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# OPEN PIT OPERATORS CONFERENCE 2022

Conference Proceedings

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**21– 22 JUNE 2022**
**PERTH, AUSTRALIA**

The Australasian Institute of Mining and Metallurgy
Publication Series No 3/2022

Published by:
The Australasian Institute of Mining and Metallurgy
Ground Floor, 204 Lygon Street, Carlton Victoria 3053, Australia

All papers published in this volume were peer reviewed before publication.



ISBN 978 1 922395 07 8

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We would like to thank the following people for their contribution towards enhancing the quality of the papers included in this volume:

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# FOREWORD

Dear industry peers,

On behalf of the Conference Committee, I am delighted to welcome you to the AusIMM's Open Pit Operators Conference 2022.

Australia is a world-leading miner and producer in many commodities, including iron ore, gold, coal and more. Hosting this conference in Perth and online allows us to bring together some of the world's best experts who are driving Australia's sector forward into an exciting future.

The plenary discussions and technical program at this year's conference showcase the latest advancements in open pit mining. Leading global experts will deliver thought-provoking keynote presentations, with a focus on digital innovations and smart mines, social licence to operate, geotechnical engineering and more.

Our presenters will share insights to help us all learn how we can continue to deliver value for our companies and communities. Alongside the learnings captured in these proceedings, presentations are available for delegates to watch on-demand, giving you ongoing access to the great content on offer.

I would like to thank our speakers and authors who have prepared presentations, compiled technical papers and travelled from far and wide to be part of this year's conference. I would also like to thank the Conference Organising Committee, paper reviewers, and all our sponsors and exhibitors including BHP, Dassault Systemes, Epiroc, Caterpillar, Dyno Nobel, Nesch Mintec and OZ Minerals.

Finally, I would like to thank you – our delegates. Without our dedicated professional community and your continued support, this event would not be possible.

Yours faithfully,

Joe Clayton

Open Pit Operators 2022 Conference Organising Committee Chair

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# Minimising coal damage and loss due to cast blasting

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## ABSTRACT

The main objective of the blasting phase in a coal mining operation is to maximise the coal recovery. However, coal loss due to blasting is a serious problem. The Australian Coal Association Research Programme (ACARP) reported in 2011 that in Australia one in every ten coalmines is completely wasted due to such losses. Coal is either lost in the spoil, largely due to the blasting operation or becomes diluted and cannot be recovered entirely. The major coal loss mechanisms include losses from the coal seam roof, the floor, the front edge and from dilution.

Coal damage and loss mechanisms during cast blasting that have been observed by various authors include:

- Excessive cast dragging the coal edge in the cast direction. Movements of this nature are often associated with the formation of trenches.
- Spatial movement of overburden where rock shears and moves in a plane above the coal seam, which damages and dilutes the coal.
- Rockfalling over the coal during the blast often producing indentations in the exposed coal.
- The inappropriate use of bulk and initiating explosives that often results in coal damage.
- Insufficient stand-off distances from the coal.

The Australian coal mining industry has gone a long way towards addressing the coal loss issues since 2011 – almost all operations now deploy touch-coal drilling and sporadically use gamma logging for loading overburden. A study was conducted to compare the effectiveness of touch-coal information in which some touch-coal blastholes were drilled on a fixed or random pattern. In another study, some blastholes were drilled to the touch-coal, gamma-logged and mapped to determine stand-off distances. It was found that the coal roof surface created by gamma-logged holes was the most effective option for minimising coal loss. As a result, a variable stand-off distance was found to be the key to minimising losses.

This paper discusses the importance of coal mapping, the rock properties of the roof and floor interface of the coal seam, and the significance of loading and initiating designs in minimising coal damage. Furthermore, the importance of using electronic initiators cannot be emphasised enough. This paper also discusses how a cast blast can be initiated from many locations by mitigating geological abnormalities and thereby avoiding spatial movement of rock shearing in a plane above the coal seam.

## INTRODUCTION

Any increase in cast to the final spoil translates into savings when applied to horizontal or gently dipping coal deposits. Stressed market conditions, in association with high stripping ratios, compel operations to maximise cast in order to stay relevant. Explosives or chemical energy remain by far the cheapest option to move more overburden to the final spoil, and thereby reduce the excavation volumes that need to be excavated by the machines.

Limited knowledge of future coal prices sets capital constraints on operations, compelling them to regulate high strip-ratio coal mining. This cost pressure won't disappear in the foreseeable future, so long as the emerging markets dictate commodity pricing. In this context, coal mining operations can be subsumed into two categories: highly mechanised or manual (less mechanised). Hence there are two types of operations trying to co-exist commercially in the global business. Under these conditions, it is difficult to reduce unit costs by preserving the 'status quo' (mining method and cost structure).

Operations with medium to long-term mine plans that provide a clear picture for the future are the biggest winners when the pressure on scheduling is not so great and somewhat obliging. It is clearly beneficial to accommodate changes in these types of operations without interfering with daily mining schedules. The goal is thus to maximise cast and coal recovery without changing schedules. In the Australian context, there are three distinct mining methods where cast blasting can be beneficial: Cast-Doze, Cast-Dragline, and Cast-Truck and Shovel, implemented according to a cost ($)/$m^3$ basis. Cast-Doze, uphill (or downhill), depending on the geometry of overburden to the final position (beyond the pivot-line) is critical for the economic benefit of this method. In this regard, at various sites Cast-Doze has proven to be the most cost-effective method. Likewise, for the Cast-Dragline operation, considerable savings result from the volume of overburden that does not require handling. Depending on the size and capacity of the dragline, muck pile shaping (eg forming a ramp, cast and stand-up with the help of blasting where required) can be created to reduce overall mining costs. Cast-Truck and Shovel has proven to be by far the most expensive depending on lead distance to the dump.

