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Evolutions of dynamic support design methodology: site perspectives

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Abstract

The subject of dynamic support design has seen various methods applied across different mining operations over the years. This paper summarises dynamic support design approaches applied at 3 different West Australian mine sites over the last decade. The first 2 design cases will discuss the progression from design methods outlined in the Canadian Rockburst Support Handbook (Kaiser et al. 1996), through to the application of updates proposed by Scott et al. (2008) and Potvin & Wesseloo (2013). The final design case will outline a method used to apply design concepts from deformation-based support design (Kaiser & Moss 2022) to a site-based design methodology aiming to simplify the assessment of dynamic support demand in an extreme demand case.

Keywords: *support design, dynamic support, deformation-based support design, seismicity*

1 Introduction

Appropriate ground support selection and, by proxy, system capacity forms one part of the control suite utilised to maintain excavation stability in the underground mining environment. As mines in Australia, including Western Australia, have progressed deeper and extracted larger portions of their orebodies, the dynamic demand induced by increasing levels of near mine seismic activity increases. This is legally recognised in the most recent iterations of the mining regulations, specifically r. 628 of the Work Health and Safety (Mines) Regulations 2022 which states the following should be considered:

> *"(i) design, installation (including time of installation), monitoring, maintenance, longevity and quality control of ground support and reinforcement;*
>
> *(j) managing and minimising adverse effects of blasting, old workings and dynamic loading;"* (Government of Western Australia 2022).

Given the specific legal requirement to manage and minimise the adverse effect of dynamic loading, design in these conditions requires the use of an assessment method that accounts for not just the static weight of the rock mass, but the additional demand resulting from movement of the rock mass. This paper will explore dynamic support designs in 3 different Western Australian mines between 2015 and 2025 with a focus on:

- the methods selected and applied to each site's design process
- input parameters selected for each design
- site-specific adaptions or updates made to each design process
- the design outcomes.

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2 Background

While a full literature review is not the intent of this paper, the origins of many ground support design methods trace back to methods outlined in the *Canadian Rockburst Support Handbook* (CRBSHB) (Kaiser et al. 1996). A more recent Australian application of this method that is widely referenced in multiple subsequent case studies and published works is shown in Figure 1.

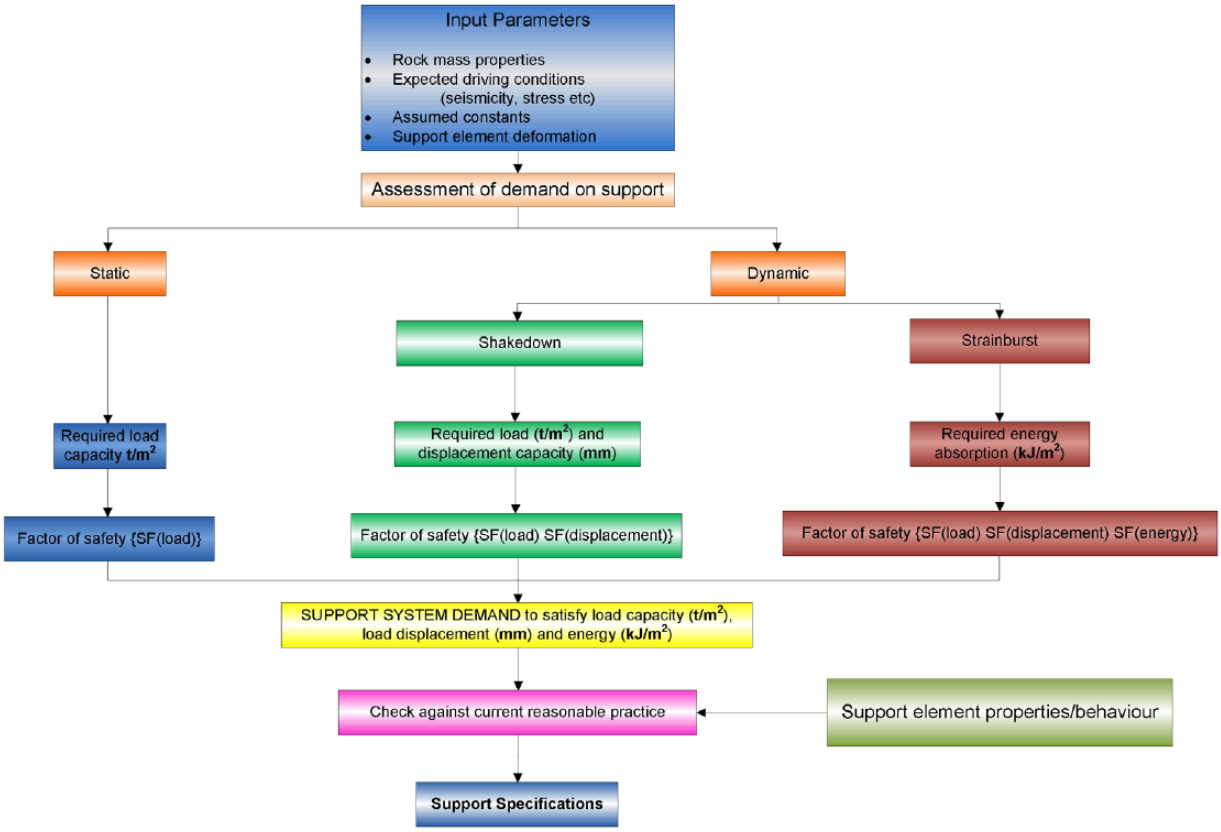


Figure 1 Support selection procedure chart (Scott et al. 2008)

