielding at high resistive force, the bonded toe section to hold the end of the bar into the hole. Centrally debonded gewie are chosen to replace hybrid bolts as observations have shown hybrids failing near the collar (likely over-tensioning, angle of installation/tensioning) when close to strainbursts/rockbursts (but not within the fall of ground)

and pulling out when within strainbursts/rockbursts as depth of failure increases due to reducing in resistive force to sliding.

- Increase in surface support coverage, bring FC and mesh down the walls to near floor level to control breakout on the walls. With the mesh lower on the walls, an additional layer (not exceeding 50 mm of FC) is sprayed on top of the mesh to prevent equipment damage.
- Addressing the common understanding of rockburst damage occurring at the weakest link of the GSS by replacing WM sheets with a roll of Minax or WM+Osro straps. Minax (GS4.0) is recommended due to the removal of 6 mesh overlaps that are required to be reinforced with additional bolting and the Osro straps (GS4.2).
- Increasing capacity of the support systems with the use of Minax compared to WM (Villaescusa et al. 2023; Brändle & Bucher 2023) and as part of a complete GSS (Brändle et al. 2022).
- The speed of installation for an effective GSS is a primary component to total cost and to achieve the increase the capacity with GS4.2 results in slower heading turnover and elevated cost.

One of the earlier papers on the use of the Minax to control the rockburst damage from a significant event that results in major damage to a scheme similar to GS4.2 is provided by Arcaro et al. (2021). The paper shows the success of the scheme in maintaining drive stability to strainburst and rockburst events, in rehabilitation of existing development, installation in new development, and the open stope brow performance.

The significance of the response of the surface support in controlling rockburst damage is examined in a series of large-scale tests by Cuello et al. (2024), with key findings of stiffness and capacity of the complete ground support scheme and the force–load transfer between support and reinforcement systems in the damage zone and those in stable zone is significant in determining the outcome of performance, particularly when using composite support systems.

Variation in GS4.0–GS4.2 can be achieved by variation to the reinforcement system capacity as part of the GSS capacity by increasing 3 parameters of density, bar diameter and bolt length of the debonded gewie bar. There are 2 key aspects that aid productivity; the pumped resin and the high-tensile chain link. The Minax approach of surface support is also more adaptable for changes in reinforcement system spacing as the only overlap location is from one cut to the next allowing the capacity of the GSS to be easily upgraded. Which can be done trailing heading advance.

Additional measures of de-stress blasting maybe required to make additional fractures around the excavation to relieve the stress/strain loading of the rock mass. As this has the same cost impact for the considered options, it is not included in the costing and productivity model. There are many papers on the design, implementation and success or otherwise of de-stress blasting, and Drover et al. (2022) is an excellent example of the requirements for de-stress blasting to be effective in changing the rock mass response.

3.5 Zone 5

The rock mass setting and its response continues from Zone 4, and with the increasing depth and stress state, the straining of the rock mass increases. Hence, the event magnitude increases and the role of structure in enabling rockburst events (whilst still being crush events on the Hudson plot) prompts deeper failures. Therefore, a requirement for cable bolts to achieve deeper embedment exists.

The role of structure contributing to rockburst events in close proximity to the excavation is identified in:

- the practical applied ground support scheme by Arcaro et al. (2021) from visual observation of performance and design to prevent the failure mechanism (also the important of corrosion protection for the barrel and wedge, and applied tension to the strand)
- the numerical modelling of rockburst by Vazaios et al. (2019) with the accumulation of strain on structure
- the depth of failure data within Heal (2010), filtering for R4, R5 damage of drives 5–6 m in width that shows an outlier population for depth of failure to the main population of the depth of failure.

Cable bolt strands use stress-relieved, high-tensile wire with little remaining plastic yield prior to rupture. Hence, the principal way the strand functions to dissipate energy is compaction of the strand under axial load and decoupling from the encapsulation medium, and ultimately sliding of the strand within the encapsulation medium. Therefore, the strand cannot be bulbed as that prevents the yielding of the cable within the grout. Changing of the encapsulation medium from cementitious grout to resin would impact the response of the strand to dynamic load, reducing the amount of deformation and hence energy dissipation.

A single 17.8 mm plain strand is chosen rather than twin 15.2 plain strands to ensure that correct tension of the strand occurs, that the second strand has not wrapped around the first strand for installation with cable bolters, and a balance of spacing of the strand is maintained, securing mesh overlaps and total scheme deformation.

The inclusion of cable bolts with cement grout in the ground support scheme can significantly increase the time to turn the heading over, particularly if required in-cycle. In these situations, the total number of headings is critical to maintain productivity and mine advance rates. The slower advance rate of the heading with in-cycle cable bolting can have a beneficial effect of allowing some strain dissipation within the rock mass across the full cycle.

4 Cost and cycle time

The cost and cycle section of the paper are separated into the inputs section and the outcomes of the study from a cost and time perspective.

Cycle times were developed to establish the possible number of metres advance that are attainable over a period of 1 month. The metres per month were then utilised in the costing model to generate the total cost for the month for the 5 zones of development with the associated GSS.

4.1 Inputs

The cost and cycle time model is developed from first principles calculations based on 3 jumbo development scenarios; a single heading with independent firing, a single heading with set firing times, and multiple headings.

4.1.1 Single heading

Single heading independent firing assumes that a development heading can be blasted as soon as charging is completed and the heading is ready to fire. This minimises delays in the development cycle and improves overall development productivity.

Single heading with firing times results in a delay in advance whilst waiting for the designated firing time. No cycle time work can take place in the heading resulting in low jumbo productivity.

Single heading means that there is only a single heading for the jumbo to advance. Following completion of drilling and ground support installation, the jumbo must wait until the remaining development cycle activities (including charging, firing, ventilation and re-entry, and bogging) have been completed before it can return to the heading and commence the next development cycle. This results in increased jumbo waiting time, reduced equipment utilisation, and lower monthly development advance.

4.1.2 Multiple headings

Multiple heading development assumes that more than one heading is available for the jumbo to advance. Once drilling and ground support installation have been completed in one heading, the jumbo can relocate to another available heading while charging, firing, ventilation and re-entry, and bogging activities are undertaken in the first heading. This allows development activities to be undertaken concurrently across multiple headings, reducing jumbo waiting time and increasing equipment utilisation. Consequently, multiple heading development generally achieves higher monthly development advance than single heading development

4.1.3 Input tables

The cycle time included; face drilling, charge up, firing time, ventilation clearance, watering down, loading out of fired rock, ground support installation, rebogs (as required by cycle chosen), delays, cable bolting (as required), and setup times.

The inputs (cost and cycle times) are summarised with tables of the inputs costs and times included in Appendix B.

- All months are 30 days.
- Operator costs are based on indicative contractor pay rates.
- Electrical power costs for jumbo and secondary ventilation are included.
- Truck haulage tonne-kilometres (tkm) are included with additional trucks being added as depth of mining is increased. The tkm calculations also include a 500 m haul on the surface to a waste dump.
- The travel times are increased to the heading as the mining depth increases (effective work time decreases).
- Operating costs (per hour or drill metre) and major plant capital are based on Sandvik costs for equipment and their indicative operating costs.
- Equipment value depreciation uses a flat line depreciation model over the life of the equipment.
- GSS components, mining consumables, and blasting consumables are based on indicative and contract rates from suppliers.
- Mesh products allows for 300 mm sheet overlap.
- FC quantities allows for roughness profiles and rebound.
- Capital heading is the same size and uses 63 blastholes and 4 reamers for all zones. The inclusion of additional non-charged holes for maintaining profile in Zone 1 or de-stress blasting holes in Zones 4/5 are not included in the cost model.
- Time to drill the face and rockbolt holes are consistent at 1.8 drill metres per minute (not modified for smaller resin bolt hole diameters).
- Length of cut is variable in the cost model, but all have been set to 4 m advance from drilling 4.3 m for the study except for GS1 which is 3 m in length for 3.3 m of drilling.
- There is 200 mm overbreak of walls and backs to allow for additional material movement (i.e. above design quantities).
- Time to install system components is based on tender documentation and supplier information.
- The depth for each ground support zone at which each scenario was costed was taken as the midpoint between the ‘From’ and ‘To’ depths listed in Table 2. This depth was used as the depth for tkm calculations to ascertain the haulage component of the cost.
- All development metres were assumed to be excavated at the depth of the applicable ground support zone and within reasonable distances that the development could be undertaken with a single jumbo.
- The total cost for a month of development was estimated on the basis of development and ground support only (i.e. no production activities), taking into account plant and equipment, labour, explosives, drill consumables, power, fuel, and general mining consumables. All the costs were consistent over the various scenarios. The total estimated cost for the month was then divided by the total metres developed in the month, to calculate an all up cost per metre of advance.

Excluded costs:

- Personnel and associated labour costs above the site foreman level.
- Primary ventilation and mine dewatering costs.
- Corporate overhead costs and contractor markup.
- Stripping.

4.2 Outcomes

The outcomes are presented graphically and tabulated:

- Capital advance rate for the 3 scenarios for the 5 zones (Table 4 and Figure 3).
- Calculated cycle times for all scenarios (Figures 4–6 and Appendix C).
- Cost per metre for the GSS for single heading independent firing (Zones 1–3) and multiple headings end of shift firing (Zones 1–5) (Table 7 and Figure 7).
- Proportion of the ground support cost of the total development cost (Figure 8).

The results should be viewed in terms of a comparison between zones to understand the change in cycle times and costs, and the difference between GSS.

4.2.1 Jumbo productivity

The jumbo is significantly more productive with multiple headings available compared to single heading advance (Table 4). This difference is larger than any impact that the selection of the GSS has on cycle time. The jumbo achieves 1.5–2.25 times as many metres on muti heading for GS1–GS4 and the 2.5–3 times as much for GS4.x CB when cable bolts are required to also advance the development.

An additional outcome for the productivity is the use of the pumped resin system in GS4.0 compared to the slower progress from cartridge resins (compared GS4.0 to GS4.1), with a difference of 22 m for a single heading and an additional 86 m per month for the multi-heading.

Table 4 Possible development metres per month: one jumbo

Support profile	Single heading independent firing (m)	Single heading end of shift firing time (m)	Multiple heading end of shift firing time (m)
GS1 Gewie+M+FC	97	81	176
GS2.0 SS+M	181	141	304
GS2.1 Hybrid +FC	154	124	347
GS3.0 Hybrid+M+FC	137	113	281
GS3.1 Gewie+M+FC	123	103	228
GS4.0 debonded gewie+Minax+FC	130	107	281
GS4.1 debonded gewie+Minax+FC	108	92	195
GS4.2 debonded gewie + M + FC + Osro	89	78	141
GS4.1 + 10 cables	50	46	151
GS4.2 + 10 cables	47	44	133
GS3.0 deep hybrid+M+FC	137	113	281

The availability of multiple headings (whilst sometimes not possible, particularly in early mine development) can be considered a function of the mine design. The number of headings has been calculated to determine the minimum required to achieve the efficiency of multi-heading (Table 5). It is determined by using the hours per heading for a single heading end of shift firing (Figure 5) rounded up to the next shift and dividing that by the number of jumbo hours for the multi-heading end of shift firing. The number of headings is then rounded up.

Table 5 Number of headings required to make multi-headings possible to be valid

Support profile	Single heading hours, end of shift firing (hours)	Round up hours to next shift (hours)	Jumbo hours multi-heading (hours)	Column 2/ Column 3 (heading)	Minimum required headings for multi to be valid (heading)
GS1 Gewie+M+FC	18.22	24	8.33	2.9	3
GS2.0 SS+M	13.70	24	6.35	3.8	4
GS2.1 Hybrid +FC	15.60	24	5.57	4.3	5
GS3.0 Hybrid+M+FC	16.71	24	6.68	3.6	4
GS3.1 Gewie+M+FC	18.28	24	8.25	2.9	3
GS4.0 debonded gewie+Minax+FC	16.89	24	6.40	3.7	4
GS4.1 debonded gewie+Minax+FC	19.75	24	9.26	2.6	3
GS4.2 debonded gewie + M + FC + Osro	23.29	36	12.80	2.8	3
GS4.1 + 10 cables	38.92	48	11.67	4.1	5
GS4.2 + 10 cables	40.50	48	13.25	3.6	4
GS3.0 deep hybrid+M+FC	16.71	24	6.68	3.6	4

The number of headings required increases when the jumbo time is low relative to other components of the cycle, i.e. GS2.1 which requires 5 headings to keep the jumbo fully utilised.

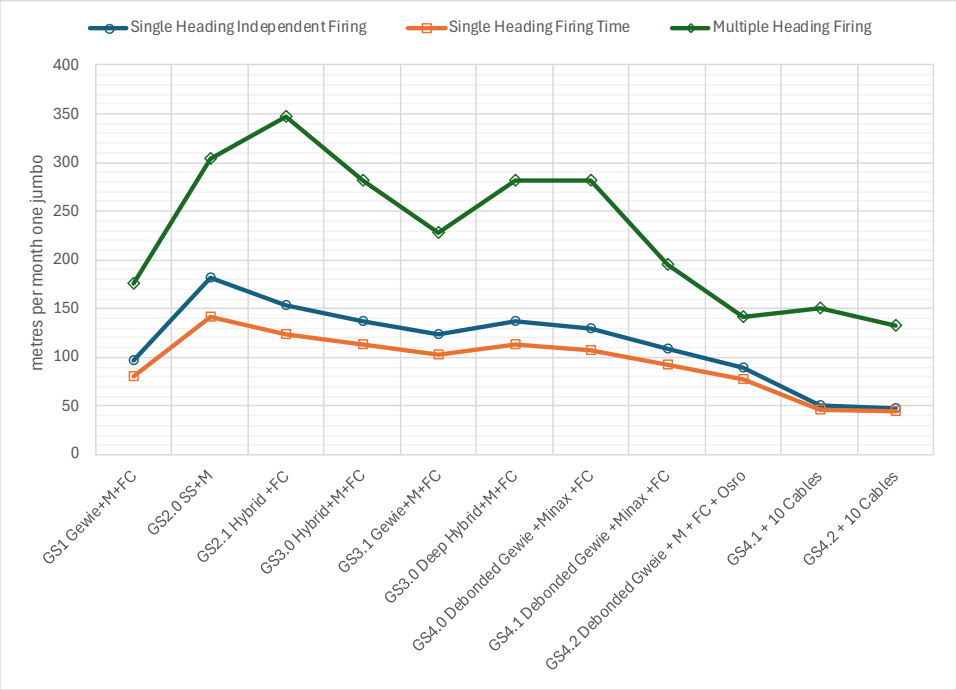


Figure 3 Development metres per month by scenario and ground support scheme installed

The inputs to achieve the metres per month are tabulated in Appendix C for the 3 development scenarios and shown graphically in Figures 4 to Figure 6. The figures show all components, although some do not impact the cycle time of the jumbo for the multi-heading scenario. No allowance has been made for charging and drilling issues or longer rebog, due to face instability with highly stressed ground in Zone 4 and Zone 5.