Other aspects that are important include the clever use of chemical energy to move overburden to final spoil, minimisation of coal movement and loss, use of various geophysical techniques, and the resourceful application of initiation designs are discussed in this paper.

## VARIOUS FORMS OF COAL DAMAGE OBSERVED IN THE FIELD

Various forms of coal damage have been noted over the years such as coal edge loss, indentation and the coal seam being pushed into the floor often due to the presence of geological abnormalities (Kanchibotla, Laing and Grouhel, 2006).

When attempts are made to maximise cast to save costs of removal, the coal edge can be dragged laterally in the direction of the cast. This single process accounts for the most loss and damage by far. Figure 1 shows both the coal edge moving laterally outward in the low wall and trenches forming. The shifting of the coal edge beyond the crestline (or excavation line) is a worst-case phenomenon when attempting to maximise cast.

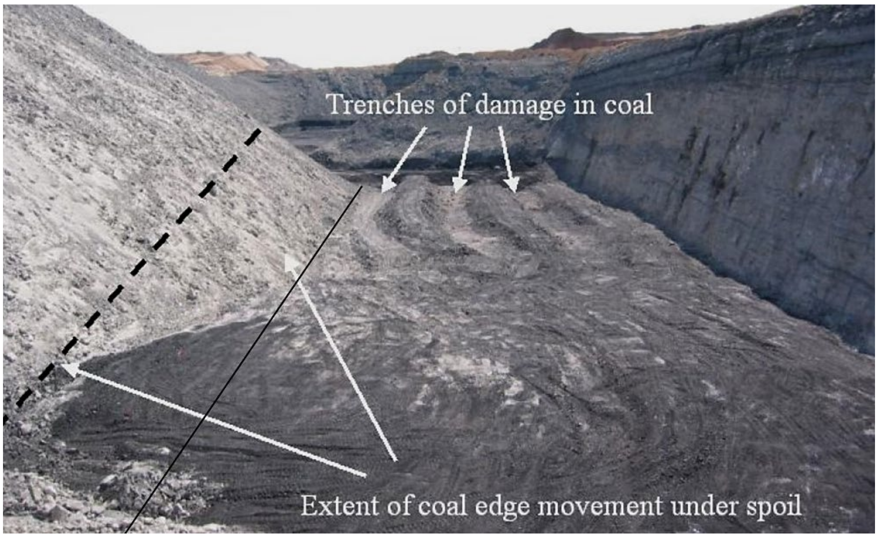


**FIG 1** – Coal edge laterally moving out into the low-wall and formation of trenches.

It is a well-known fact that the front three to four rows contribute to the cast. This depends on the bench height, strike length, width, availability of void space, and geology (rock type, jointing and structures). Burden profiles of these front rows can be modified to maximise cast percentages by controlling various contributing factors such as face burdens and burdens of subsequent rows, explosive type and loading and timing between rows. In the past, when non-electric initiators were the only choice for cast blasting, the aim was to achieve consistent face velocities across the whole face to produce the required muck pile profile. With these, choices were limited because the blast had to be fired in one direction by maintaining a burning front to avoid cut-offs. However, with the advent of extremely accurate and programmable electronic initiators, significant progress has been made in maximising blast outcomes.

In the past, face velocities were an important measure which influenced percentage cast. However, face velocities only provided information on the effect of the front-row holes. They gave no direct information on the effect of the later firing rows, which contribute to overall cast. Various methods have been used to study the relationship, including the use of microwave and radar units and high-speed filming; it has been concluded that the percentage cast increases with an increase in the face velocity (Henley and Felice, 1992). It has also been concluded that face velocities of 20–22 m/s are the optimum achievable in Australian coalmines. However, at velocities exceeding 22 m/s 'face burst' cannot be ruled out.

With the use of electronic initiators, a blast can be initiated from multiple locations so that maintaining both a burning front and a face velocity of 20 m/s is no longer necessary. Such multiple location initiation also depends on the location of the shearing plane above the coal roof interface (which is influenced by geological abnormalities) and the expected blast outcome. For example, a cast blast starting in a 'v' at the endwall, turning around row-by-row, stopping and restarting from the other end of the blast, and finally a centre zip used in the back four rows to stand up the muck pile. The required outcome from this blast was to control vibration on nearby power lines, dynamically form a ramp for dragline entry, to maximise cast and to raise the muck pile for the dragline pad closer to the key to aid bucket reach (Goswami *et al*, 2015).

Other coal damage events have also been observed. Rockfall over the coal roof during the blast sometimes produces indentations, which is experienced in the field. There are three distinct movements that occur during any cast blast (Figure 2):

- In a typical cast blast scenario, both the toe and the collar of the blast lag behind, with the face undergoing flexural rupture. As a result of this movement, a horizontal piston effect is generated due to rockfalling into the void. This movement can be controlled by timing and burden (Goswami *et al*, 2015).
- At the collar of the blastholes, the atmosphere acts as a free face which helps in creating a vertical piston effect by lifting the rock into the atmosphere. This phenomenon can be controlled by adjusting the timing between the rows and the total blast duration.
- The most important movement from coal damage point of view is difficult to characterise, is the shearing of higher density rock directly above the coal roof, usually of lower density. This shearing plane can sometimes be seen in high resolution videos taken from the front of the face (in the direction the blast is firing). In most cases however the dust cloud quickly obliterates the underlying rock which means that the footage cannot be used to modify the next blast design. These days drones are used to video blasts to better understand movement of these shearing planes. One such incident is discussed below (section 'Loading and initiating a cast blast to protect coal') where the shearing plane above the coal changes its course to follow a relatively weaker layer disrupting the coal seam and laterally shifting in the low-wall.