The wider CRBSHB method has also since been described and, in some cases, amended in various bodies of work such as those published by Potvin & Wesseloo (2013) and Louchnikov & Sandy (2017). Arguably, the most pertinent review of this method, however, came from one of the original authors in 2013 (Kaiser & Cai 2013). This review published 3 factual conclusions summarised in the abstract:

> *"(1) standard, assumed ground motion patterns are unacceptably flawed; (2) ground motion-related energy transfer mechanisms rarely constitute the main source of support loading; and (3) dynamically stressed rock fails by brittle, extensional failure causing disproportional excavation convergences and support damaging deformations."*

This body of work (Kaiser & Cai 2013) openly questioned certain foundations of the original CRBSHB work and was followed up shortly after with the initial publication of the deformation-based support design methodology (DBSD) concepts (Kaiser 2014). In response to the 3 facts stated above regarding the CRBSHB, this initial publication on DBSD proposed that "As a consequence, support needs to cope with large, localised deformations and thus all support components in a support system must satisfy a deformation-based design criterion".

This publication has since been followed by multiple publications on the topic including those by Kaiser (2016), Cai & Kaiser (2018), Kaiser & Moss (2022), and Kaiser & Malovichko (2022), among others.

The case studies reviewed in this paper follow this progression of support design methodology using the CRBSHB method (Kaiser et al. 1996) outlined by Scott et al. (2008) in Mine A. Mine B uses this method with

adaptions proposed by Potvin & Wesseloo (2013). Finally, Mine C applies an approach to utilising the DBSD concept in a manner suited to rapid site reassessment and application.

3 Mine A: Kaiser method

Mine A consisted of 2 longhole open stope (LHOS) mines operating below 1,000 m depth ('A-1' and 'A-2'), with a recent history of multiple seismic related falls of ground, as outlined in Table 1. While exact production impacts are omitted from this paper, the failures below had resulted in various degrees of equipment damage, lost production and, in isolated cases, personnel injury.

Table 1 Dynamic support related failures at Mine A leading up to dynamic support design

Mine	Occurrence	Mass	Date
A-2	Rockburst – blocky ground	>10 t	October – Year 3
A-2	Rockburst – blocky ground	>10 t	October – Year 3
A-2	Seismic failure – bedded ground	<5 t	October – Year 3
A-2	Rockburst – blocky ground	<5 t	September – Year 3
A-2	Rockburst – blocky ground	>50 t	September – Year 3
A-1	Rockburst – blocky ground	<1 t	January – Year 3
A-1	Rockburst – unknown	<1 t	December – Year 2
A-1	Rockburst – bedded ground	<1 t	June – Year 2
A-1	Seismic failure – intrusive unit	>50 t	January – Year 2
A-2	Seismic failure – intrusive unit	–	June – Year 1

3.1 Method

The design methodology employed in Mine A directly reproduced that published by Scott et al. (2008) which builds upon the CRBSHB method published by Kaiser et al. (1996). While calculation of the dynamic demand at Mine A was not in itself unusual, the domaining and identification of higher hazard areas was unique in the author's experience. The incidences of seismic damage to development were in almost all cases associated with mine development intersecting certain volcanic intrusive units at acute angles. It was hypothesised that a strength and stiffness differential between these units and the surrounding host rock contributed to these failures by causing localised stress concentrations in certain lithologies. A series of site-specific design rules were developed to be applied at a mine design stage to reduce instances of development that would create this hazard:

1. Acute angles formed by cross cutting intrusive units at angles <30° will require secondary support and will be assessed on a case-by-case basis. When this occurs with potential seismic sources, these conditions will require face support and face de-stress procedures.
2. Four-way intersections should be avoided where possible. If unavoidable, they must be placed wholly within a single domain, not on intrusive/domain margins.
3. Avoid placing 3-way intersections on the margin of intrusives.
4. If developing parallel to thin (<3 m) rhyolite, it is preferable to position the drive such that the rhyolite runs parallel along the middle of the backs. If development enters/exits a rhyolite to achieve this, secondary support may be required.
5. In wide intrusives, it is preferential to develop parallel through the centre of the unit maintaining at least a 5 m stand-off from the contacts.

The application of these rules surpassed most other design considerations and, in many cases, resulted in irregular decline loop design, curved oredrive geometry and unfavourable capital placement. To develop appropriate confidence in intrusive position, specific diamond drill programs were implemented along with localised wireframe updates to site lithology models. This approach was aimed at eliminating or heavily engineering controls to the seismic hazard, with ground support reviewed as a secondary control measure only after other options had been attempted. While highly effective, it is the author's opinion this method only functioned because of the uniquely interdependent relationship between the geology, mine planning and geotechnical teams at Mine A that is rarely replicated to this degree.

3.2 Input parameters

Numerous papers have been published on the selection of support design input parameters, their recommended values, or those chosen for specific case studies. As such, each input parameter will not be discussed in detail, but the key parameters chosen for Mine A and Mine B are presented in Table 2. Parameters that warrant further discussion for Mine A are discussed in Sections 3.2.1–3.2.3.