The key outcomes from the detailed scheduling for the single heading scenarios is that grout curing for cable bolting is the most significant time penalty for single heading and also extends the number of headings required for efficient jumbo productivity. Options of a tighter reinforcement pattern (1.1 × 1.4 m pattern to a 1.1 × 1.0 m pattern) and longer resin encapsulated bolts (from 2.4 to 3.0 m) could be evaluated.

The key outcomes from the detailed scheduling for the multi-heading scenarios are:

- Multi-heading is more efficient rather than having the jumbo waiting for the other cycle steps.
- FC is faster than weld mesh, with additional cost and time benefits of wider bolt patterns from using pumped resin encapsulated bolts.
- There is not a significant time penalty for the introduction of FC prior to meshing (and the second pass FC is done over the lower mesh lagging by one cut).
- Pumped resin for ground support installation takes half as long as cartridges in highly stressed ground where hole collapse is an issue.
- It takes 50% more time in multi-headings using weld mesh with Osro straps compared to using Minax mesh).

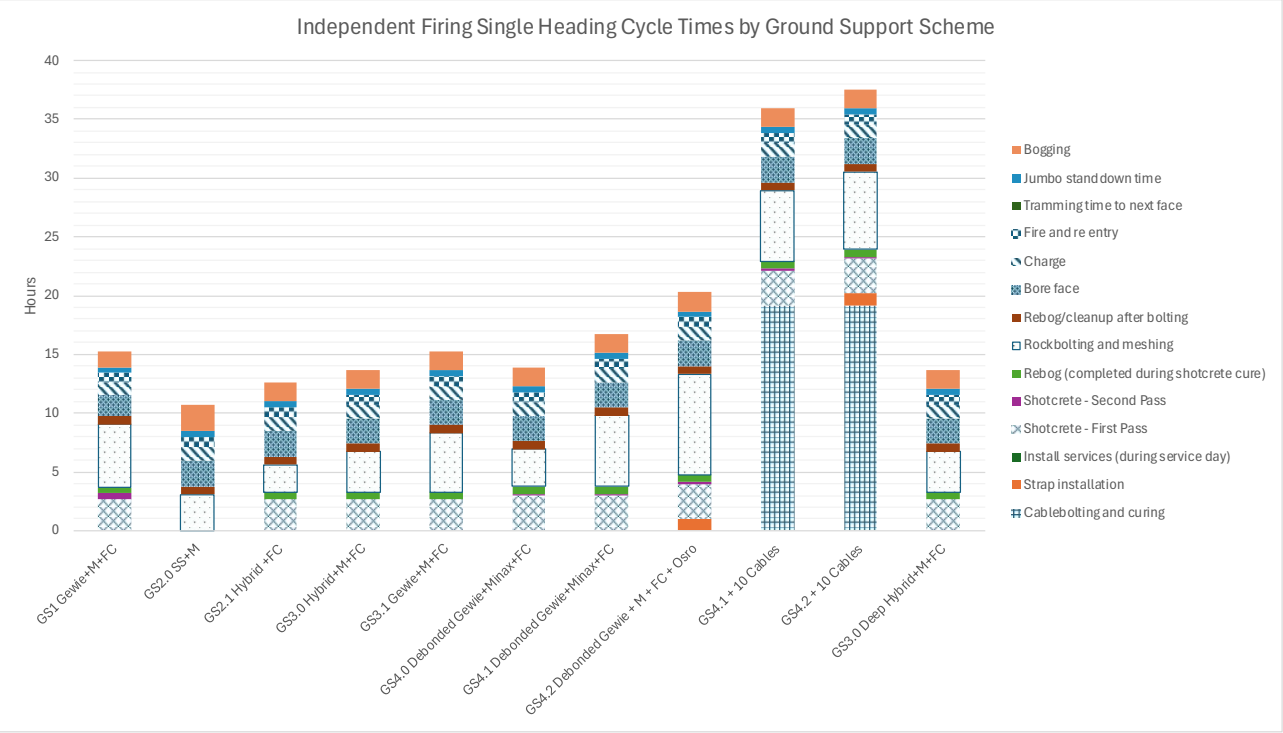


Figure 4 Components that contribute to cycle time for single heading independent firing

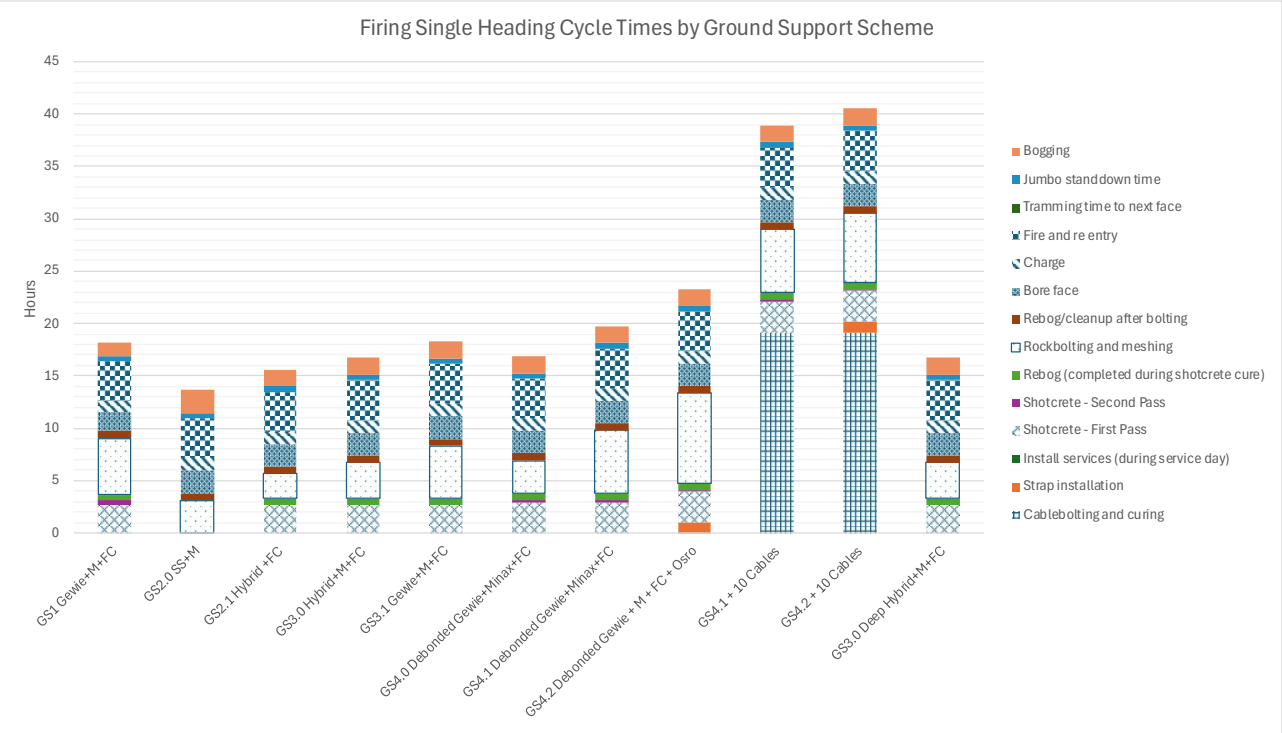


Figure 5 Components that contribute to cycle time for single heading end of shift firing

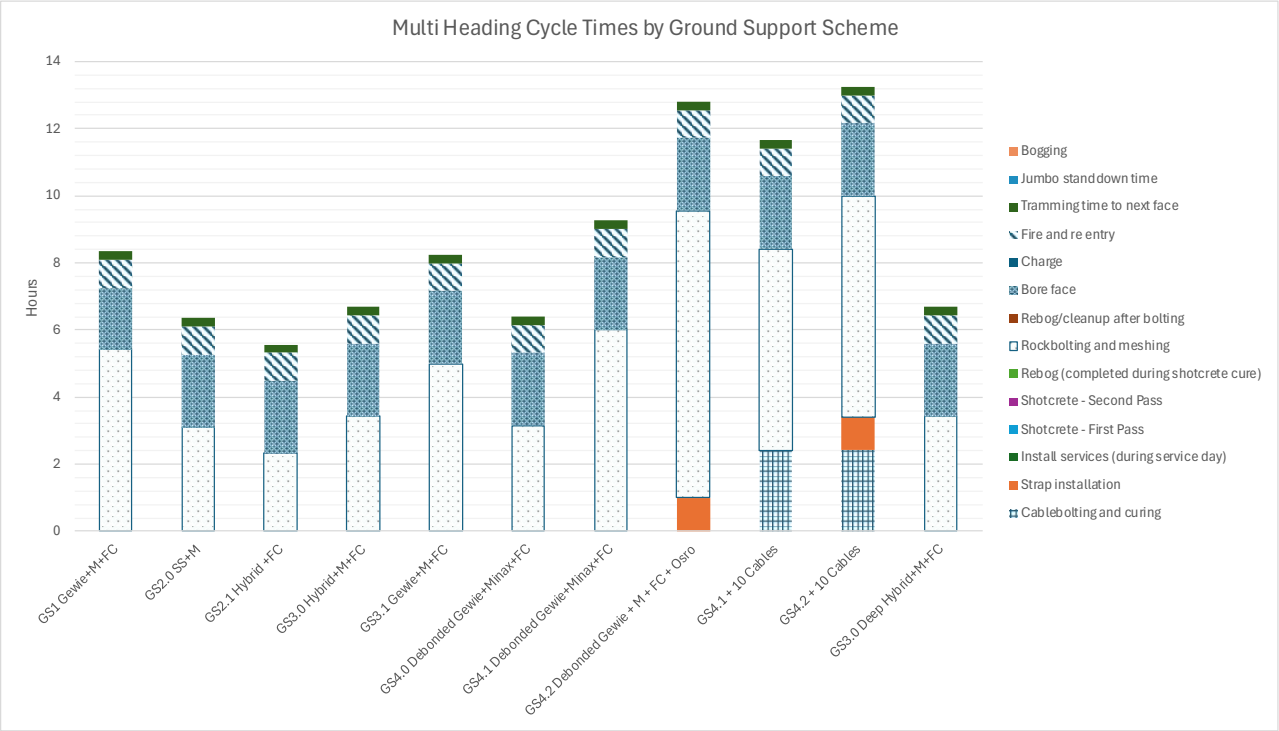


Figure 6 Components that contribute to cycle time for multi-heading end of shift firing

4.2.2 Cost per metre of development

The cost per metre of development (Table 7) is a summation of the costs associated with the development for the month (Table 6) and dividing by the development metres (Table 4). These costs include the maintenance and supervisory team with the associated mining equipment (provided in the plant table) supporting a single jumbo. The maintenance and supervisory team should be able to handle a somewhat large fleet without additional manning costs.

The single heading cycle time, when constrained by firing times, achieves fewer metres per month. Its cost per metre is higher and the option is not discussed as it's an unlikely development scenario.

The independent firing of a single heading cycle time is not costed for Zones 4 and Zone 5 as it's an unlikely development scenario.

Table 6 Total development cost for the month

Support profile	Independent firing (AUD M)	Firing time	Multiple headings (AUD M)
GS1 Gewie+M+FC	2.083		2.439
GS2.0 SS+M	1.905		2.204
GS2.1 Hybrid +FC	2.205		2.746
GS3.0 Hybrid+M+FC	2.250		2.749
GS3.1 Gewie+M+FC	2.181		2.528
GS4.0 debonded gewie+Minax+FC			2.999
GS4.1 debonded gewie+Minax+FC			3.204
GS4.2 debonded gewie + M + FC + Osro			2.758
GS4.1 + 10 Cables			2.509
GS4.2 + 10 Cables			2.625
GS3.0 deep hybrid+M+FC			2.544

Table 7 Cost per metre by development scenario

Support profile	Independent firing (AUD/m)	Firing time	Multiple headings (AUD/m)
GS1 Gewie+M+FC	21,470		13,860
GS2.0 SS+M	10,530		7,250
GS2.1 Hybrid +FC	14,320		7,910
GS3.0 Hybrid+M+FC	16,420		9,780
GS3.1 Gewie+M+FC	17,730		11,090
GS4.0 debonded gewie+Minax+FC			11,770
GS4.1 debonded gewie+Minax+FC			14,140
GS4.2 debonded gewie + M + FC + Osro			17,790
GS4.1 + 10 Cables			17,390
GS4.2 + 10 Cables			19,130
GS3.0 deep hybrid+M+FC			10,950

The greatest understanding of the cost per metre of development is obtained by plotting the cost per metre on the y-axis and jumbo development metres per month on the x-axis, with the data points by development scenario and GSS (Figure 7). The most significant outcomes are:

- Independent single heading firing compared to the multiple heading firing are separate populations.