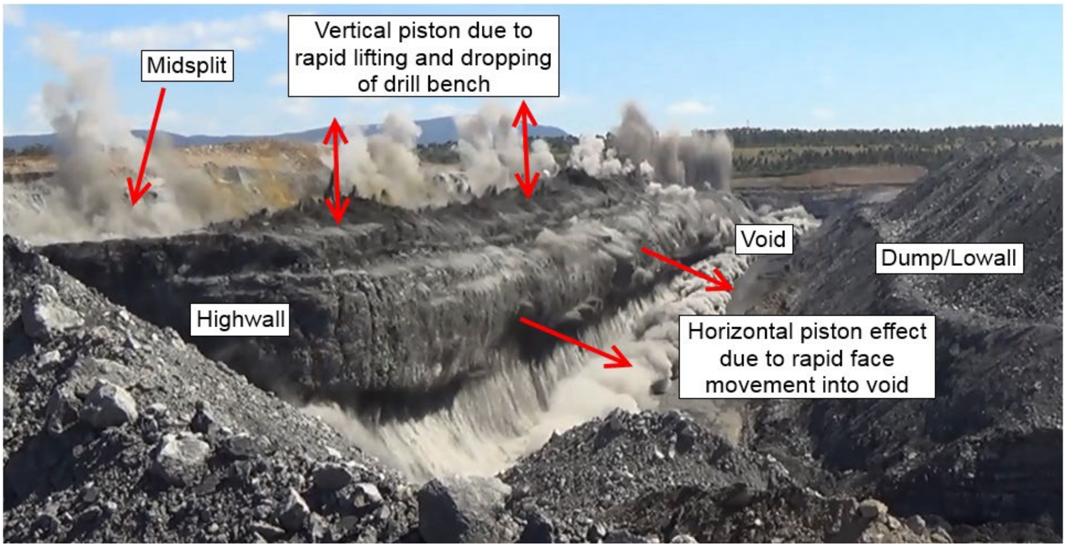


**FIG 2** – Various movements that occur during a cast blast (Goswami *et al*, 2015).

Mapping of the coal roof, accurately identifying deformities, working out stand-off distances, and determining the likely effect of the shearing plane are key in protecting coal movement and damage.

## MAPPING THE COAL ROOF SURFACE

It is important when designing a cast blast to accurately locate and map the coal roof. An attempt has been made here to compare the best options that are available for mapping the roof of the coal seam to protect it from blast induced damage and movement. These options include:

- Geological versus touch-coal model.
- Extrapolated surface from previous strip versus touch-coal.
- Touch-coal surface versus exposed coal roof (surveyed) on a regular pattern.
- Touch-coal surface versus exposed coal roof (surveyed) on a random pattern (only 5 per cent of the total number of productions blastholes being drilled first prior to drilling the rest of the pattern).

### Geological model versus touch-coal model

Various surfaces were created to determine the discrepancies of the geological model.

Figure 3 compares the geological model with a surface created from the exposed (surveyed) roof of the coal. Differences between the two surfaces along the cross-sections labelled ‘A’, ‘B’, and ‘C’ were calculated to be 4, 1.2 and 2 m respectively (Table 1).

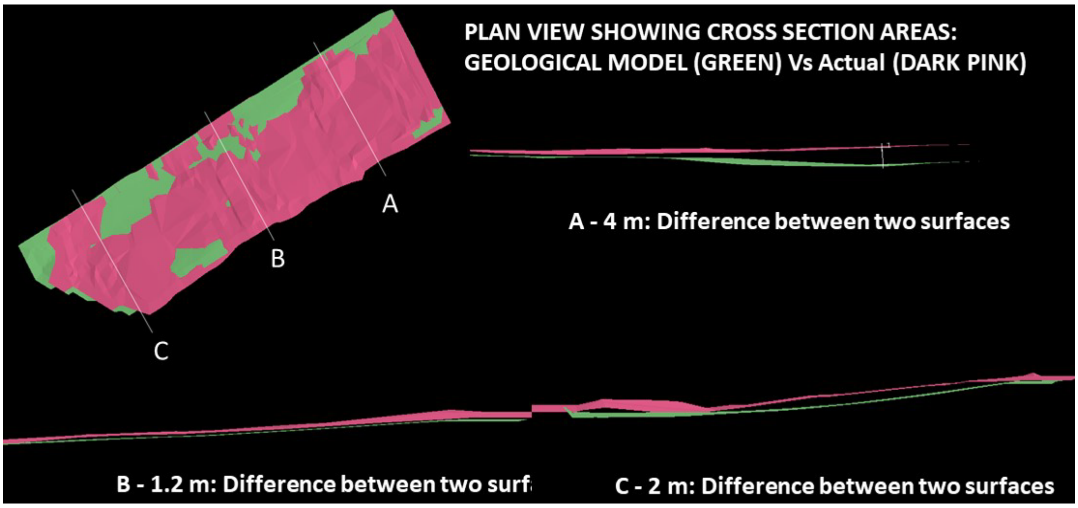


**FIG 3** – Surfaces showing difference between the geological model versus touch coal.

**TABLE 1**

Variation showing coal location – a surface created by using touch-coal information from random holes shows the best correlation to the exposed roof of coal.