Table 2 Mine A and Mine B dynamic support design input parameter ranges

Parameter	Mine A	Mine B	Description
c*	0.25 m^2/s	0.25 m^2/s	Constant between 0.2–0.3 in peak particle velocity calculation
α	–	0.53	Constant for R_0 calculation between 0.53–1.14
R	12, 16.5, 21	10	Distance from source
M_{MAX}	1.6	1.5	Magnitude (local)
n	4	4	Incidence Angle
ρ	2,800–3,500 kg/m^3	2,700 kg/m^3	Density
C_s	3,808 m/s	3,440 m/s	Shear wave velocity
W	5 m, 5.5 m	5.5 m	Drive width
UCS	103–200 MPa	115 MPa	Uniaxial compressive strength
L	5 m, 5.5 m	5.5 m	Potential failure length
η	0.1	0.1	Energy efficiency
FOS	1.5	1.5	Factor of Safety
σ_{max}	140–150 MPa	85 MPa	Maximum tangential stress
b_f	0.05	0.05	Bulking factor
l	4.2 m	4 m	Cut length
M_d	1.25	1.25	Support element multiplier
n	1	1	Ejection velocity ratio
C	0.81–1.34	0.65	Stress fracture constant
S	0.25	1	Strainburst surface factor
d_{ult}	0.15 m	0.1 m	Ultimate deformation capacity (support)

3.2.1 Distance from source

As discussed previously, the largest single influencing factor that led to damaging events at Mine A was the acute intersection of certain lithologies. This led to variable distance to source selection, based on relative position to these intersecting lithologies. As shown in Figure 2, Mine A assigned 3 support demand categories, 'High', 'Medium' and 'Low' by seismic distance to source. The distances in Figure 2 were selected based on a detailed review of seismic event distance to source, as well as the failures included in Table 1.

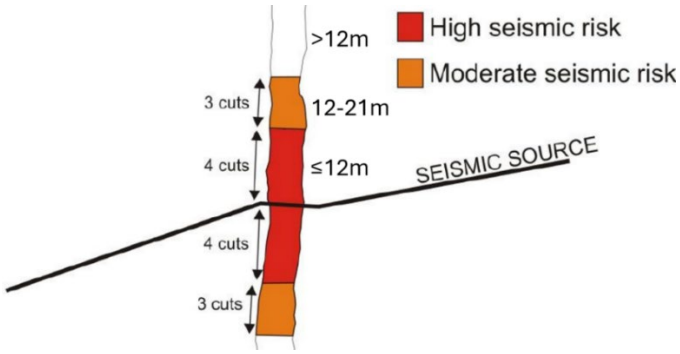


Figure 2 Mine A seismic risk category by distance to source

3.2.2 Magnitude

Selection of a design magnitude for Mine A used Equation 1, at the time attributed to Kijko & Funk (1994). On reflection, Equation 2 is the original equation published by Kijko & Funk (1994), while Equation 3 is a similar variant whereby σ_M is often quoted to be within the range of 0.2-0.3 (Bath 1973; Panza & Bela 2020).

$$M_{design} = M_{max} + 0.3 \quad (1)$$

$$\hat{m}_{max} = X_{max} + (X_{max} - X_{n-1}) \quad (2)$$

$$M_{design} = M_{max} + \gamma_{EM}\sigma_M \quad (3)$$

where:

M_{max} (X_{max}) = magnitude of the largest event currently recorded

M_{design} ($\hat{m}_{max}$) = magnitude of the design event used in support or hazard design assessments

X_{n-1}, $\gamma_{EM}\sigma_M$, 0.3 = modifiers to convert M_{max} to M_{design} as detailed in referenced publications.

Kijko & Funk (1994) described the original intent of this equation for use in limited data cases below:

> *"In the case of limited and/or doubtful seismic data, or when one wants to obtain quick results without going into sophisticated analysis"* (Kijko & Funk 1994).

Upon reflection, the author would caution against its standalone use in many cases, particularly given that more robust statistical options are now commonly available (mXrap 2026).

3.2.3 Stress fracture constant 'C'

The constant 'C' is a site-specific adjustment factor utilised in the calculation of the dynamic depth of failure 'd_f' as shown in Equation 4 (Scott et al. 2008).

$$d_f \sim {}^{W}/_{\sqrt{2}} \times \left(1.25\left({}^{\sigma_{max}}/_{UCS}\right) - C\right) \quad (4)$$

where:

UCS = unconfined compressive strength (MPa)

W = width of drive (m)

C = stress fracture constant.

In the case of Mine A, and most others, depth of fracture inspections are usually completed with borehole measurements in safely accessible development. In the author's opinion using these measured fracture depths to calculate 'C' from Equation 4 is one of the lowest confidence inputs to the design process given potential differences in localised bulking factors, loading timeframes, installed support pressure, lithology and drive orientation among others. The more recent introduction of LiDAR, and more specifically drone based options, allows for the measurement of 'd_f' post failure. Extensive databases of this nature are rare though given each of these measurements requires a failure accompanied by an accurate pre-failure measurement being taken between the most recent support application and failure occurrence.

3.3 Outcomes

The support system design outcomes for Mine A resulted in the creation of nine different dynamic support categories shown in Table 3, with 3 different ground support systems shown in Table 4.