- Single heading use of FC increases costs by ~35% in low rock mass demand situation but when part of multiple headings, the increase is <10% (GS2.1 to GS2.0).
- Comparing GS2.0 and GS2.1 when heading moves from single heading independent firing to multiple heading firing GS2.1 becomes faster/achieves more metres than GS2.0 because the jumbo is only bolting and not installing mesh as FC curing is separate (as long as there is sufficient headings).
- The use of cartridge resin in overstressed ground increases the cost by ~20% due to longer cycle times when compared to pumped resin (GS4.1 to GS4.0).
- Osro strap GSS development are ~50% more than the cost of Minax mesh with pumped resin bolts (GS4.2 to GS4.0).
- GS4.0 pumped resin debonded bolts with Minax mesh is within ~7% of the GS3.0D GSS options with significantly higher GSS toughness and capacity.
- GS1 and GS4.1 are similar in cost and metres advance because of the use of resin cartridges in broken ground (either due to poor rock mass condition or overstress causing hole collapse).

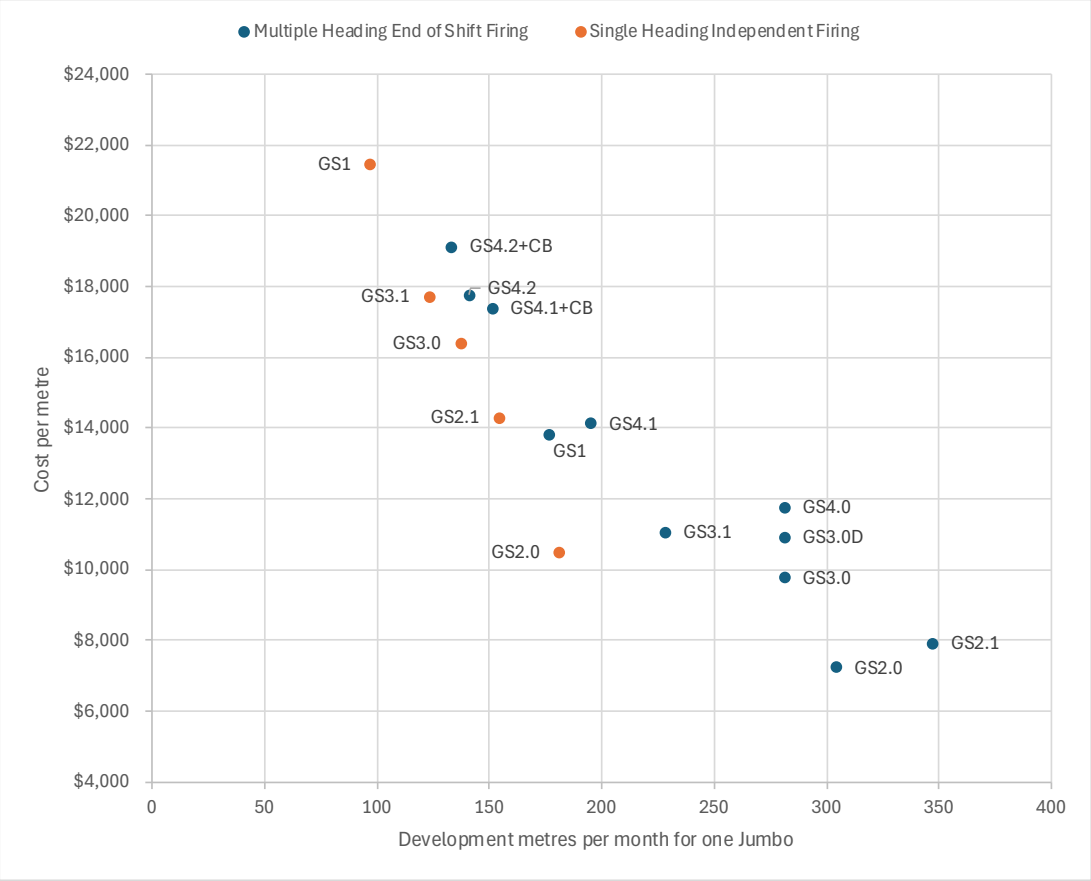


Figure 7 Calculated cost per metre and development advance by the jumbo

4.2.3 Cost of ground support scheme as percentage of development cost

The cost of the GSS is evaluated as 2 alternate percentages. Both are correct but give 2 potentially different views on percentage costs based on the inclusion or exclusion of relative costs:

1. The only included activities relate to the single jumbo development with the associated ground support and haulage. The cost utilised in this analysis only includes ground support material costs.

2. The only included activities relate to the single jumbo development with the associated ground support and haulage. The cost utilised in this analysis includes the direct costs such as the relative proportion of the jumbo capital cost, operator labour, drill consumables (based on the time of the activity), and the ground support material costs.

The cost comparison of the GSS materials is evaluated in Figure 8:

- The percentage of the consumable costs of the ground support materials in a single heading independent firing is less than the scheme for a multi-heading.
- Adding FC significantly increases the percentage of the ground support costs of the development cost in both situations.
- The relative order of costs exists between single independent firing and multiple end of shift firing showing the significant impact of the heading access on total costs.

The y-axis is extended to 35% to compare to the costs in Figure 9. The cost comparison of the material and relative proportion of the direct costs is shown in Figure 9:

- The inclusion of FC can double to triple the percentage of the GSS and relative direct cost of the total and hence becomes a significant consideration in mines decision to implement (when only evaluated on relative cost).
- Once FC is used, the percentage cost differential can still be relatively significant depending on the cycle times that are achieved.
- Nearly all other GSS (materials plus relative cost of equipment) brackets are within 20–30% of the total costs which means trying to optimise on unit cost alone will not have a significant impact on the total development cost.

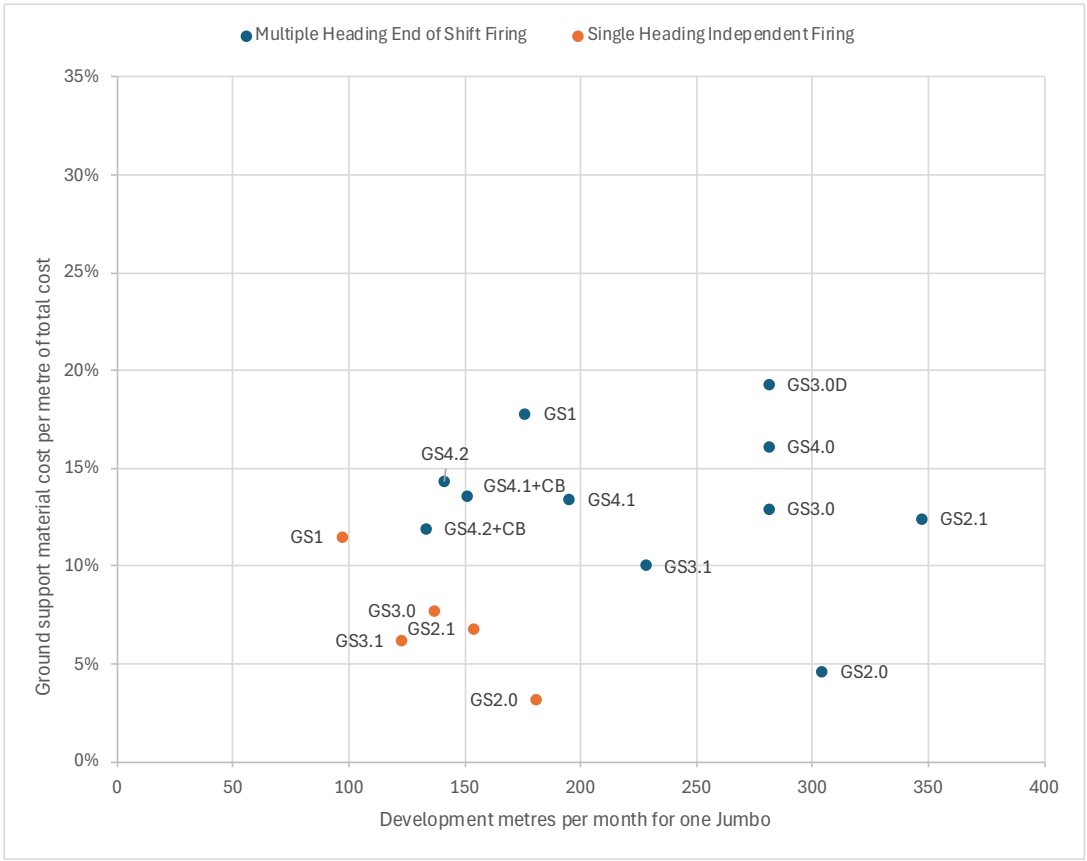


Figure 8 Ground support scheme material cost as a percentage of total cost for development by ground support scheme type and development scenario

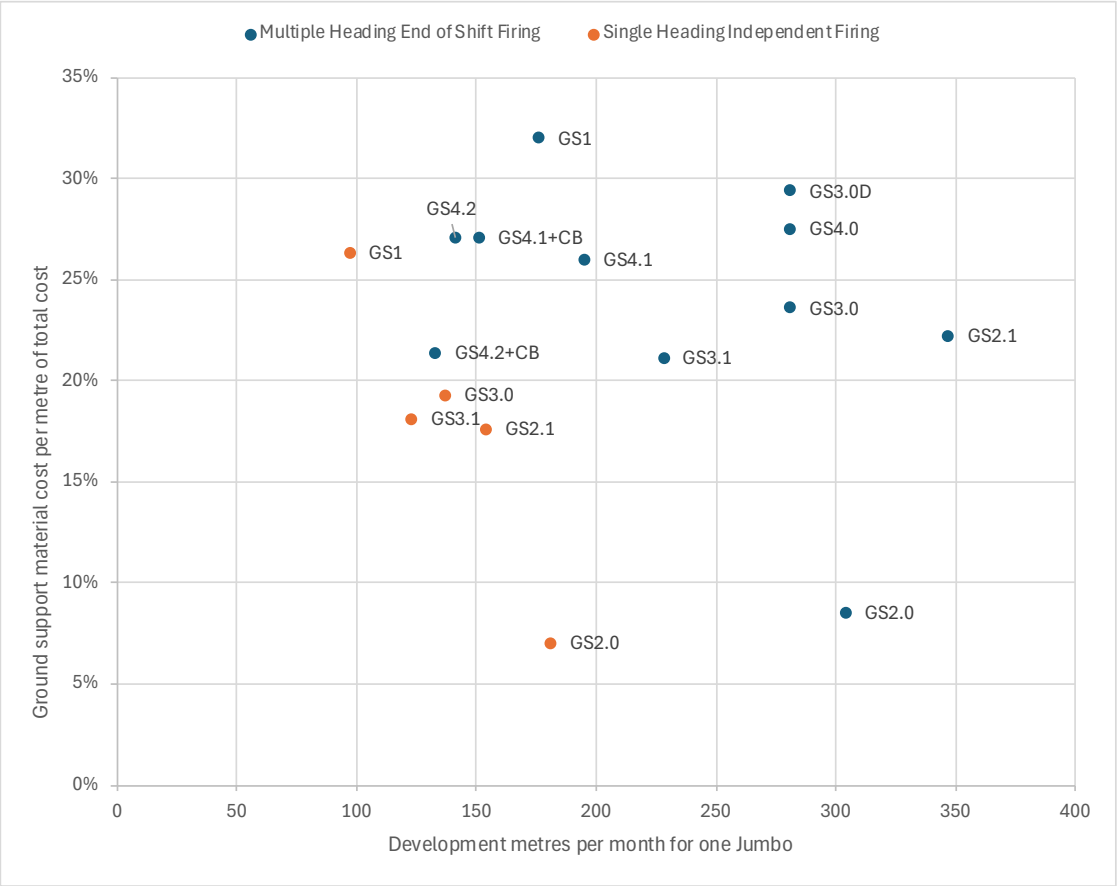


Figure 9 Ground support scheme material cost and the relative proportion of direct costs as a percentage of total cost for development by ground support scheme type and development scenario

5 Assessment of potential loss in value

In this paper, the value is assessed by the loss in profit due to a single fall of ground in the capital development. This analysis does not consider the additional fixed costs from extending the mine life or undertaking the rehabilitation to recover the decline and upgrade the ground support standards.

This comparison test extends the use of GSS from low demand into a Zone that requires a higher demand.

For example, an underground gold mine producing 750,000 tpa at 4 g/t decides to maintain GS3.0D in the capital development rather than change to GS4.0 or GS4.1 or GS4.2. This has been undertaken for 5 levels at a 25 m vertical metre level intervals. The incremental cost per metre for multi-heading is shown in the third column in Table 8 and the total development cost difference is shown in the fourth column.

The table clearly shows that the slower installation rates of cartridge resin in over stressed ground and then particularly the incorporation of Osro straps to strengthen the weld mesh overlaps (GS4.2) have a significant negative increase to cost and loss in productivity (Table 4).

Table 8 Comparison in cost on selection of ground support scheme

Support profile	Multiple heading (AUD/m)	Difference to GS3.0D per metre (AUD/m)	Difference per metre (%)	Additional development cost (AUDM)
GS3.0 deep hybrid+M+FC	10,950			
GS4.0 debonded gewie+Minax+FC (pumped resin)	11,400	819	7	0.717
GS4.1 debonded gewie+Minax+FC (cartridge resin)	14,140	3,196	29	2.797
GS4.2 debonded gewie + M + FC + Osro	17,790	6,839	62	5.984
GS4.1 + 10 cables	17,390	6,437	59	5.632
GS4.2 + 10 cables	19,130	8,183	75	7.160

The project risk is evaluated by assessing the lost profit in one month due to a range of tonnage reductions as a result of a fall of ground in the capital development (Table 9). The profit per ounce after the fall of ground can allow for change of site profitability due to the relative increase in fixed costs.

Table 9 does not consider the cost to rehabilitate or replace the required levels as each site will make its own decision with regards to upgrading or replacement of development depending on the severity of the event. Adding these costs effectively reduces the amount of lost ounces for an equivalent value.