| Comparison | Section A | Section B | Section C |
|---|---|---|---|
| Geological versus touch-coal model | 4 m | 1.2 m | 2 m |
| Extrapolated surface from previous strip versus touch-coal | 6.3 m | 5.2 m | 3.1 m |
| Touch-coal surface versus exposed coal roof on a regular pattern | 1.3 m | 1.5 m | 0.6 m |
| Touch-coal surface versus exposed coal roof on random pattern | 0.8 m | 0.2 m | 0.2 m |

## Extrapolated surface from the previous strip versus touch-coal

To streamline the anomalies of a geological model, an attempt was made to expand the surveyed coal roof surface of the strip in front. Unfortunately, the discrepancies increased to 6.3, 5.2, and 3.1 m respectively (Figure 4) at the three labelled cross-sections.

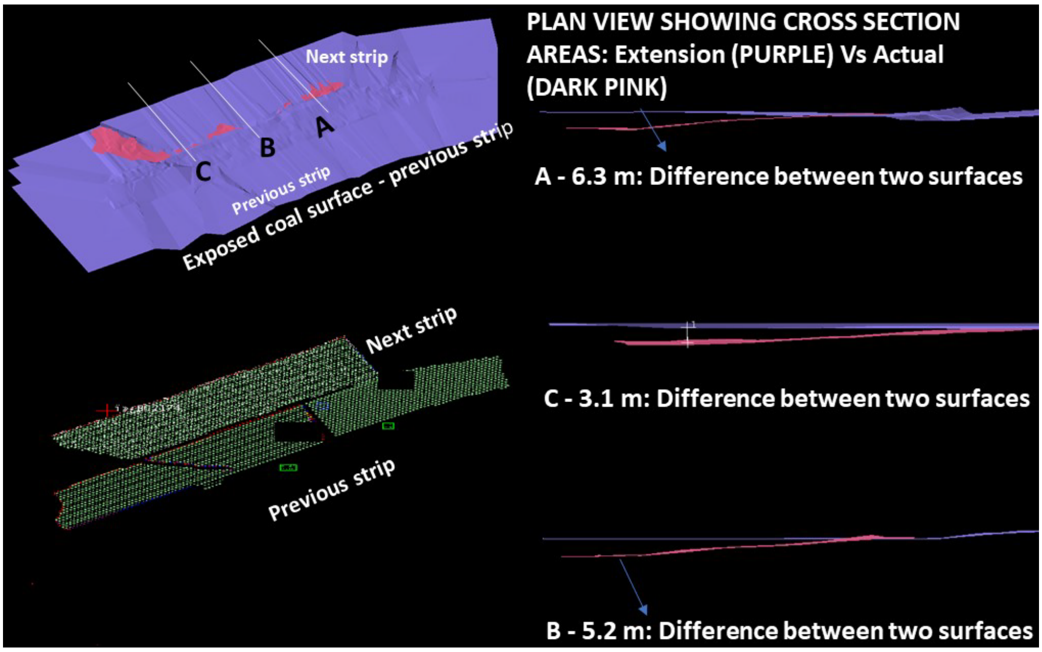


**FIG 4** – Expanded surface from previous strip showing significant variation in coal location.

Although the model displayed significantly larger discrepancies, some insights were gained from undertaking this exercise:

- The extension of the exposed coal surface from the previous strip cannot be used to derive a pattern from which it is possible to arrive at a powder factor that will result in efficient digging of the muck pile and will avoid any coal damage.
- The extension of coal surface from the previous strip carried geological abnormalities. By properly mapping the extent of these abnormalities, useful changes can be made in designing the blast in order to minimise coal damage/loss.

## Touch-coal surface versus exposed coal roof on a regular pattern

Surface accuracy created from driller's touch-coal has been compared with the surveyed exposed coal surface. Most coalmines in Australia drill touch-coal holes every third or fourth hole along the row, and then repeat this every third row. A surface is then created from the driller's manually noted touch-coal logs, which are usually inaccurate. After applying a site-specific stand-off distance, another surface is created. Loadsheets with backfill, mass of explosives per hole, stemming height etc, are then produced from this model.

There seems to be insignificant deviation between the two models in this particular exercise. However, touch-coal holes must be drilled along with the production drilling based on an assumed powder factor and a pattern. In other words, all production and touch-coal holes are drilled simultaneously (touch-coal holes are not drilled first). Once the whole blast is drilled, it is too late to make alterations; the pattern cannot be adjusted and therefore neither can the powder factor. It should be noted here that in the author's experience there is only a metre of standoff which can influence hard dig vis-à-vis coal damage, depending of course on the rock property immediately above the coal roof.

A similar exercise showed distances between touch and exposed coal surfaces; 1.3 m, 1.5 m and 0.6 m differences were calculated (Table 1). This close proximity confirms the good quality of drilling and modelling; however, it does not necessarily help produce a logically correct design for the next strip.

## Touch-coal surface versus exposed coal roof on a random pattern

This exercise examined the benefits of creating a surface by drilling 5 per cent of the total production blastholes in a random order to determine how accurately this can predict the coal roof surface. A surface was created using random holes and was found to be most accurately correlated to the actual exposed coal roof, as shown in Figure 5.