Table 3 Mine A support system demand

Profile	Source proximity	Risk	Dynamic demand	Static demand	Displacement	Depth of failure	Support system
5.5 × 5.5m Arched Capital	<12 m	HIGH	16 kJ/m^2	11 t/m^2	129 mm	2.6 m	System A
	12–21 m	MED	10 kJ/m^2	9 t/m^2	109 mm	2.2 m	System B
	>21 m	LOW	9 kJ/m^2	8 t/m^2	99 mm	2.0 m	System B
5.0 × 5.0 m Arched Ore Drives	<12 m	HIGH	13 kJ/m^2	13 t/m^2	103 mm	2.1 m	System A
	12–21 m	MED	10 kJ/m^2	9 t/m^2	87 mm	1.7 m	System B
	>21 m	LOW	6 kJ/m^2	8 t/m^2	79 mm	1.6 m	System C
5.0 × 5.5 m Arched Capital	<12 m	HIGH	15 kJ/m^2	10 t/m^2	124 mm	2.5 m	System A
	12–21 m	MED	10 kJ/m^2	9 t/m^2	105 mm	2.1 m	System B
	>21 m	LOW	6 kJ/m^2	8 t/m^2	96 mm	1.9 m	System C

Table 4 Mine A ground support systems

System	Surface support	Bolting element
A	75 mm fibrecrete 5.6 mm weld mesh to grade line	3 m de-bonded Posimix bolts 1.4 × 1.5 m spacing
B	5.6 mm weld mesh to grade line	3 m de-bonded Posimix bolts 1.4 × 1.5 m spacing
C	5.6 mm weld mesh to grade line	3 m Kinloc (backs) with 3 m friction bolts (walls) 1.4 × 1.5 m spacing

Patterns were applied at a mine design level based on an understanding of the location of seismic sources outlined in Section 3.1. This allowed a targeted approach to the application of higher capacity dynamic support where relevant. It must be stressed that this approach at Mine A was based on a rigorous hazard management approach applied at a mine design level, with dynamic support upgrades being utilised as the final and least relied upon control. It is not uncommon to see this targeted style of support capacity increase applied to situations where there is low confidence in the location or nature of seismic sources. This is compounded by any low confidence assumptions made during the support demand calculations. This can lead to support demand calculations that may be notably lower confidence than is sometimes published in accompanying design documentation.

4 Mine B: Kaiser method with modifications

Mine B consisted of 2 LHOS mines operating below 500 m depth ('B-1' and 'B-2'). The design assessment at Mine B differed from Mine A in that Mine B had not experienced notable seismicity to date, with only 'B-1' having recently commissioned a seismic monitoring system. Stress related damage was minimal at the time of assessment, though life of mine modelling forecast moderate levels of seismicity at depth and support demand more than 3-times the planned support capacity for some areas.

The sequence of Mine B was such that areas of capital and ore drives being developed during and immediately post the dynamic support design period would be exposed to these conditions in an expected 4–6 years' time triggering the early dynamic support design implementation.

4.1 Method

The Mine B design case presents an updated version of the design methods applied at Mine A, incorporating refinements proposed by Potvin & Wesseloo (2013) addressed at peak particle velocity. Additionally various assumptions were required given the lack of input data available.

*4.1.1 Peak particle velocity and c**

The concept of near field saturation has been discussed in numerous publications and was addressed by Potvin & Wesseloo (2013) with the introduction of Equation 5. As discussed in Section 4.2.1, the consideration of near field seismic events led to this equation being utilised in the Mine B design case in lieu of those discussed by Kaiser et al. (1996) and Scott et al. (2008).

$$ppv = \frac{c^* \cdot 10^{0.5(m_L+1.5)}}{R+R_0} \quad (5)$$

where:

c* = assumed at Mine B to be 0.25 (Scott et al. 2008; Louchnikov & Sandy 2017).

R = distance to source

R_0 = the source radius estimated by (Kaiser et al. 1996).

$$R_0 = \alpha \cdot 10^{\frac{1}{3}(m_L+1.5)} \quad (6)$$

Where:

α = 0.53 – 1.14.

4.2 Input parameters

The input parameters for Mine B are published in the previously included Table 2 for easy comparison with those of Mine A. Similarly to the Mine A case, not all input parameters will be discussed in detail. The focus of discussion for the Mine B case will be on the assumptions made for dynamic support selection in an environment that had not yet developed notable seismicity or exhibited seismic related failures.

4.2.1 Magnitude 'M_{MAX}' and distance to source 'R'

With limited seismic history available there was insufficient information available to conduct any normal variants of M_{MAX} assessment. Additionally, the support design aimed to apply the highest-level support at level closeout areas that were expected to be mined in several years, meaning any short-term seismic relationships would be of limited value. Published benchmarking studies were reviewed (Louchnikov & Sandy 2017; Morissette & Hadjigeorgiou 2017; Dunn 2017) showing a range of magnitudes between 1.5 to 3.3 on the Mine B local magnitude scale. The final dynamic demand calculation utilised a M_{design} of 1.5 M_L given the early nature of the design process and seismic character of Mine B.

The selection of 'R' was based on an understanding of the expected seismic event drivers as well as a singular case of a 0.9 M_L event that had occurred immediately proximal to development. It was forecast that seismicity would likely be driven by localised stress accumulation in closing pillar regions and as such would be occurring near development. The final dynamic demand calculation utilised R = 10 m.

4.2.2 Stress fracture constant 'C'

At the time of design, no stress fracture measurements had been undertaken at Mine B. Numerical modelling forecasts suggested moderate damage with in situ stress levels in the order of 60 MPa, or 0.3-times to 0.5-times the expected rock mass unconfined compressive strength (UCS), towards the latter stages of the mine sequence. At the time this seemed to warrant selection of a comparatively low depth of failure, with d_f set as 0.1 m and a calculated C value of 0.63.

4.2.3 Building conservatism

The level of uncertainty involved in the Mine B design meant a conscious decision was made to add a degree of conservatism to some design assumptions. This was done through the selective use of either lower or upper bound assumptions at various points of design.