The table clearly identifies why there is a reluctance (when only ground support costs and not total costs are evaluated) to increase the GSS capacity in the capital development until there is a safety case or lost production case due to damage to the rock mass and GSS. Cost for increase in capacity from GS3.0D to GS4.2 for the 5 levels is essentially the same as the lost profit (at AUD 1,500/oz) for half the mine tonnage for one month.

However, new GSS standards such as GS4.0 with the pumped resin change this difference to less than 4 days of 50% lost production at AUD 1,500/oz profit.

Table 9 Lost tonnage and ounces per month due to capital development fall of ground

	Lost Ounces	4,020	2,650	2,010
	Lost Tonnage	**50%**	**33%**	**25%**
		(AUD M)	**(AUD M)**	**(AUD M)**
Profit per ounce	250	1.005	0.663	0.502
	500	2.009	1.326	1.005
	750	3.014	1.989	1.508
	1,000	4.019	2.652	2.009
	1,500	6.028	3.979	3.014

Mining companies in high stress environments will make significant savings with the installation of high toughness GSS (such as GS4.0) that can be significantly upgraded in capacity with additional bolting installed. Hence the emphasis should be on making the installation practice of pumped resin and chain link work with support (training and salary of the jumbo operators while acquiring the skills to use the new schemes) rather than trying to optimise lower capacity schemes that are more expensive.

6 Conclusion

The evaluation of the GSS capacity and toughness to rock mass demand has been evaluated for 5 zones of development from near surface to approximately 2,000 m below surface. The recommended GSS should be considered as starting points for each individual mine as they assess their rock mass demand followed by response of the GSS to strainbursts and rockbursts.

The paper calculates the cost of the GSS by component (5–20% of the total development cost) or component plus installation allowing for relative equipment usage (which makes up 15–30% of the total development cost).

A focus on this relatively small cost (GSS as a percentage of total) implies a degree of certainty of the dynamic rock mass failure mechanisms and location. In comparison to selection of a GSS performance achieved by a modern ground support standard, GS4.0, which mitigates the uncertainty. The increase in capacity and toughness plus the reduction in cycle time of GS4.0 with pumped resin identifies that improvements in GSS do not need to wait until after a rockburst event has caused damage to an excavation. GS4.0 will improve the operations profit when installed where stress conditions and rock mass responses are forecast to be dynamic due to the security of excavation performance.

The paper then demonstrates that the total cost of the actual advance rate of the jumbo, particularly in high stress conditions, is not particularly sensitivity to the GSS toughness and capacity but rather on the productivity achieved in the installation of the GSS.

Acknowledgement

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Richard Gorzkos of Ripago Underground Mining Services Pty Ltd is also thanked for the multiple costing models used in this paper, as well as Richard Varden and Nicole Player of MineGeoTech for their review, input, support, and discussions during this course of preparing this paper.

Appendix A Ground support patterns

GS1 Gewie + M + FC

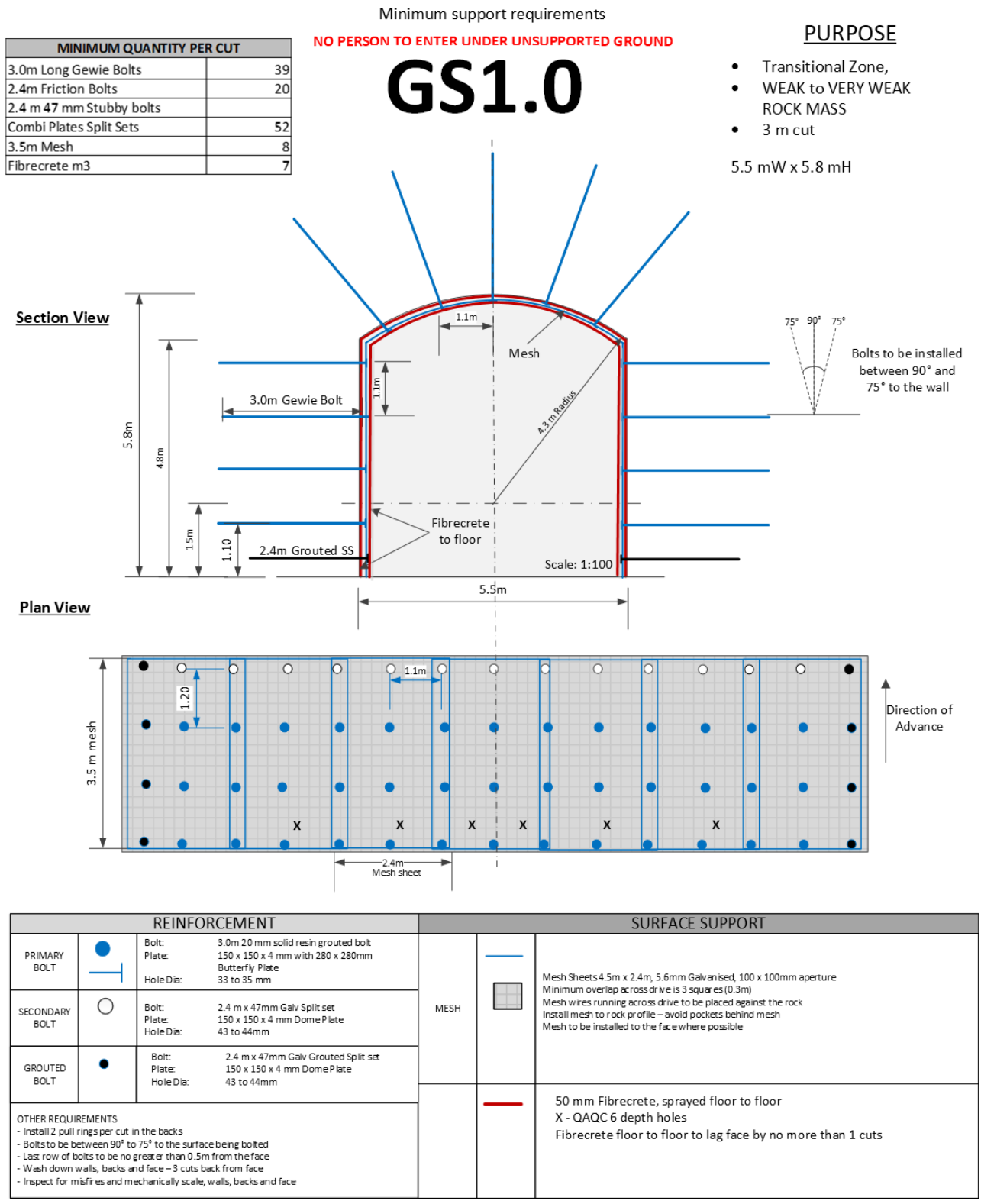


Figure 10 GS1.0 Gewie + Weld Mesh + Fibrecrete

GS2 SS + M

Minimum support requirements

MINIMUM QUANTITY PER CUT	
2.4m Long Split Set Bolt	33
0.9m Long Stubby	11
Combi Plates	33
Dome Plates	11
Mesh 4.5 m	5
Pull Rings	2

NO PERSON TO ENTER UNDER UNSUPPORTED GROUND

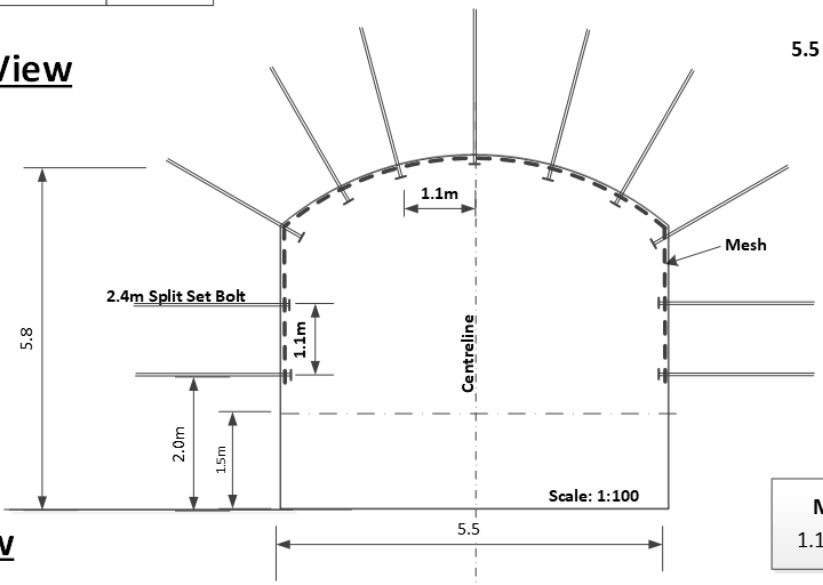
GS2.0

PURPOSE

- To be utilised in **ASEISMIC conditions** *(Not Seismic)*
- Good rock mass
- Mesh to be maximum of 2.5 m off floor.
- 4 m cut

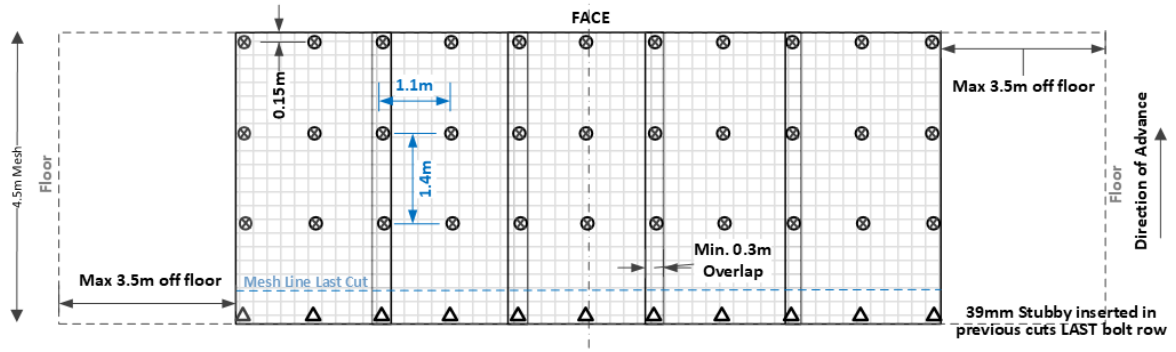
5.5 m W x 5.8 m H

Section View



MAX. BOLTING PATTERN
1.1m Spacing x 1.4m Burden

Plan View



REINFORCEMENT			SURFACE SUPPORT		
PRIMARY BOLT	⊗	**Bolt:** 47mm Galv Split Set Bolt **Plate:** 300 x 280mm Combination Plate **Hole Dia:** 43 to 45mm	MESH		**Mesh Sheets** 5.6mm Galvanised, 100 x 100mm aperture **Minimum overlap across drive is 3 squares (0.3m)** Mesh wires running across drive to be placed against the rock Install mesh to rock profile – avoid pockets behind mesh **Mesh to be installed to the face where possible** - Refer to TARP in case of variance **Mesh height above the floor** *not to exceed 3.5m* (Add extra mesh to achieve 3.5m if req.)
TIE-IN BOLT	△	**Bolt:** 0.9m x 39mm Galv Split Set Stubby Bolt **Plate:** 300 x 280mm Combination Plate **Hole Dia:** For insert into 47mm Split Set (as req'd)			

OTHER REQUIREMENTS
- Install 2 pull rings per cut in the backs
- Bolts to be between 90° to 75° to the surface being bolted
- Last row of bolts to be no greater than 0.5m from the face

Figure 11 GS2.0 Friction stabilisers plus Weld Mesh

GS2.1 HYBRID + FC

Minimum support requirements

MINIMUM QUANTITY PER CUT	
2.4m HYBRID Bolt	27
0.9m Long Stubby	
Combi Plates	27
Dome Plates	
Mesh 4.5 m	
Pull Rings	2
Fibrecrete m3	4.1

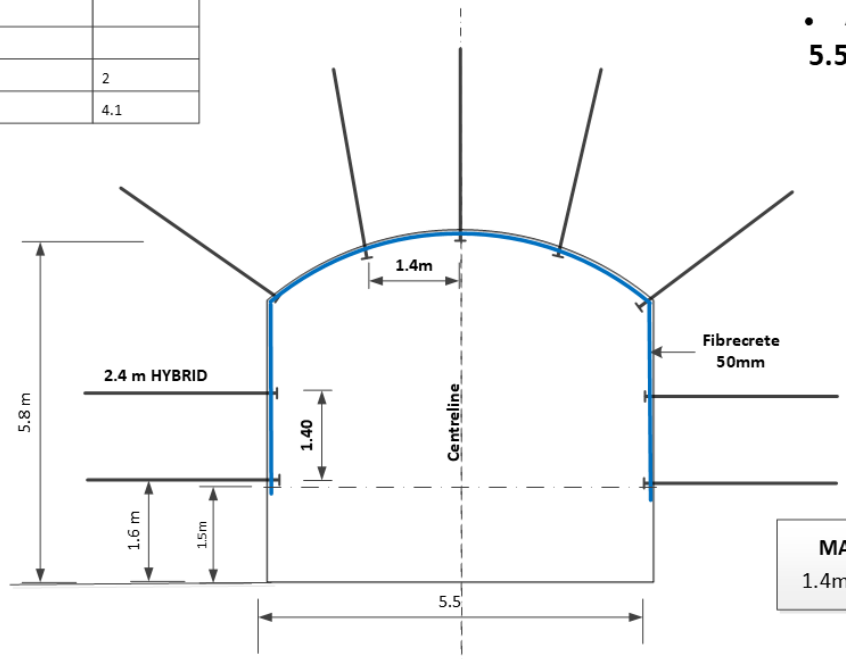
NO PERSON TO ENTER UNDER UNSUPPORTED GROUND