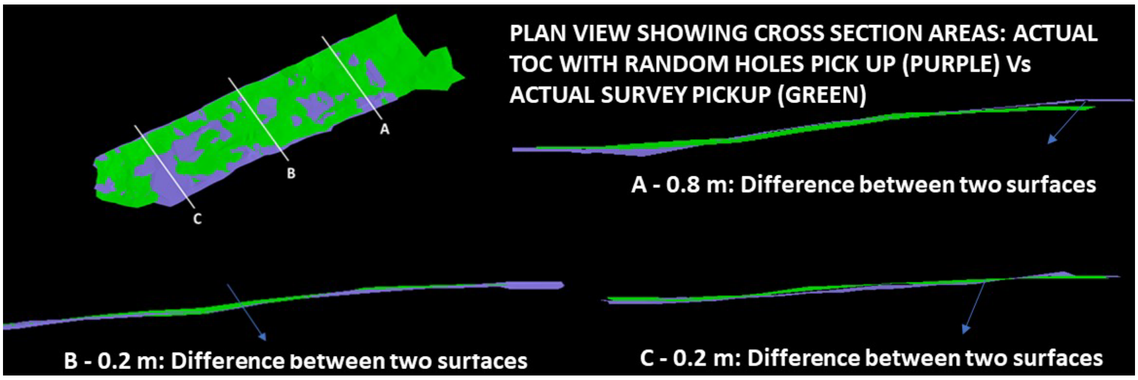


**FIG 5** – Randomly drilled touch coal holes (only 5 per cent) prior to commencing production drilling.

Three cross-sections as labelled showed differences of 0.8, 0.2 and 0.2 m respectively. It is therefore recommended that 5 per cent of the total number of holes be drilled to touch-coal prior to drilling the production pattern. For example, a blast with 600 holes requires only 30 holes to be drilled as touch-coal, randomly located across the whole blast. A surface from this data should then be used to create a coal roof surface model that is used to determine the load sheet.

This may not always be practical but once it is known that the area typically produces hard dig or coal damage, this will assist in resolving issues and improving digability. This is also expected to save time, effort and importantly cost of backfilling.

Figure 6 shows an example of a coal roof surface created from drill holes randomly drilled covering the whole blast block. These touch-coal holes were gamma logged for density, natural gamma and calliper. This practice has been repeated on several occasions and proven to be the most effective method to map the pre-blast coal roof. A fold (roll) and a fault are identifiable in the figure where the blast was initiated from the centre of an anticlinal fold to protect the coal from moving. Whereas a normal fault was initiated in the direction of footwall to hanging wall blocks (Figure 6). By accurately mapping these geological abnormalities, the blast was loaded and initiated carefully, and coal damage and loss was prevented. Coal recoveries were monitored in three consecutive strips where these abnormalities existed and are reproduced in Table 2.

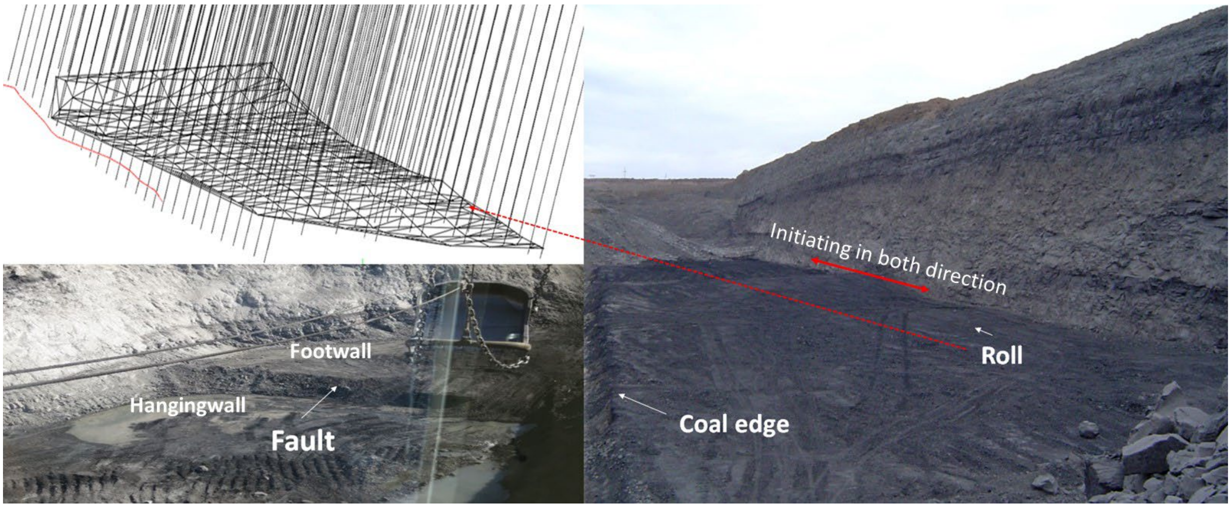


**FIG 6** – Pre-blast coal surface versus exposed coal roof showing presence of a fold and a fault (source: Goswami, Brent and Hain, 2008).

**TABLE 2**

Coal seam recovery for three strips compared to XPAC model predictions.

| Strip | 25 | 26 | 27 |
|---|---|---|---|
| Expected coal as per XPAC model including % dilution/loss allowed by mine | 454 243 t | 417 658 t | 388 939 t |
| Actual recovery | 460 048 t | 436 466 t | 391 468 t |
| **% Recovery** | **101.3%** | **104.5%** | **100.7%** |

## ROCK PROPERTY

The rock condition immediately below and above the coal seam is seldom given much attention. However, it is believed to play a significant role. For example, if the floor of the coal is soft (such as saturated mudstone or shale), excessive explosive energy might push the coal down into the floor, thereby diluting the coal. Conversely, if the floor comprises competent rock (such as conglomerate), then the possibility exists for the coal to move out horizontally (or laterally) and thus to be lost in the spoil. It is important to drill a few holes through the coal seam in a blast block to determine what type of floor the seam rests on.