The constant 'α' used in the calculation of R_0 was used as the most conservative selection from a range of 0.53–1.14 (Potvin & Wesseloo 2013). This selection flows through design calculations as an increase in design peak particle velocity and resultant increase in remote triggered strainburst and seismic shakedown demand.

Despite Mine B utilising 5 × 5 m oredrive development, design calculations were only completed for 5.5 × 5.5 m development leading to larger failure volumes and increased demand for all 3 failure modes.

Lower bound values for UCS were used given no basis by which to accurately assign seismic support by domain. During early implementation support designs were assigned in a broad-brush approach to target the stress concentrations resulting from the mine sequence rather than specific seismic sources as in Mine A.

4.3 Adaptions

To address the gap in input data, Mine B undertook a benchmarking assessment that attempted to link dynamic design outputs, numerical modelling forecasts and published case study information. A simplified version of this process is depicted in Figure 3 below.

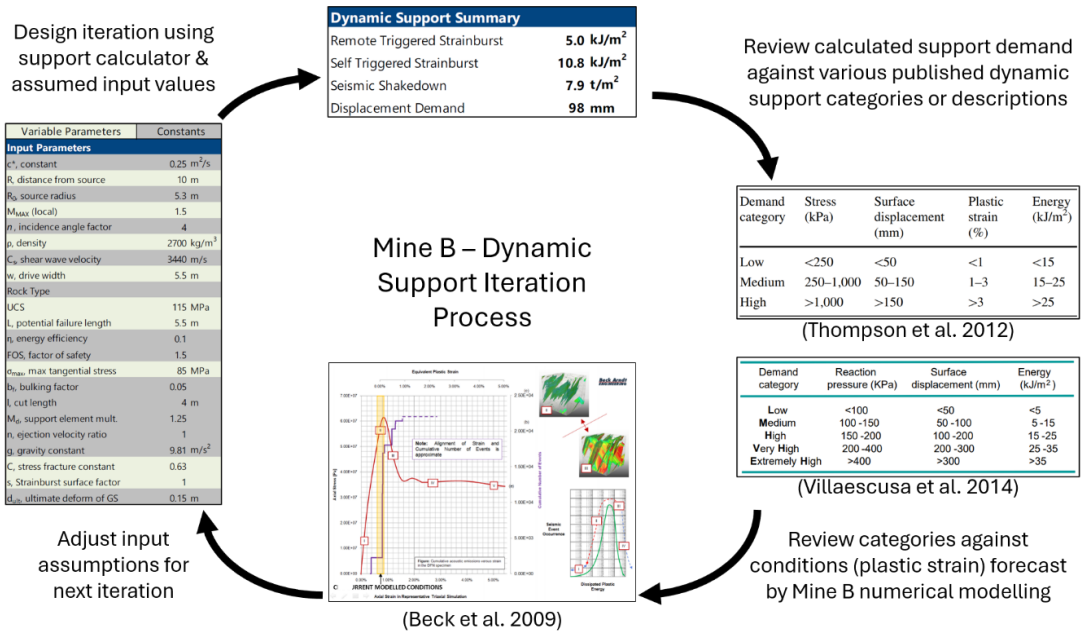


Figure 3 Mine B dynamic support design iteration process with reference to publications from Beck et al. (2009), Thompson et al. (2012), and Villaescusa et al. (2014)

4.4 Outcomes

Mine B presented an implementation challenge given the introduction of dynamic support prior to any seismic damage occurring. This was compounded by the all-too-common resistance from operations and management teams to the notion of changing from friction bolts to resin bolts. This required multiple iterations of support profiles, with the first design iteration described in Table 5 and the final accepted profile variants and implementation strategy shown in Table 6 and Figure 4.

Table 5 Mine B initial support profile classifications

GS#	Surface support	Bolting element
5/10	5.6 mm weld mesh to grade line	2.4 m de-bonded Posimix bolts 1.4 m x 1.5m spacing
11	50 - 75 mm fibrecrete floor to floor 5.6 mm weld mesh to grade line	

Table 6 Mine B final support classifications

GS#	Type	Use	Surface support	Bolting element
5	LOW	Capital (far)	5.6 mm weld mesh to 2.5 m	2.4 m Posimix bolts (backs) 2.4 m friction bolts (walls)
6	MED	Operational	5.6 mm weld mesh to grade line	2.4 m Posimix bolts
7	MED	Capital (near)	50-75 mm fibrecrete to floor 5.6 mm weld mesh to grade line	2.4 m Posimix bolts
10	MED-HIGH	Operational (seismic)	5.6 mm weld mesh to grade line	2.4 m de-bonded Posimix bolts
11	HIGH	Operational (structure)	50–75 mm fibrecrete to floor 5.6 mm weld mesh to grade line	2.4 m de-bonded Posimix bolts