GS2.1

PURPOSE

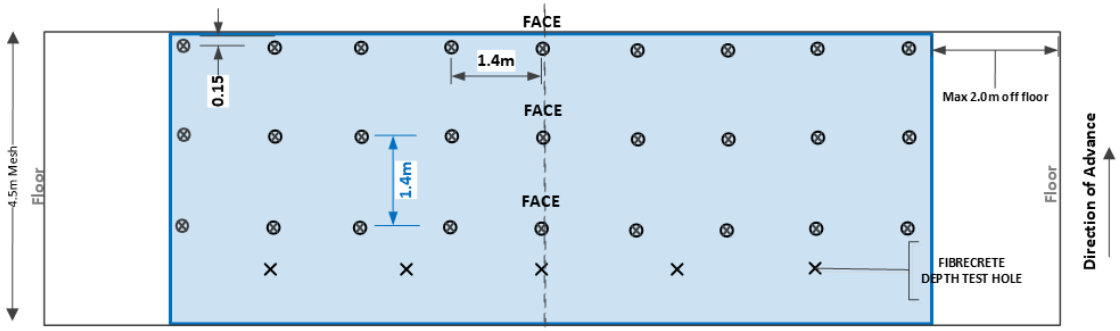
- Low demand ASEISMIC
- Good rock mass
- 4 m cut

5.5 m W x 5.8 m H



MAX. BOLTING PATTERN
1.4m Spacing x 1.4m Burden

Plan View



REINFORCEMENT			SURFACE SUPPORT		
PRIMARY BOLT	⊗	Bolt: 47mm Galv HYBRID Bolt Plate: 300 x 280mm Combination Plate Hole Dia: 43 to 45mm	Fibrecrete		50mm thickness Hydroscaling to be completed prior to application Grade to grade wall coverage and spray up to the face Depth of testing to be completed at 5 to 6 locations as indicated by a cross (X) Follow re-entry procedure

OTHER REQUIREMENTS
- Bolts to be between 90° to 75° to the surface being bolted
- Last row of bolts to be no greater than 0.5m from the face

Figure 12 GS2.1 Hybrid + Fibrecrete

GS3.0 HYBRID + M + FC

Minimum support requirements

MINIMUM QUANTITY PER CUT	
2.4m Long HYBRID Bolt	33
02.4m Long Split sets	11
Combi Plates	33
Dome Plates	11
Mesh 4.5 m	5
Pull Rings	2
Fibrecrete m3	4.1

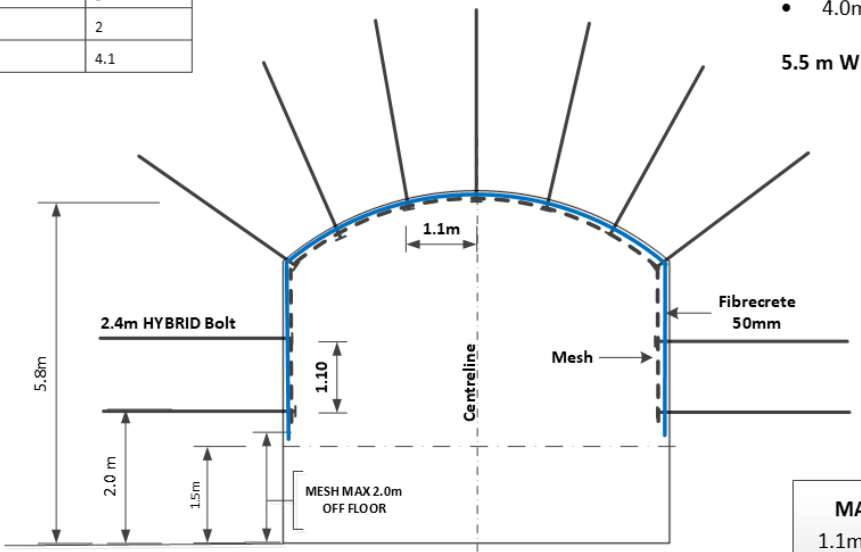
NO PERSON TO ENTER UNDER UNSUPPORTED GROUND

GS3.0

PURPOSE

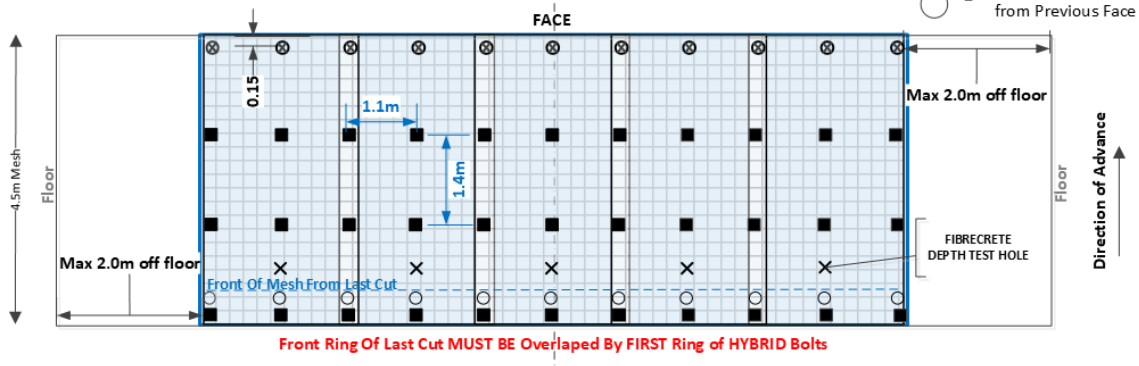
- Increasing stress damage or poor rock mass
- Mesh to be maximum of 2.0m off floor.
- 4.0m cut

5.5 m W x 5.8 m H



MAX. BOLTING PATTERN
1.1m Spacing x 1.4m Burden

Plan View



Front Ring Of Last Cut MUST BE Overlaped By FIRST Ring of HYBRID Bolts

REINFORCEMENT			SURFACE SUPPORT		
PRIMARY BOLT	■	Bolt: 47mm Galv Hybrid Bolt Plate: 300 x 280mm Combination Plate Hole Dia: 43 to 45mm	MESH	▭	Mesh Sheets 5.6mm Galvanised, 100 x 100mm aperture **Minimum overlap across drive is 3 squares (0.3m)** Mesh wires running across drive to be placed against the rock Install mesh to rock profile – avoid pockets behind mesh **Mesh to be installed to the face where possible** - Refer to TARP in case of variance **Mesh height above the floor** *not to exceed 2.0m* (Add extra mesh to achieve 2.5m if req.)
SECONDARY BOLT	⊗	Bolt: 47mm Galv Split Set Bolt Plate: 300 x 280mm Combination Plate Hole Dia: 43 to 45mm			
			FIBRECRETE	▭	50mm thickness Hydroscaling to be completed prior to application **Grade to Grade coverage and spray up to the face** **Depth of testing** to be completed at 5 to 6 locations as indicated by a cross (X) Follow re-entry procedure

OTHER REQUIREMENTS
- Bolts to be between 90° to 75° to the surface being bolted
- Last row of bolts to be no greater than 0.5m from the face
- HYBRID BOLTS ARE USED TO TIE IN NEW MESH TO PREVIOUS CUT

Figure 13 GS3.0 Hybrid + Weld Mesh + Fibrecrete

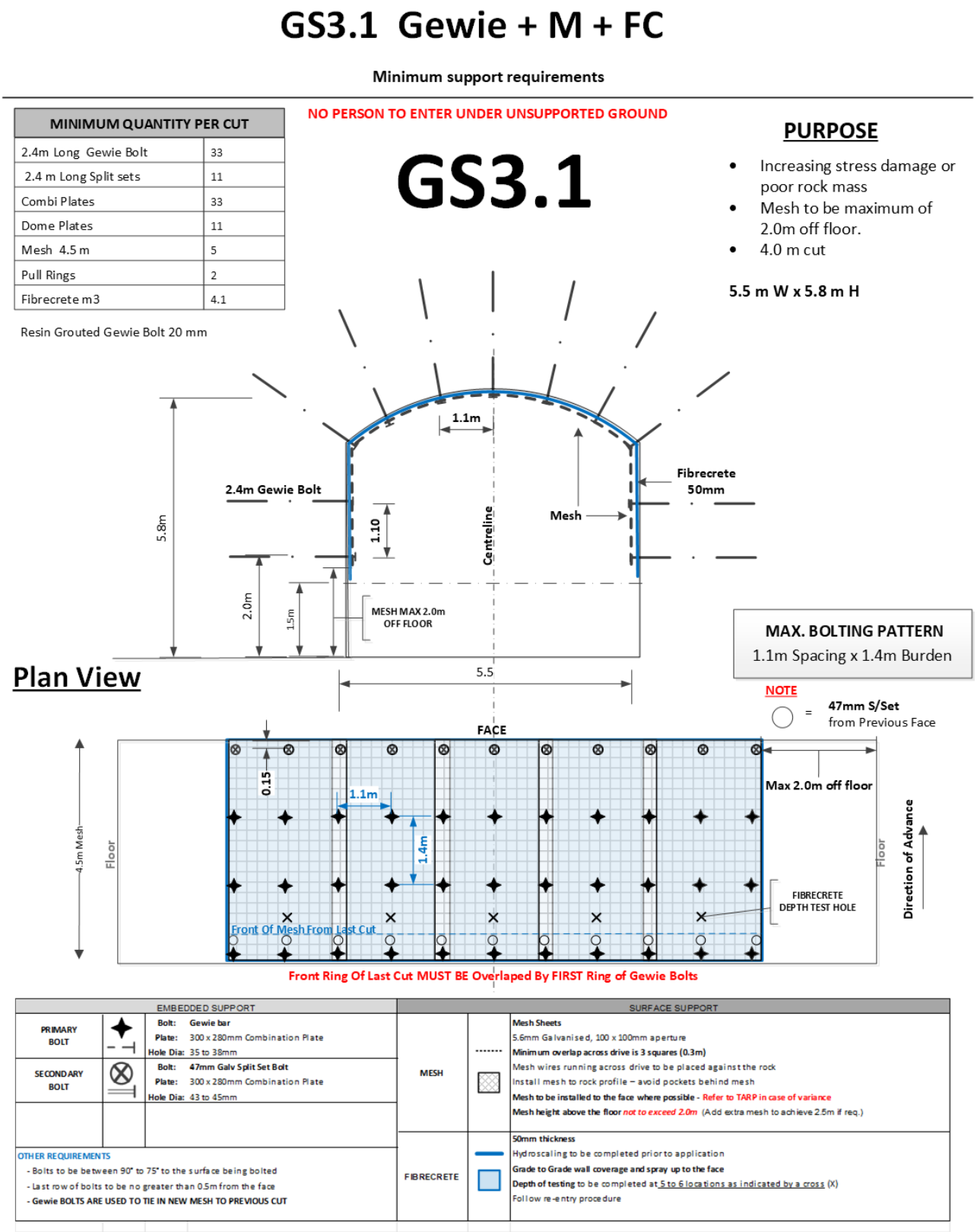


Figure 14 GS3.1 Gewie + Weld Mesh + Fibrecrete

GS4.0 Debonded Gewie + Minax + FC

Minimum support requirements

MINIMUM QUANTITY PER CUT	
2.4m Debonded Gewie Bolt	33
2.4m Long Split sets	11
Combi Plates	33
Dome Plates	11
Roll of Minax 17 m	1
Pull Rings	2
Fibrecrete m3	6.8

Debonded Gewie bolt – 20mm diameter

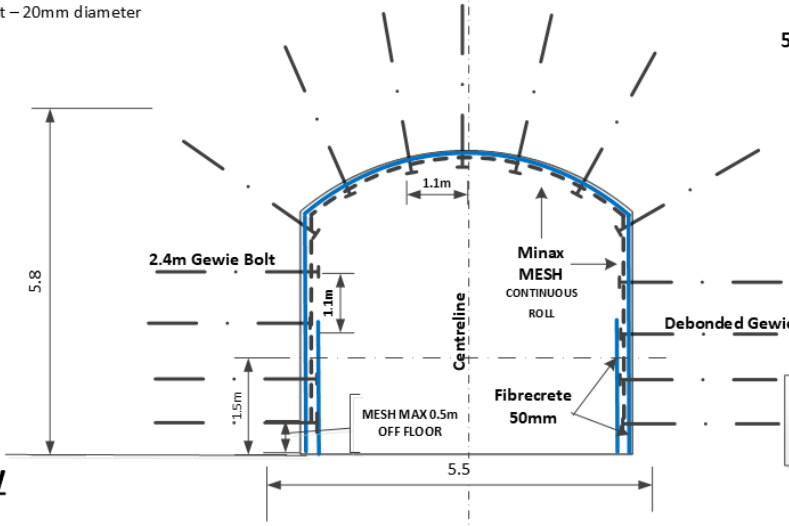
NO PERSON TO ENTER UNDER UNSUPPORTED GROUND

GS4.0

PURPOSE

- SEISMIC - Dynamic demand
- Mesh to be maximum of 0.5m off floor. Seek approval for increase of height
- PUMPED RESIN ENCAPSULTED BOLTS
- 4.0m cut