Addressing the rock condition immediately above the coal seam is equally important, especially in cast blasting, in order to avoid various forms of coal damage and movement. Many techniques such as ski jumping, buffering, and baby decking, are currently used with mixed success (Marton, 1988). Not enough attention is given to proper assessment of the rock properties above and below the coal seam. As a result, although these techniques are sometimes successful, they remain vulnerable to failures.

When estimating rock strength, density, natural gamma and calliper logging are by far the most useful of the presently available downhole geophysical techniques. These measurements do not require water retention in the blastholes, are easy to use, and are inexpensive. Operations where coal damage is an issue have a duty to embed gamma logging as part of their drill and blasting (D&B) practice.

Density (or gamma-gamma logging) measures the back scattering of incident gamma radiation from blasthole walls (in atmospheric media) which is directly related to the electron density of the rock and hence its bulk density. Clearly, this is very useful for determining the rock types (in-hole) involved in blasting. Most density logs are presented in logarithmic format, emphasising the location of coal (density 1.3 to 1.6 t/m$^3$). In linear format, these logs can identify other rock layers (usually 2.1 to 2.4 t/m$^3$) while above this range the density/strength relationship ceases to be significant. This property is very useful in effectively determining standoff distances which can help avoid coal damage.

Other properties, such as natural gamma [logging] and calliper [measurements], must be used together with the rock density. Any significant deviation between these three measurements, ie density, natural gamma and calliper, provide useful information. Typical logs can be seen in Figures 7 and 8. Figure 7 highlights the zones above and below the coal seam where the density and gamma change considerably. Figure 8 suggests that decreasing gamma with increasing density is indicative of an altering rock type requiring attention in designing the blast. Once all the gamma logged holes are analysed, a stand-off distance for each hole is calculated and a surface is created for generating a load sheet. This also allows to create various shearing horizons for a multipoint initiation. Many operations have rock properties from exploration holes which are also useful in verifying geological abnormalities and should be incorporated in the D&B process. However, from a blasting point of view, continuous inhole logs are preferable to rock properties derived from other forms of sampling such as cores or drill cuttings. This is because these only provide spot samples which are not necessarily representative of properties along the whole length of a blasthole.

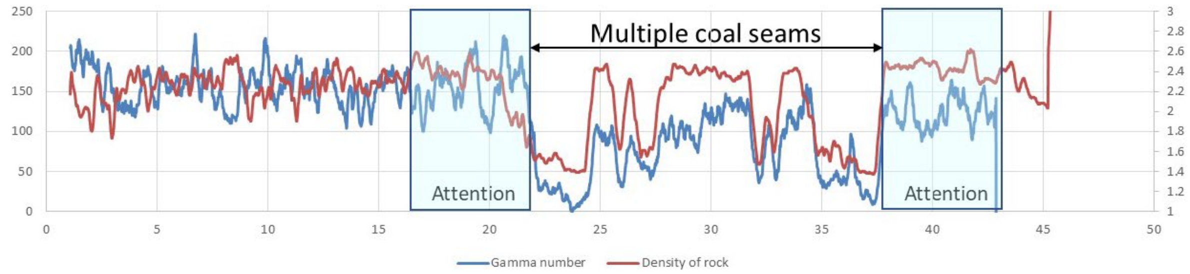


**FIG 7** – Natural gamma and density plot highlighting rock properties requiring increased stand-off distances.

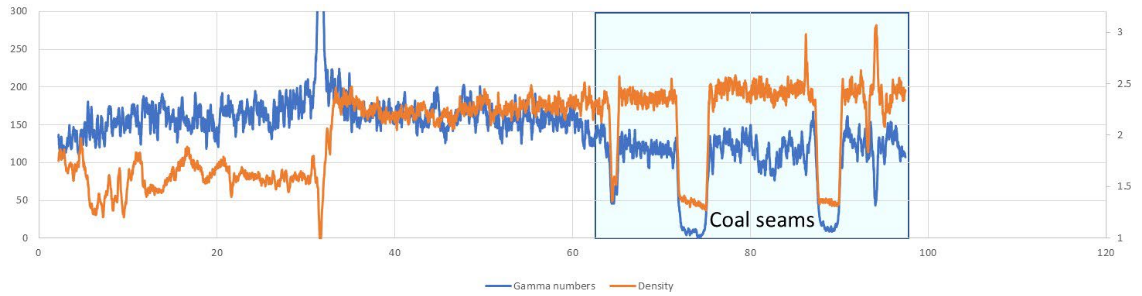


**FIG 8** – Increasing gap between gamma and density indicating altering rock type.

## LOADING AND INITIATING A CAST BLAST TO PROTECT COAL

Once all randomly drilled holes are analysed, the roof of the coal is located along with any geological abnormalities. A coal roof surface is then created. One important step here is to pay attention to the floor of the coal, determining whether it is soft (shale, mudstone, siltstone or competent sandstone etc), and accordingly making allowances in the stand-off distances. It has been observed that a soft floor with an inadequate stand-off distance tends to push the coal into the floor. Conversely, if the floor is competent rock, the coal edge may shift out laterally. In such cases, stand-off distances in the front rows must be adjusted.