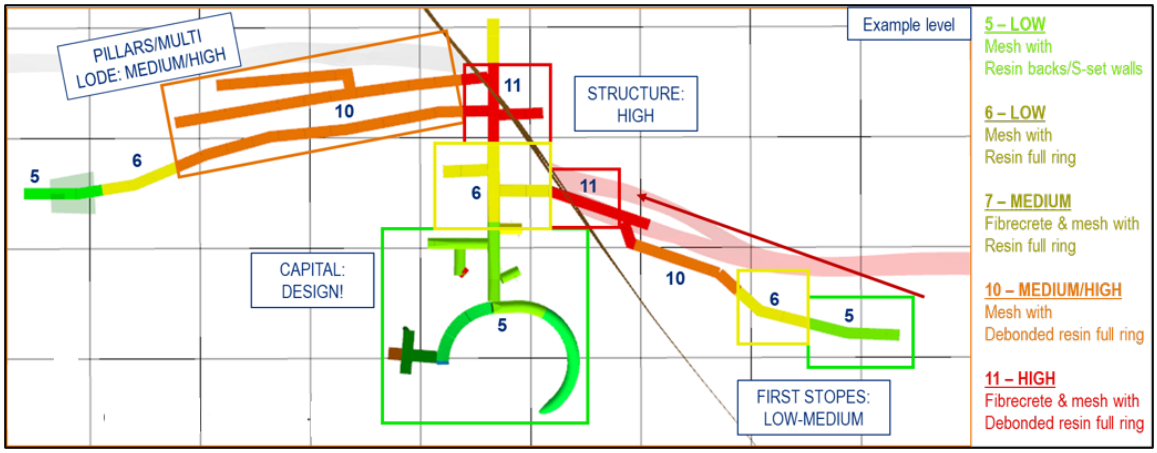


Figure 4 Mine B application of support design profiles by region

Arguments could be made either way for the success, or not of the Mine B support design. While an improved level of support (over friction bolts and mesh) was installed, the use of full grouted resin bolts driven by cost savings was not suitable for the conditions in the long run. The changes between Tables 5 and 6 resulted in a cost saving of ~AUD 700,000 over 2 years of mine life. Given the rehab costs and production delays that Mine B experienced through production stoping, an argument could be made that this initial design was insufficient. Alternatively, an argument could be made that noteworthy benefits were realised by designing

for increased support demand before signs of seismic damage were ever observed and that given the significant resistance to implementation, every area of support that was not friction bolts and mesh contributed to lower levels of damage, rehabilitation and overall cost throughout the mine's life.

5 Mine C: deformation-based support design

Mine C marks the final case study and a shift in design approach. The design assessment for Mine C focused on a sublevel caving operation that prior to design had experienced multiple large magnitude seismic events, damage and failures. This included events exceeding 2.5 M_L occurring at active development headings.

5.1 Method

Given the nature of Mine C, a decision was made to undertake the support design using DBSD techniques (Kaiser & Cai 2013; Kaiser & Moss 2022). DBSD was viewed by the author as industry leading at the time, as well as the most realistic reflection of dynamic loading mechanics currently proposed in a support design methodology. Figure 5, and the subsequent descriptions below, outline the dynamic failure mechanisms a drive is exposed to and their applicable assessment mechanisms.

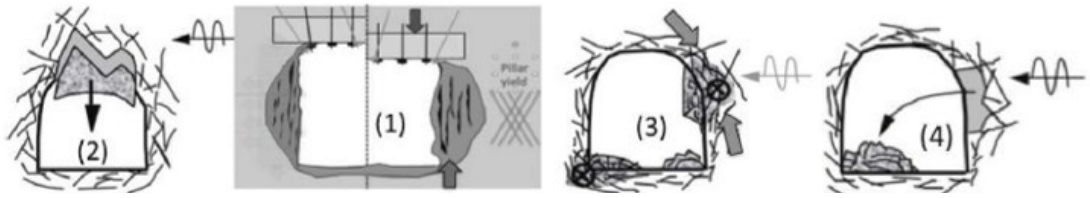


Figure 5 Drive dynamic failure mechanisms (after Cai & Kaiser 2018)

1. Slower (squeezing) strain-induced failures in backs, walls and floors not covered by this design.
2. Remotely triggered rockfall (shakedown) where acceleration forces from a remote event increase static load demand on a system. Assessed as per Chapter 8 of the CRBSHB (Kaiser et al. 1996).
3. Dynamic ground motion and energy (strainburst) where DBSD replaces previous CRBSHB concepts. Demand on support is realised as deformation that results from 2 related loading mechanisms:
    a. Static displacement demand occurring when brittle failure rock bulks as a result of fracturing caused by tangential stress on the drive. Calculated by estimating static depth of failure and linear bulking factor.
    b. Dynamic displacement demand occurring where dynamic depth of failure is increased by an incremental depth proportional to the depth of an associated strainburst event. Dynamic displacement requires the definition of burst depth, duration and thus velocity. It is also in theory an iterative process whereby each successive strainburst incrementally consumes displacement of the support, increases in situ fracture depth, and in effect becomes a new static depth of failure baseline.
4. Rock ejection from distant events ejecting a volume of rock mass. Represented by a displaced mass of material ejected against the ground support using standard kinetic energy calculations.

The design case for Mine C assessed failure modes 2 through 4 above to assess different support demand likely under different failure modes. In the interest of brevity for this publication, only the methodology applied to strainburst demand calculation will be discussed in detail for Mine C.

5.2 Input parameters

A selection of the input parameters used for Mine C are outlined below. This paper does not aim to describe the entire process in detail, with numerous referenced sources covering this topic. The intent is to discuss the selection of specific Mine C parameters.

5.2.1 Pre-burst conditions

DBSD assumes that a static depth of failure exists prior to the occurrence of a strainburst. This consists of a highly damaged zone (HDZ) and moderately damaged zone (MDZ). Each of these zones is assumed to make up a proportion of the pre-burst damage zone, and has its own unique BF driven by the level of fracture, support pressure and distance to excavation free face. The range of values used in Mine C are shown in Table 7. The concept of how these zones contribute to pre-burst deformation is illustrated in Figure 6.