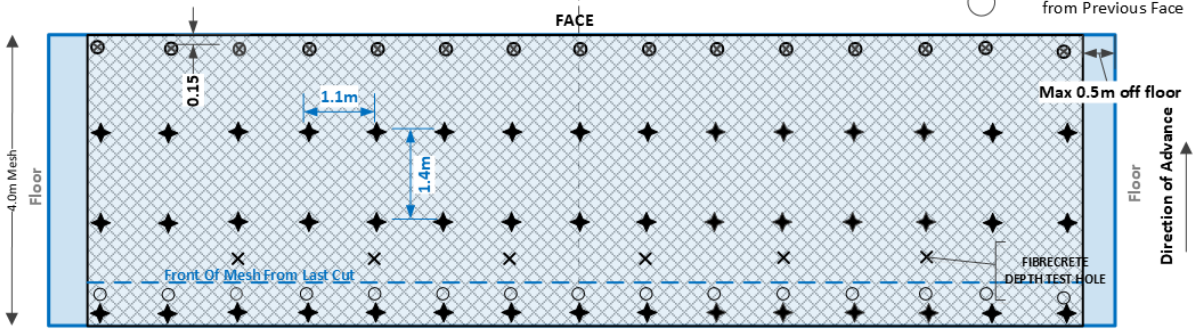
5.5 m W x 5.8 m H



MAX. BOLTING PATTERN
1.1m Spacing x 1.4m Burden

Plan View

NOTE
◯ = 47mm S/Set from Previous Face



Front Ring Of Last Cut MUST BE Overlaped By FIRST Ring of Debonded Gewie Bolts

EMBEDDED SUPPORT		
PRIMARY BOLT		Bolt: Debonded Gewie Bolt & PUMPED RESIN Plate: 300 x 280mm Combination Plate Hole Dia: 35 to 38mm
SECONDARY BOLT		Bolt: 47mm Galv Split Set Bolt Plate: 300 x 280mm Combination Plate Hole Dia: 43 to 45mm
OTHER REQUIREMENTS - Last row of bolts to be no greater than 0.5m from the face - DEBONDED GEWIE BOLTS ARE USED TO TIE IN NEW MESH TO PREVIOUS CUT		

SURFACE SUPPORT		
MESH	-------	High tensile chainlink Mesh - Roll length varies depending on drive size Product: G80/4.6 MINAX Mesh 4.6mm diameter Install mesh to rock profile – avoid pockets behind mesh Mesh to be installed to the face where possible - Refer to TARP in case of variance Mesh height above the floor *not to exceed 0.5m* ***Extra length or patches maybe required for corners, turnouts or larger development .*
FIBRECRETE		50mm thickness Hydroscaling to be completed prior to application Floor to floor wall coverage and spray up to the face Depth of testing to be completed at 6 locations as indicated by a cross (X) Follow re-entry procedure

Figure 15 GS4.0 Debonded Gewie + Minax + Fibrecrete

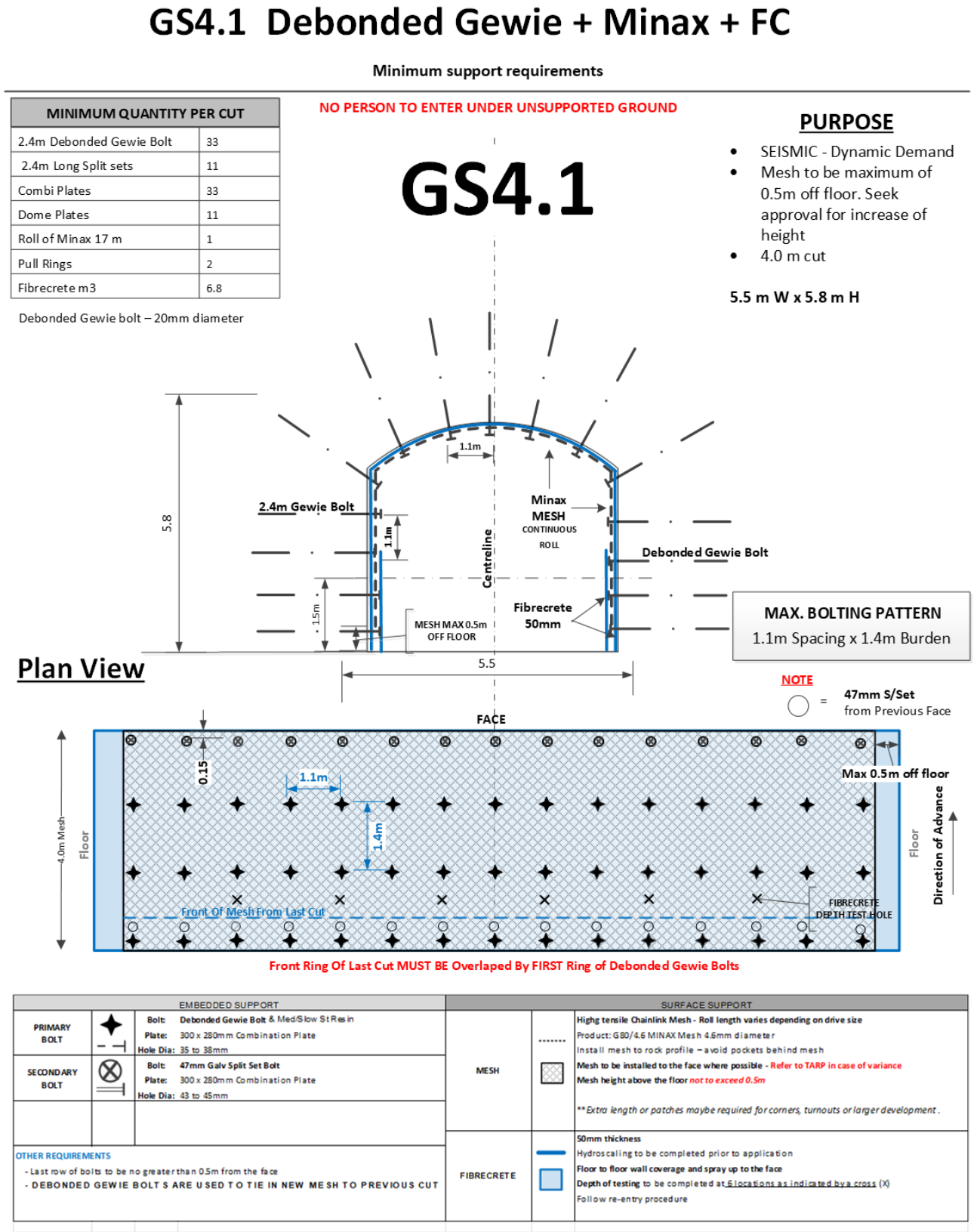


Figure 16 GS4.1 Debonded Gewie + Minax + Fibrecrete

GS4.2 Debonded Gewie + M + FC + Straps

Minimum support requirements

MINIMUM QUANTITY PER CUT	
2.4m Debonded Gewie Bolt	33
02.4m Long Splits sets	11
Combi Plates	33
Dome Plates	11
Mesh 4.5 m	7
Pull Rings	2
Fibrecrete m3	6.8
Straps	4

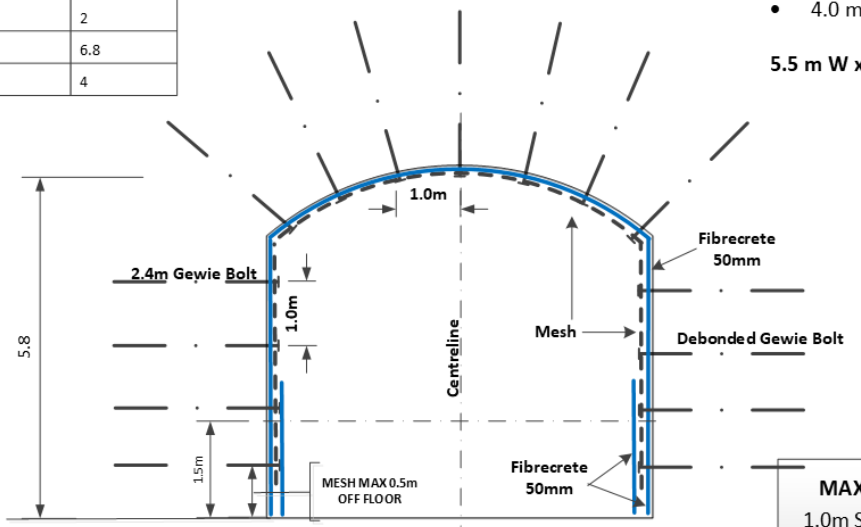
NO PERSON TO ENTER UNDER UNSUPPORTED GROUND

GS4.2

PURPOSE

- SEISMIC - Dynamic Demand
- Alternative surface support strategy to Minax Mesh
- Mesh to be maximum of 0.5 off floor.
- 4.0 m cut

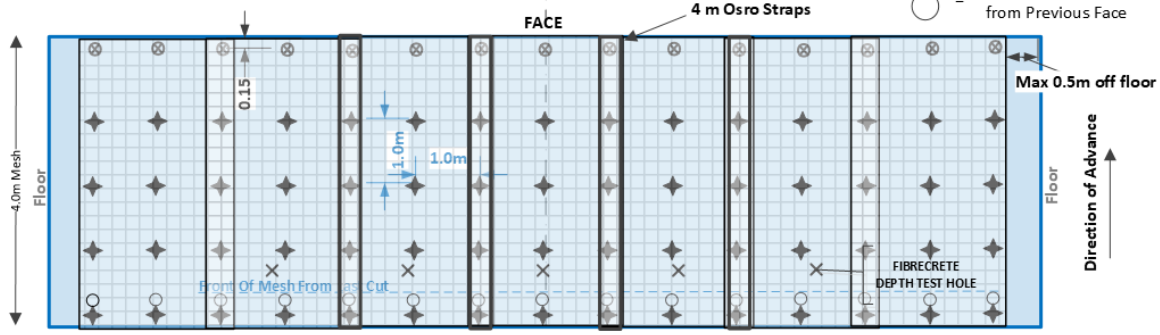
5.5 m W x 5.8 m H



MAX. BOLTING PATTERN
1.0m Spacing x 1.0m Burden

NOTE
◯ = 47mm S/Set from Previous Face

Plan View



Front Ring Of Last Cut MUST BE Overlaped By FIRST Ring of Debonded Gewie Bolts

REINFORCEMENT			SURFACE SUPPORT		
PRIMARY BOLT	✦	Bolt: Debonded Gewie & Med/Slow St Resin Plate: 300 x 280mm Combination Plate Hole Dia: 35 to 38mm	MESH	-------	Mesh Sheets 6.3mm Galvanised, 100 x 100mm aperture, 6m x 2.4m Minimum overlap across drive is 3 squares (0.3m) Mesh wires running across drive to be placed against the rock Install mesh to rock profile – avoid pockets behind mesh Mesh to be installed to the face where possible - Refer to TARP in case of variance Mesh height above the floor *not to exceed 0.5m* (Add extra mesh to achieve 0.5m if req.)
SECONDARY BOLT	⊗	Bolt: 47mm Galv Split Set Bolt Plate: 300 x 280mm Combination Plate Hole Dia: 43 to 45mm			
			FIBRECRETE	▭	50mm thickness Hydroscaling to be completed prior to application Floor to floor wall coverage and spray up to the face Depth of testing to be completed at 6 locations as indicated by a cross (X) Follow re-entry procedure

OTHER REQUIREMENTS
- Last row of bolts to be no greater than 0.5m from the face
- DEBONDED GEWIE BOLTS ARE USED TO TIE IN NEW MESH TO THE PREVIOUS CUT

Figure 17 GS4.2 Debonded Gewie + Weld Mesh + Fibrecrete + Osro straps

10 Cables + (GS4.0 or GS4.1 or GS 4.2)

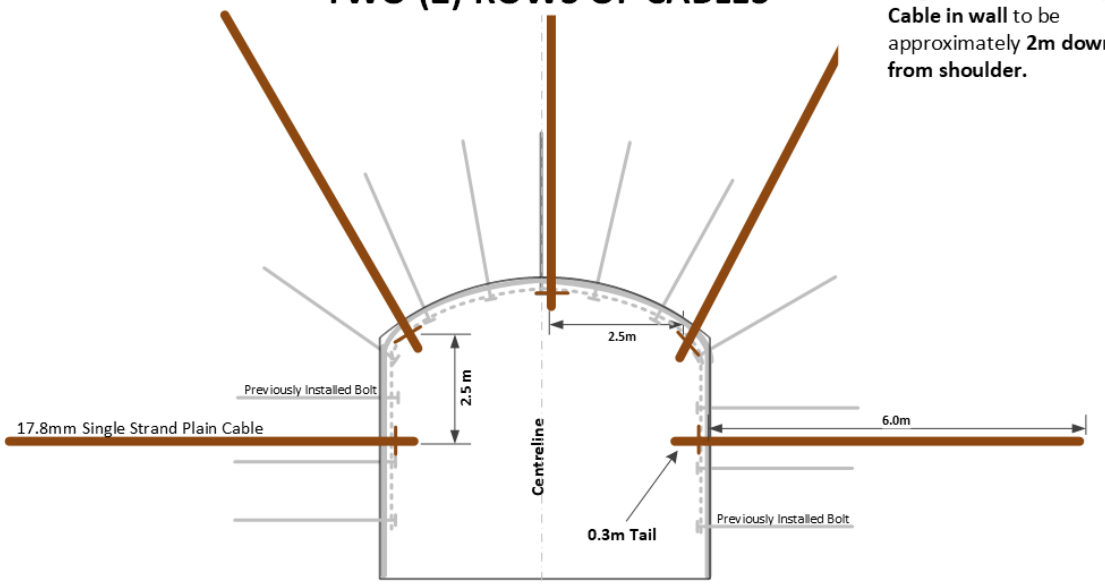
Minimum support requirements

NO PERSON TO ENTER UNDER UNSUPPORTED GROUND

Section View

GS4.x

TWO (2) ROWS OF CABLES

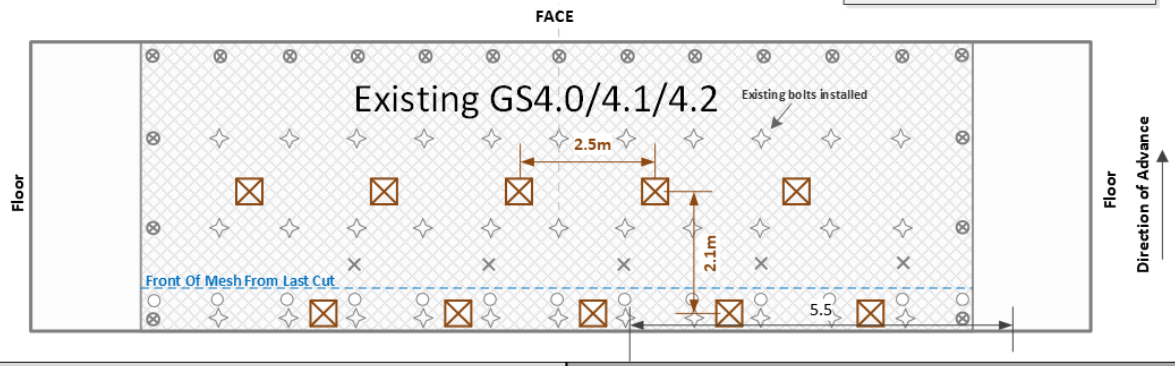


PURPOSE

- To be utilised For HEAVY dynamic ground support
- Cables to be added to GS4.0,4.2,4.2
- Height off floor will vary. **Cable in wall** to be approximately **2m down from shoulder.**