Adjustment of the stand-off distances for every blasthole from the roof of the coal is equally important. In a high energy release situation, weak layers tend to provide easy passage. Identifying layers and allowing stand-off distances for such formations is crucial for protecting coal. Figure 9 shows how the presence of a weak layer propagating energy release above the coal seam as well as above the dyke can damage and move coal with it. These situations have been successfully averted by carefully allowing extra stand-off distances and by initiating the blast from multiple locations. A stand-off distance of up to nine metres have been successfully used to protect the coal without creating hard dig. Multipoint initiation has also been successfully used to mitigate vibration and overpressure issues (Goswami *et al*, 2015) at several environmentally sensitive operations.

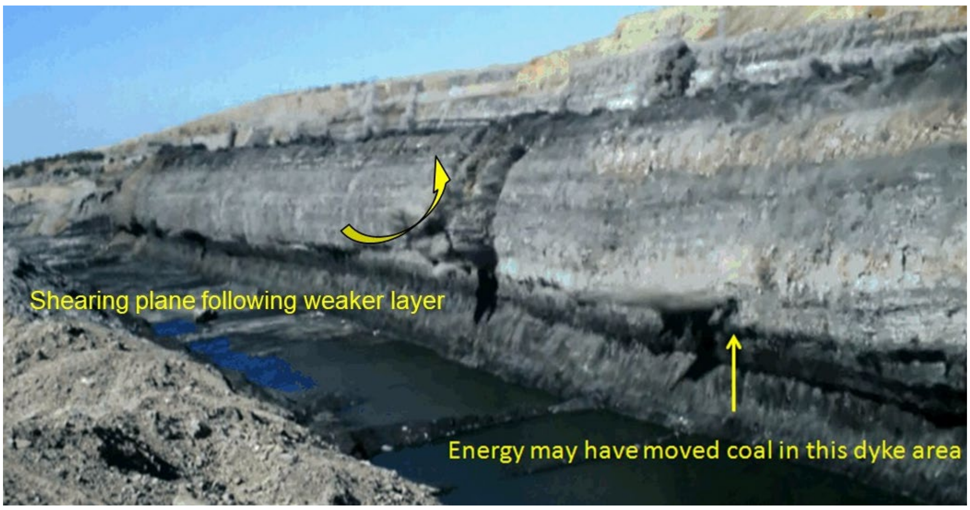


**FIG 9** – Presence of a weaker layer providing a conduit for energy transfer/release.

The presence of in-hole water must be given due consideration. Dewatering holes, gas bagging and selecting explosives that produce less gas are found to be good practices.

## INITIATING A CAST BLAST TO PROTECT COAL

With the usual practices of initiating a cast blast using non-electric initiators there are small delay intervals between the spacing and larger delays between rows. Once a practice is deemed successful at a site, it is generally adopted as standard for all cast blasts and is seldom changed.

With the widespread acceptance of more accurate electronic initiators, the industry is evolving, and attempts are being continuously made to not only avoid coal damage but also to increase productivity. The latter is achieved, in part, through muck pile profiling by dynamically producing ramp, cast and stand-up within the one blast event. Cast blasts with longer strike lengths, reduction in the number of blast events, reduction in stoppage times and environmental compliance are some other benefits that are claimed.

Electronic systems do not require a burning front because once the detonators are programmed, all forms of communication (predominantly lead wires) with the initiators become redundant. This provides greater flexibility and opportunities for initiating a blast from multiple points.

For example, a blast 1.3 km long, 60 m wide and 55 m high is shown in Figure 10. It was a requirement to blast carefully to protect an upper thin 0.6 m layer of premium coal seam from any damage or loss, no vibration exceedance on the nearby power poles. There were approximately 1400 blastholes in this blast. Out of these, 60 gamma logged holes were drilled first, prior to commencing drilling the pattern. Of these 60, ten randomly scattered holes were drilled through the floor of the coal seam. Holes were drilled 8 m from the seam floor to accommodate the gamma tool.