Table 7 Mine C pre-burst deformation parameters

Parameter	Highly damaged zone (HDZ)	Moderately damaged zone (MDZ)	Source
Bulking factor (BF)	BF_{HDZ} = 4–8%	BF_{MDZ} = 3–8%	Kaiser et al. (2023)
% of d_{pre}	q_{HDZ} = 20–40%	q_{MDZ} = 60–80%	Perras & Diederichs (2016)

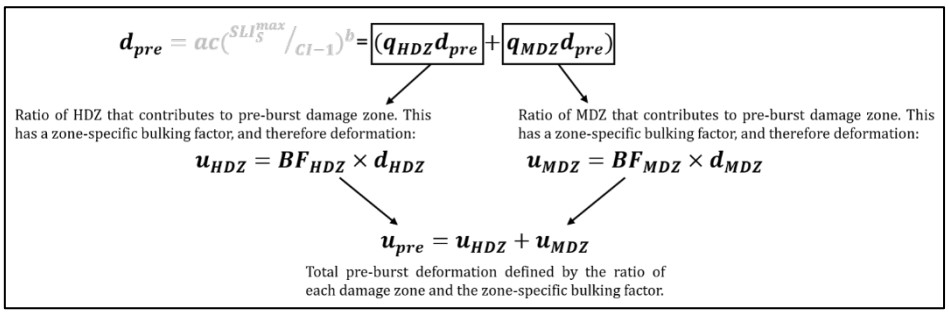


Figure 6 Pre-burst deformation components after Kaiser et al. (2023). HDZ = highly damaged zone; MDZ = moderately damaged zone

5.2.2 Strainburst demand

The deformation demand produced by a strainburst is similarly defined as shown in Equation 7:

$$u_{SB} = BF_{SB} \times d_{SB} \tag{7}$$

where:

d_{SB} = strainburst depth between 0.5 to 2.5 m based on literature review from Kaiser et al. (2023), after Kaiser & Malovichko (2022)

BF_{SB} = strainburst bulking factor ranging from 1 to 8%, noting that this factor trends from 0% at the edge of the strainburst damage zone (undamaged rock) to higher values as the rock is more damaged (Kaiser et al. 2023).

5.2.3 Strainburst velocity

In addition to the failure depth, rock mass bulking level and, therefore, deformation demand applied to support, the energy levels released during this process are dependent upon the velocity at which this failure zone is ejected, as defined by Equation 8.

$$v_{SB} = {}^{u_{SB}}/_{t_{SB}} \tag{8}$$

where t_{SB} is strainburst duration.

Strainburst duration should be back analysed by rock ejection distances from weakly supported failure zones or calculated from the duration of seismic signals associated with corresponding seismic events. Neither of these sources were available for Mine C meaning durations of between 10 to 40 ms were selected from literature review of values published by Kaiser & Malovichko (2022) and Lowther et al. (2022).

5.3 Outcomes

As all input data for the DBSD calculations were in the form of ranges of likely values, the calculation outcome did not provide for a single demand value as the design output. Rather, the input ranges for all parameters, as well as a likely median value, were used in a Montecarlo simulation. For each domain, 2,000 calculations were conducted, resulting in outputs as shown in Figure 7.

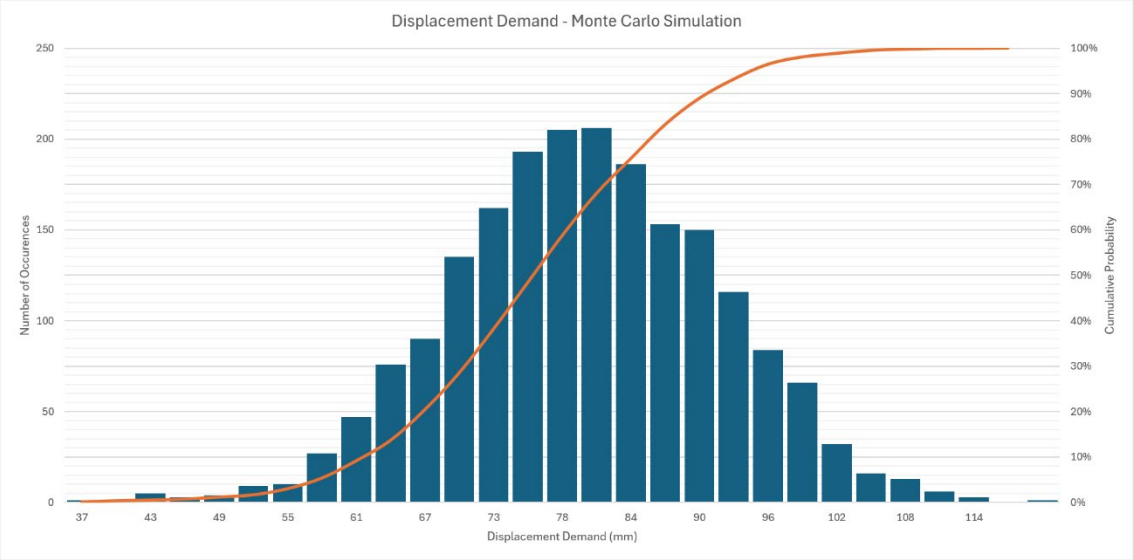


Figure 7 Mine C displacement demand Monte Carlo simulation output example

This output methodology allowed the selection of support demand based on likely ranges given the expected drive conditions, position, lifespan, influencing structure and geometry, among others. While a change in circumstance at Mine C resulted in further dynamic support no longer being required, indicative support profiles were expected to consist of:

1. 24 mm diameter resin bolts to increase early support pressure in development drives.
2. Minax 80/4 mesh encapsulated in dual layer fibrecrete to increase surface support capacity to reduce widespread surface support upgrades/replacement.
3. Patterned cable bolting between 4–5 m in length, depending on expected depth of damage.
4. A systematic process driven preventative support maintenance program to routinely restore support capacity where deformation had been measured.