Plan View

MAX. CABLE PATTERN
2.5m Spacing x 2.1m Burden



CABLE BOLT DETAILS		INSTALLATION REQUIREMENTS
Bolt:	6.3m x 17.8mm Single Plain Strand Cable	**FIRST ROW** of bolts to be installed on **MESH OVERLAP** from previous cut
Plate:	300 x 300mm 12mm	Cable bolts must be **clean and free of grease**
Hole Dia:	64mm (nominal)	Check hole condition prioir to grouting for length and ground conditions
Hole Length:	6.0m	**Grout samples** to be collected and tested from **EACH ROW OF CABLES**
Grout	Low Heat Cement **Maximum water:cement** 0.4 : 1 ratio *(target 0.35 to 0.4 water)*	**Molybond GOG** applied **to barrel and wedge** to prevent corrosion and aid tensioning
Tail:	min. 300mm	Progressively tension to prevent damage to strands
Tension:	Tension to **100kN** (10 tonne) after min. 12 hrs curing time *CT25 jack = 4350psi or 310Bar*	**TO BE INSTALLED AFTER EVERY TWO CUTS OR AS DIRECTED**

Figure 18 GS4.2x Adds 10 single strand cables bolts to GS4.0 or GS4.1 or GS4.2

Appendix B Inputs and rates

Table 10 Manpower

Position	Shift rate (ex oncost) (AUD)	Number of personnel
Mine superintendent/foreman	1,438	1
Underground supervisor	1,096	1
Fitter: leading hand	1,080	4
Fitter: underground/breakdown	1,020	4
Fitter: workshop	960	2
Electrician: underground	1,080	4
Two boom jumbo operator	1,800	4
Two boom jumbo offsider/nipper	540	4
Loader operator LH621i	1,020	4
Truck driver	660	4
Chargeup: development/production	900	4
Service crew leading hand	780	2
Service crew	720	4
Shotcrete agitator driver	660	4
Shotcrete nozzleman	840	4

Table 11 Plant (capital, capital recovery period, operating costs, op cost units)

	Capital cost (AUD)	Capital recovery month	Operating cost (AUD)	Unit
Sandvik Jumbo DD422i	2,541,000	48	3.11	/drill metre
LH621i	2,563,000	48	214.30	/hr
TH663	2,436,000	48	136.80	/hr
Jacon MX3 Maxijet	721,900	48	75.00	/hr
Jacon TMX6	1,140,000	48	35.00	/hr
Charmec MC 605 DA(V)	1,070,000	36	55.00	/hr
Volvo L120 - underground	601,000	36	38.00	/hr
Toyota Landcruiser Ute (tray back): jumbo	136,400	24	3,000.00	/month
Toyota Landcruiser Ute: explosives with 4 door bomb box	150,400	24	3,000.00	/month
Toyota Landcruiser Ute (tray back): fitters	136,400	24	3,000.00	/month

	Capital cost (AUD)	Capital recovery month	Operating cost (AUD)	Unit
Toyota Landcruiser Ute (tray back MC): shiftboss	136,400	24	$3,000.00	c
Decline fan: Clemcorp CC1400 Mk4 (2x110kW inc. silencer)	68,420	36	$ -	
Pump, Flygt, 10 kw, HT, 1000 v	17,500	24	$ -	
Distribution board: 4-way	18,550	36	$ -	
Jumbo/pump box: jumbo	18,150	36	$ -	
1kV 5 – 37 kW pump starter	17,500	36	$ -	
Clemcorp 90-110 kW 2-stage starter	12,890	36	$ -	
6 person refuge chamber (mobile)	73,500	36	$ -	
HE explosive magazine (hire)	-	-	2,495	/month
Detonator magazine (hire)	-	-	1,250	/month
MEWP compliant work cage	60,000	36	$ -	
Grout pump and mixer and bowl [1]	11,190	36	155.00	/shift
Cable bolt tensioner kit [1]	8,925	36	100.00	/shift
Resin pump (hire) [2]	10,000			/month

Note: 1 - Only required for GS4.2 requiring cables; 2 - Only required for GS4.0

Table 12 Drilling costs

Drilling activity	Drill string size	Cost per drill metre (CPM rate) input (AUD)	Reamer cost (AUD)
Development face	45	1.10	2.30
Ground support (friction bolt)	43	1.72	
Ground support (resin bolt)	35	1.50	
Ground support (cable bolt)	64	3.83	

Table 13 Ground support scheme consumable Costs

Bolt and mesh: main component	Input cost: main component (AUD)	Unit
Friction stabiliser; 47 × 2400 mm gal	25.57	Each
Friction stabiliser; 39 × 900 mm gal	8.56	Each
Friction stabiliser; 47 × 2,400 mm gal - grouted	30.82	Each
2.4 m hybrid bolt	85.68	Each
Weld mesh 4.5 × 2.4 m gal	9.01	m^2
Mesh 3.0 × 2.4 m gal	9.01	m^2
3.0 m full resin encapsulated gewie	88.15	Each
2.4 m full resin encapsulated gewie	65.10	Each
2.4 m debonded gewie (cartridge resin encapsulation)	70.88	Each
2.4 m debonded gewie (pumped resin encapsulation)	92.40	Each
Minax mesh	25.20	m^2
Osro strap	84.00	Each
50 mm fibrecrete	400.00	m^3
6.3 m × 17.8 mm single plain strand cable	87.07	Each

Table 14 Auxiliary costs

Item	Rate (AUD)	Units
Explosive and associated	1,245	Per 4 m cut
Power	0.25	AUDkwHr
Return flights	800	Per man per roster
Accommodation and messing	180	Per man-day

Table 15 Delay times for jumbo by zone

Item	Zone 1	Zone 2	Zone 3	Zone 4	Zone 5
Non-productive time (min) for jumbo/shift	63.5	71.5	90.4	115.6	128.2
Scheduled delays per month (shifts)	7.5	7.5	7.5	7.5	7.5
Plant availability (%)	85	85	85	85	85

Table 16 Shotcrete spraying times and delays

Item	Rate	Units
Setup shotcrete	20	Minute
Hydroscale	20	Minute
Spray rate	6	m^3 per hour
Move out	20	Minute
Cure time prior to bolt	60	Minute

Table 17 Jumbo drilling and installation times

Item	Unit	Duration
Manoeuvring time per hole	Minutes	0.5
Rockbolt hole drilling rate	Drill metre per minute	1.8
47 mm friction stabiliser	Minutes	2
3.0 m/2.4 m resin fully encapsulated rockbolt	Minutes	5
Hybrid Bolt	Minutes	3
Pumped resin 2.4 m debonded rockbolt	Minutes	3
Cartridge resin 2.4 m debonded rockbolt: difficult ground	Minutes	6
Time to install each weld mesh sheet (excluding the bolting)	Minutes	5
Time to install 17 m roll of Minax mesh (excluding the bolting)	Minutes	120
Time for face drilling	Hours	2.3
Face charge time	Hours	1.22

Appendix C Cycle time breakdown

Table 18 Cycle time breakdown: single heading independent firing

Cycle activity	Unit	GS1 Gewie + M + FC	GS2.0 SS + M	GS2.1 Hybrid + FC	GS3.0 Hybrid+ M + FC	GS3.1 Gewie+ M + FC	GS4.0 debonded gewie + Minax+ FC	GS4.1 debonded gewie + Minax + FC	GS4.2 debonded gewie + M + FC + Osro	GS4.1 + 10 Cables	GS4.2 + 10 Cables	GS3.0 deep hybrid + M + FC
Cablebolting and curing	hr	–	–	–	–	–	–	–	–	19.17	19.17	–
Strap installation	hr	–	–	–	–	–	–	–	1.00	–	1.00	–
Install services (during service day)	hr	–	–	–	–	–	–	–	–	–	–	–
Shotcrete: first pass	hr	2.72	–	2.68	2.68	2.68	2.96	2.96	2.96	2.96	2.96	2.68
Shotcrete: second pass	hr	0.45	–	–	–	–	0.18	0.18	0.18	0.18	0.18	–
Rebog (completed during shotcrete cure)	hr	0.48	–	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64
Rockbolting and meshing	hr	5.41	3.10	2.32	3.43	5.00	3.15	6.01	8.55	6.01	6.59	3.43
Rebog/cleanup after bolting	hr	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67
Bore face	hr	1.84	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17
Charge	hr	1.02	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18
Fire and re-entry	hr	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83
Tramming time to next face	hr	–	–	–	–	–	–	–	–	–	–	–
Jumbo standdown time	hr	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Bogging	hr	1.30	2.26	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62
Independent firing total cycle time per cut	hr	15.22	10.70	12.60	13.71	15.28	13.89	16.75	20.29	35.92	37.50	13.71
Total cycle time per metre	hr	5.1	2.7	3.1	3.4	3.8	3.5	4.2	5.1	9.0	9.4	3.4
Average hours available per month	hr	488	482	482	468	468	450	450	450	440	440	450
Cuts per month		32	45	38	34	31	32	27	22	12	12	33
Metres per day	m	3.20	5.97	5.08	4.52	4.05	4.29	3.56	2.93	1.65	1.55	4.35
Cuts per day		1.07	1.49	1.27	1.13	1.01	1.07	0.89	0.73	0.41	0.39	1.09
Metres per month	m	97	181	154	137	123	130	108	89	50	47	132

Table 19 Cycle time breakdown: single heading end of shift firing

Cycle activity	Unit	GS1 Gewie + M + FC	GS2.0 SS + M	GS2.1 Hybrid + FC	GS3.0 Hybrid+ M + FC	GS3.1 Gewie+ M + FC	GS4.0 debonded gewie + Minax+ FC	GS4.1 debonded gewie + Minax + FC	GS4.2 debonded gewie + M + FC + Osro	GS4.1 + 10 Cables	GS4.2 + 10 Cables	GS3.0 deep hybrid + M + FC
Cablebolting and curing	hr	–	–	–	–	–	–	–	–	19.17	19.17	–
Strap installation	hr	–	–	–	–	–	–	–	1.00	–	1.00	
Install services (during service day)	hr	–	–	–	–	–	–	–	–	–	–	–
Shotcrete: first pass	hr	2.72	–	2.68	2.68	2.68	2.96	2.96	2.96	2.96	2.96	2.68
Shotcrete: second pass	hr	0.45	–	–	–	–	0.18	0.18	0.18	0.18	0.18	–
Rebog (completed during shotcrete cure)	hr	0.48	–	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64	0.64
Rockbolting and meshing	hr	5.41	3.10	2.32	3.43	5.00	3.15	6.01	8.55	6.01	6.59	3.43
Rebog/cleanup after bolting	hr	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67
Bore face	hr	1.84	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17
Charge	hr	1.02	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18	1.18
Fire and re-entry	hr	3.83	3.83	3.83	3.83	3.83	3.83	3.83	3.83	3.83	3.83	3.83
Tramming time to next face	hr	–	–	–	–	–	–	–	–	–	–	–
Jumbo standdown time	hr	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Bogging	hr	1.30	2.26	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62	1.62
Independent firing total cycle time per cut	hr	18.22	13.70	15.60	16.71	18.28	16.89	19.75	23.29	38.92	40.50	16.71
Total cycle time per metre	hr	6.1	3.4	3.9	4.2	4.6	4.2	4.9	5.8	9.7	10.1	4.2
Average hours available per month	hr	488	482	482	468	468	450	450	450	440	440	450
Cuts per month		27	35	31	28	26	27	23	19	11	11	27
Metres per day	m	2.67	4.65	4.09	3.73	3.40	3.53	3.03	2.57	1.52	1.45	3.56
Cuts per day		0.89	1.16	1.02	0.93	0.85	0.88	0.76	0.64	0.38	0.36	0.89
Metres per month	m	81	141	124	113	103	107	92	78	46	44	108

Table 20 Cycle time breakdown: multi-heading end of shift firing

Cycle activity	Unit	GS1 Gewie + M + FC	GS2.0 SS + M	GS2.1 Hybrid + FC	GS3.0 Hybrid+ M + FC	GS3.1 Gewie+ M + FC	GS4.0 debonded gewie + Minax+ FC	GS4.1 debonded gewie + Minax + FC	GS4.2 debonded gewie + M + FC + Osro	GS4.1 + 10 Cables	GS4.2 + 10 Cables	GS3.0 deep Hybrid + M + FC
Cablebolting and curing	hr	–	–	–	–	–	–	–	–	2.41	2.41	–
Strap installation	hr	–	–	–	–	–	–	–	1.00	–	1.00	–
Install services (during service day)	hr	–	–	–	–	–	–	–	–	–	–	–
Shotcrete: first pass	hr	–	–	–	–	–	–	–	–	–	–	–
Shotcrete: second pass	hr	–	–	–	–	–	–	–	–	–	–	–
Rebog (completed during shotcrete cure)	hr	–	–	–	–	–	–	–	–	–	–	–
Rockbolting and meshing	hr	5.41	3.10	2.32	3.43	5.00	3.15	6.01	8.55	6.01	6.59	3.43
Rebog/cleanup after bolting	hr	–	–	–	–	–	–	–	–	–	–	–
Bore face	hr	1.84	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17	2.17
Charge	hr	–	–	–	–	–	–	–	–	–	–	–
Fire and re-entry	hr	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83	0.83
Tramming time to next face	hr	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Jumbo standdown time	hr	–	–	–	–	–	–	–	–	–	–	–
Bogging	hr	–	–	–	–	–	–	–	–	–	–	–
Independent firing total cycle time per cut	hr	8.33	6.35	5.57	6.68	8.25	6.40	9.26	12.80	11.67	13.25	6.68
Total cycle time per metre	hr	2.8	1.6	1.4	1.7	2.1	1.6	2.3	3.2	2.9	3.3	1.7
Average hours available per month	hr	488	482	482	468	468	450	450	450	440	440	450
Cuts per month		59	76	87	70	57	70	49	35	38	33	67
Metres per day	m	5.80	10.02	11.44	9.26	7.52	9.26	6.43	4.65	4.98	4.38	8.90
Cuts per day		1.93	2.51	2.86	2.32	1.88	2.32	1.61	1.16	1.24	1.10	2.23
Metres per month	m	176	304	347	281	228	281	195	141	151	133	270

References

Arcaro, C, Villaescusa, E, Hassell, R, Talebi, R & Kusui, A 2021, 'Implementation of high energy dissipation ground support scheme'. *Underground Operators Conference 2021*, Australasian Institute of Mining and Metallurgy, Melbourne, pp. 122–134.

Askaripour, M, Saeudu , A, Rouleau A & Mercier-Langevin P, 2022, 'Rockburst in underground excavations: A review of the mechanisms, classification, and prediction methods', *Underground Space*, vol. 7, pp. 577–607, https://doi.org/10.1016/j.undsp.2021.11.008

Bernard, E & Thomas, A 2020, 'Fibre reinforced sprayed concrete for ground support', *Tunnelling and Underground Space Technology*, vol. 99, no. 103302, https://doi.org/10.1016/j.tust.2020.103302

Brändle, R & Bucher, R 2023, 'Large-scale tests of weld mesh versus high-tensile chain link mesh on mesh overlap load transfer', in J Wesseloo (ed.), *Ground Support 2023: Proceedings of the 10th International Conference on Ground Support in Mining*, Australian Centre for Geomechanics, Perth, pp. 399–410, https://doi.org/10.36487/ACG_repo/2325_26

Brändle, R, Fonseca, R, von Rickenback, G, Fischer, G, Vallejos, J, Marambio, E & Burgos, L 2022, 'Single impact dynamic testing of a ground support system with high-tensile steel wire mesh at the Walenstady testing facility', *RaSIM 10*, Society for Mining, Metallurgy & Exploration, Tucson.

Cuello, D, Orrego, C, Vallejos, J & Velásquez, J 2024, 'The role of surface support in controlling rockbursting conditions', in D Johansson & H Schunnesson (eds), *MassMin 2024: Proceedings of the 9th International Conference & Exhibition on Mass Mining*, Luleå University of Technology, Luleå, pp. 948–964, https://doi.org/10.36487/ACG_repo/2435_G-07

Davidson, G 2005, 'Spraying the surface or just scratching the surface – what are the real benefits of in-cycle fibrecrete,' *9th AusIMM Underground Operators Conference*, Australasian Institute of Mining and Metallurgy, Melbourne.

Dehkhoda, S, Reusch, F & Putzar, G 2023, 'A numerical approach for a displacement-based ground support capacity consumption forecast', in J Wesseloo (ed.), *Ground Support 2023: Proceedings of the 10th International Conference on Ground Support in Mining*, Australian Centre for Geomechanics, Perth, pp. 573–590, https://doi.org/10.36487/ACG_repo/2325_39

Diederichs, M 2018, 'Early assessment of dynamic rupture hazard for rock burst risk mitigation in deep tunnel projects', *Journal of the Southern African Institute of Mining and Metallurgy*, vol. 118, pp. 193–204, http://dx.doi.org/10.17159/2411-9717/2018/v118n3a1

Drover, C, Villaescusa, E & Onederra, I 2022 'Design, implementation and field performance of a face destress blasting method for mine development', *Ausrock 2022*, Australasian Institute of Mining and Metallurgy, Melbourne, pp. 60–72.

Dunn, MJ & de Graaf, PHJ 2023, 'Suggested framework for design effort and acceptance criteria for underground mine excavations', in J Wesseloo (ed.), *Ground Support 2023: Proceedings of the 10th International Conference on Ground Support in Mining*, Australian Centre for Geomechanics, Perth, pp. 535–546, https://doi.org/10.36487/ACG_repo/2325_36

Dunn, MJ & Parrott, TT 2023, 'Mine portal design: considerations, methods and practices', in J Wesseloo (ed.), *Ground Support 2023: Proceedings of the 10th International Conference on Ground Support in Mining*, Australian Centre for Geomechanics, Perth, pp. 237–254, https://doi.org/10.36487/ACG_repo/2325_16

Esterhuizen, G 2026, *JBlock*, https://sites.google.com/view/ayzprojects

Ghazvinian, E, Orrego, C & Fuenzalida, M 2023, 'Numerical forecast of central access ground support behaviour at Cadia East PC1-2', in J Wesseloo (ed.), *Ground Support 2023: Proceedings of the 10th International Conference on Ground Support in Mining*, Australian Centre for Geomechanics, Perth, pp. 521–534, https://doi.org/10.36487/ACG_repo/2325_35

He, M, Canbulat, I, Suorineni, F, Karakus, M, Nie, W, Liu, D, Zhang, C, Nagibin, A & Rustembek B 2025, 'Definition and classification of rockburst', *Rock Mechanics Bulletin*, vol. 4, issue 3, https://doi.org/10.1016/j.rockmb.2025.100206

Heal, D 2010, *Observations and Analysis of Incidences of Rock burst damage in Underground Mines*, PhD thesis, The University of Western Australia, Perth.

Hoek, E 2006, *Practical Rock Engineering*, https://static.rocscience.cloud/assets/resources/learning/hoek/Practical-Rock-Engineering-Full-Text.pdf

Hudson, J & Harrison, J 1997, 'Underground excavation instability mechanisms', *Engineering Rock Mechanics – An Introduction to the Principles*, Elsevier.

Hutchins, WR, Bywater, S, Thompson, AG & Windsor, CR 1990, 'A versatile grouted cable dowel reinforcing system for rock', *The AusIMM Proceedings 1990*, Australasian Institute of Mining and Metallurgy, Melbourne, pp. 25–29.

Kaiser, P, Diederichs, M, Martin, C, Sharp, J & Steiner, W 2000, 'Underground works in hard rock tunnelling and mining', *Proceedings of the GeoEng2000 Conference*, Technomic Publishing Company, Lancaster, pp. 841–926.

Lang, A & Stubley, C 2004, 'Rockfalls in Western Australian underground metalliferous mines' in E Villaescusa & Y Potvin (eds), *Proceedings of the 5th International Symposium on Ground Support in Mining and Underground Construction,* Taylor & Francis, pp. 697–706.

Li, C 2017, 'Design principals for underground rock support', *3rd Nordic Rock Mechanics Symposium*, Finnish Association of Civil Engineers RIL, Helsinki, pp 8–18.

Lorig, LJ & Varona, P 2013, 'Guidelines for numerical modelling of rock support for mines', in Y Potvin & B Brady (eds), *Ground Support 2013: Proceedings of the Seventh International Symposium on Ground Support in Mining and Underground Construction*, Australian Centre for Geomechanics, Perth, pp. 81–105, https://doi.org/10.36487/ACG_rep/1304_04_Lorig

Mbenza, J, Potvin, Y & Wesseloo, J 2024, 'Fibre reinforced shotcrete strategy for optimisation of ground support efficacy', in D Johansson & H Schunnesson (eds), *MassMin 2024: Proceedings of the 9th International Conference & Exhibition on Mass Mining*, Luleå University of Technology, Luleå, pp. 1534–1543, https://doi.org/10.36487/ACG_repo/2435_R-02

Morton, E 2009, Static Testing of Large Scale Group Support Panels, Masters thesis, Western Australia School of Mines, Curtin University of Technology, Perth, https://curate.curtin.edu.au/articles/thesis/Static_testing_of_large_scale_ground_support_panels/31460398

MOSHAB 1999, *Surface Rock Support for Underground Mines Code of Practice*, https://www.worksafe.wa.gov.au/system/files/documents/2025-02/MSH_COP_SurfaceRockSupportUGMines.pdf

NGI 2025, *NGI Using the Q-System, Rock Mass Classification and Support Design*, https://www.ngi.no/globalassets/bilder/forskning-og-radgivning/bygg-og-anlegg/q-handbook_2025_english_final.pdf

Owen, ML 2009, 'Workforce exposure to rockfalls in underground mines', in PM Dight (ed.), *SRDM 2009: Proceedings of the First International Seminar on Safe and Rapid Development Mining*, Australian Centre for Geomechanics, Perth, pp. 33–46, https://doi.org/10.36487/ACG_repo/902_04

Player, J 2004, *Rock Mass Damage and Excavation Response – Big Bell Longitudinal Sub Level Caving Operation*, Masters thesis, Curtin University of Technology, Perth.

Player, J, Morton E, Thompson, A & Villaescusa, E 2008, 'Static and dynamic testing of steel wire mesh for mining applications for rock surface support', *6th International Symposium on Ground Support in Mining and Civil Engineering Construction,* Southern African Institute of Mining and Metallurgy, Johannesburg, pp. 693–706.

Reardon, D, Nester, T, Barsanti, B & Page, A 2022, 'Friction bolt performance over the past ten years in Australian mines', *Ausrock Conference 2022*, Australasian Institute of Mining and Metallurgy, Melbourne, pp. 410-427.

Rocscience 2026, *RocTunnel3*, computer software, https://www.rocscience.com/help/roctunnel3/overview/technical-specifications

Roberts, T, Talu, S & Wang, Y 2018, 'Design and testing of high capacity surface support', in *Proceedings of the Fourth Australasian Ground Control in Mining Conference*, Australasian Institute of Mining and Metallurgy, Melbourne, pp. 303–322.

Sandy, M & Player, J 1999, 'Reinforcement design investigations at Big Bell', *Proceedings of the International Symposium of Ground Support*, Balkema, pp. 301–316.

Sandy, MP, Gibson, W & Gaudreau, D 2007, 'Canadian and Australian ground support practices in high deformation environments', in Y Potvin (ed.), *Deep Mining 2007: Proceedings of the Fourth International Seminar on Deep and High Stress Mining*, Australian Centre for Geomechanics, Perth, pp. 297–311, https://doi.org/10.36487/ACG_repo/711_22

Stacey, T 2003, 'Rock engineering – good design or good judgement', *Journal of the Southern African Institute of Mining and Metallurgy,* pp. 411–422, https://www.saimm.co.za/Journal/v103n07p411.pdf

Stacey, T 2009, 'Design – a strategic issue', *Journal of the Southern African Institute of Mining and Metallurgy*, vol. 109, pp. 157–162, https://scielo.org.za/pdf/jsaimm/v109n3/02.pdf

Thompson, A & Windsor, C 1992, 'A classification system for reinforcement and its use in design', *Proceedings of the Western Australian Conference on Mining Geomechanics*, Western Australian School of Mines, Perth, pp. 115–125.

Thompson, A, Windsor, C & Villaescusa, E 2012, 'Ground support terminology and classification: an update', *Geotechnical and Geological Engineering*, vol. 30, pp. 553–580, https://doi.org/10.1007/s10706-012-9495-4

Vakili, A, Sandy, M, Mathews, M & Rodda, B 2013, 'Ground support design under highly stressed conditions', in Y Potvin & B Brady (eds), *Ground Support 2013: Proceedings of the Seventh International Symposium on Ground Support in Mining and Underground Construction*, Australian Centre for Geomechanics, Perth, pp. 551–564, https://doi.org/10.36487/ACG_rep/1304_38_Vakili

Varden, R, Hassell, R & Player, J 2023, 'Friction rock stabilisers QAQC and a Process to Investigate High Pull Test Failure Rates', *UGOPS 2023*, Australasian Institute of Mining and Metallurgy, Melbourne, pp. 94–109.

Vazaios, I, Diederich, M & Vlachopoulos, N 2019, 'Assessment of strain bursting in deep tunnelling by using the finite-discrete element method', *Journal of Rock Mechanics and Geotechnical Engineering*, vol. 11, pp. 12–37, https://doi.org/10.1016/j.jrmge.2018.06.007

Villaescusa, E, Thompson, AG & Player, JR 2013a, 'A decade of ground support research at the WA School of Mines', in Y Potvin & B Brady (eds), *Ground Support 2013: Proceedings of the Seventh International Symposium on Ground Support in Mining and Underground Construction*, Australian Centre for Geomechanics, Perth, pp. 233–245, https://doi.org/10.36487/ACG_rep/1304_15_Villaescusa

Villaescusa, E, Thompson, AG & Player, JR 2013b, 'Static and dynamic testing of welded and woven mesh for rock support', in Y Potvin & B Brady (eds), *Ground Support 2013: Proceedings of the Seventh International Symposium on Ground Support in Mining and Underground Construction*, Australian Centre for Geomechanics, Perth, pp. 187–196, https://doi.org/10.36487/ACG_rep/1304_11_Villaescusa

Villaescusa, E, Thompson, A & Windsor, C 2018, ;Probabilistic estimate of rock mass static and dynamic demands for underground excavation stabilization', *Journal of Rock Mechanics and Geotechnical Engineering*, vol. 11, issue 3, https://doi.org/10.1016/j.jrmge.2018.08.009

Villaescusa, E, Thompson, A, Windsor, C & Player, J 2023, *Ground Support Technology for High Stressed Excavations*, CRC Press.

Wood, D 1999, 'Estimating volume of shotcrete for mining application', *Shotcrete Magazine*, pp. 12–16.