6 Reflections on support design

The Mine A case study specified design demand to the nearest kilojoule. On reflection, quoting this level of detail was more than ambitious. Acknowledgements of the lack of knowledge on the intricate mechanics of dynamic support design are present through literature on the subject from numerous authors. With the benefit of hindsight, the only thing the author is confident of is that all 3 designs discussed in this paper were incorrect. It is the degree to which that is important.

The approach taken at Mine A was more robust than the author has seen in many examples since. Mine design was foundational to the management of seismic risk and associated drive stability. This appeared highly effective at the time and was likely a more significant factor than any level of support design and selection. As shown by Mine B, the use of calculations in support design allows engineers to easily quote dynamic demand values, even where many input factors are poorly calibrated for site or assumed from literature. Given the input data, the design demand calculated for Mine B was of reasonably low confidence and various support failures and revised input assumptions over the mine life were perhaps evidence of this.

The approach utilised at Mine C attempts to acknowledge this lack of confidence through reporting design demand in ranges by using variable input values. This approach by Mine C has been retrospectively applied to an 'Assumed' case that covers the input ranges of Mines A and B. For brevity, only the variable input ranges, as well as the calculated static demand outputs have been shown in Table 8.

Table 8 Hypothetical support displacement demand (shakedown)

Case	R (m)	ρ (kg/m³)	C_s (m/s)	UCS (kg/m³)	σ_{max} (MPa)	b_f (%)	C	Demand (T/m²)
Mine A	21	2,800	3,808	115	140	5	0.83	13.2
Mine B	10	2,700	3,440	115	85	5	0.65	9.7
Assumed	30–300	2,800–3,000	3,700–3,900	110–120	105–180	4–6	0.65	10–26

R = Distance from source; ρ = Density; C_s = Shear wave velocity; UCS = unconfined compressive strength; σ_{max} = Maximum tangential stress; b_f = Bulking factor; C = Stress fracture constant

Figure 8 presents Monte Carlo output ranges as a 'Low', 'Likely' and 'High' case. This provides a more realistic indication of the demand ranges that support is likely to encounter in the expected range of conditions, allowing engineers to make better informed decisions around support application while accounting for additional factors such as data confidence and failure consequence.

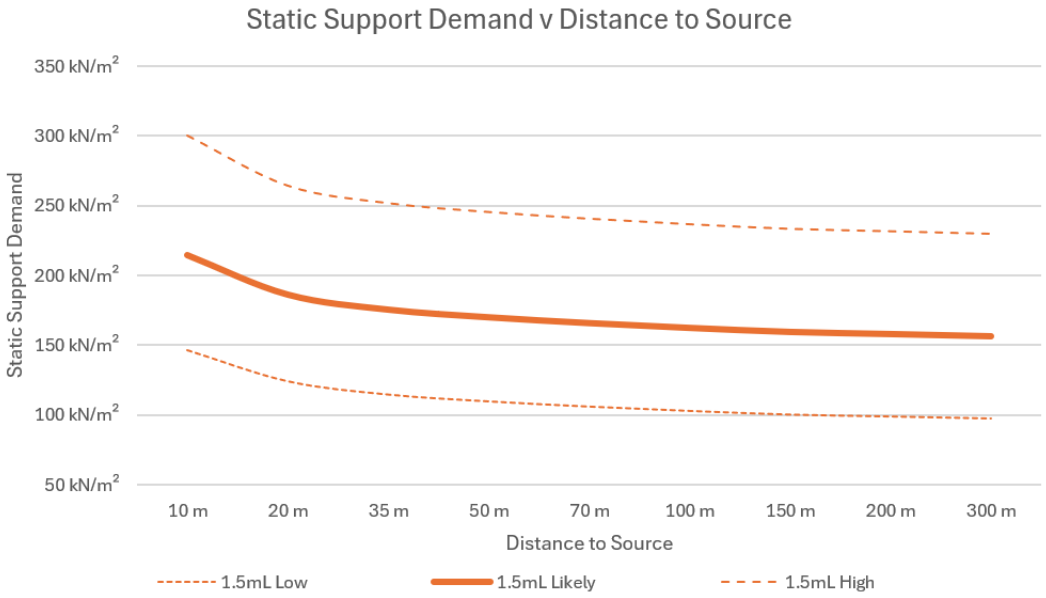


Figure 8 Static support demand versus distance for 'assumed' parameters from Table 7

7 Conclusion

This paper presents a review of 3 dynamic support design approaches applied at different Western Australian mine sites, highlighting the evolution of dynamic support design methods and their validity, effectiveness, limitations, and lessons learned. The nature of dynamic support design is such that inputs are variable and often of low confidence, while conditions, by their nature, evolve rapidly. Over the evolution of these designs, there has been a transition from prescriptive, definitive design to an acknowledgement of the variable input parameters and the associated lack of surety in design demand. This field of dynamic design is still evolving and the wider application of methods such as DBSD provide new avenues for design, though limited confidence in many input values will likely remain for most practicing engineers. This can be addressed through a move away from prescriptive single value design outputs to more realistic acknowledgement of the range of conditions that could be expected. Holistic application of elimination and engineering controls, often the last of which is ground support, can then be utilised in combination to manage the risk of seismic related falls of ground.

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