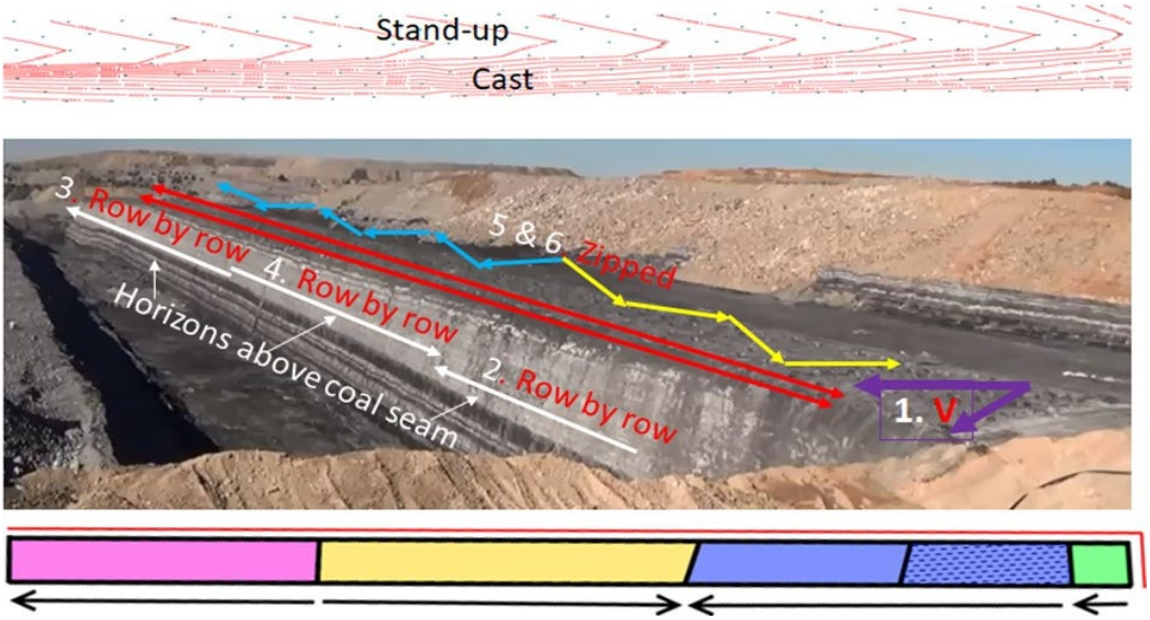


**FIG 10** – A cast blast being initiated from seven locations.

All the gamma logs were analysed for the three previously discussed important aspects which control the coal damage and movement: the location of the roof of the coal, determined with the help of natural gamma, density, and calliper data; the rock density five metres above the coal roof, and, the quality of the coal floor, ie soft, hard, etc. A stand-off distance for each hole was ascertained from these measurements to establish the proximity of the explosive charges. These variable stand-off distances in effect form various shearing planes (horizons) which are then used to generate load sheets and initiation details for blasting from multiple locations. It is then vitally important that the blast is loaded by following the load sheets exactly with each hole being dipped several times as backfilling holes accurately is key to averting any coal damage/loss.

The blast in Figure 9 which followed the above design procedures is an exemplary example of a multiple-point cast blast initiated from nine separate locations designed to control vibration, to form a ramp, to maximise cast and to stand-up the last rows to facilitate the dragline operation. And most importantly protect premium coal seam from any blast induced damage/loss. This blast involved:

1. A mid-split along the strike and endwall initiated from three different locations.
2. A lower powder factor zone was initiated as a ‘V’ at the eastern end to control vibration and to form a dragline entry ramp.
3. A row-by-row cast blast fired from the eastern end moving to the west and stopped after firing along 500 m of strike length to control vibration.
4. This is followed by another cast blast initiating from the western end travelling in both directions.
5. Once the four rows cast blast is fired, the blast is stopped for several seconds. The back three rows are then initiated with a centre zip going in both directions. The whole blast in this particular instance lasted for 27 seconds to maximise benefits.

Cast blasts initiated from multiple locations using electronic initiators have been successfully used and are proving to be a most effective option for addressing issues relating to coal damage and loss. The veracity of such claims has been shown by surveying the coal edge (after it is exposed) and substantiating coal recoveries with the model predictions (geological or XPAC model). Successful use of creating coal roof surface, gamma logging and well-crafted loading and timing design has been reported earlier (Goswami, Brent and Hain, 2008) and coal recoveries are reproduced in Table 2.

This process produces cleaner coal and plant washing indices such as yield, water and energy usage have also shown improvement. More work is required in this area as every so often coal from various parts of the pit is stockpiled separately, processed and mixed in order to meet the quality demand.

This approach appears to be cumbersome but more and more operations are realising the value this brings when combining it with the D&B process that firstly involves the random drilling of gamma holes that are logged to then produce a load sheet and an initiating design.

## CONCLUSIONS

Maximising the recovery of coal resources remains key to a successful operation. Technological advancement in the field of logging services has led to progress in blasting in recent years. Explosive products, science and technology are playing key roles in lowering the costs of mining by maximising mineral recovery. Improved software is contributing to creating surfaces which are accurate and precise, providing 3D analysis, and locating geological abnormalities. Blasting practitioners are now able to design blasts with flexibility and a greater degree of confidence to benefit the operations. Until recently, blasts were fired lasting milliseconds to about a second. With electronic initiators, which has been extended to almost a minute, allowing stop-start blasts for desired outcome, multipoint and multidirectional initiation, and the ability to combine zip and row-by-row firing. The blasting of multiple layers and benches are now successfully accomplished to bring about economic benefit from the technological advancements.

While staying within the daily mining schedules, benefits such as maximising cast, where desirable, and minimising coal damage and loss can be realised. A systematic method has been developed for mapping coal roof, understanding rock interface above and below coal seam and skilfully incorporating them in the blast design processes to minimise damage/loss. These steps have proven to maximise coal recovery and lower the overall cost of production.

## ACKNOWLEDGEMENTS

My previous employment with Orica over 25 years has provided me with valuable experience and has given me extensive exposure to blasting science and technology which allowed me to further my knowledge in the sphere of D&B and mining operation. I remain grateful. Also, I wish to thank D&B team of Idemitsu Boggabri Coal (IBC) for the blast model runs and output including Figures 3, 4 and 5.

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# Improvement of blast efficiency at Rio Tinto Borates and Lithium Mine in California, USA

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## ABSTRACT

Rock blasting outcomes are often a shared responsibility of both the mine operations group and the mine geotechnical group. From an operational logistics and planning perspective, blasting concerns include optimal fragmentation for subsequent crushing, ensuring the blast is energetic enough to allow easy digging, and understanding the effect of muck from a previous blast on the current blast. From a geotechnical perspective, wall damage and catch bench capacity are the primary concerns. Other concerns for both teams include damage to mine infrastructure by fly rock or blast-induced vibrations, and the stability of open stopes under the pit. This work presents two blast modelling applications (or 'apps'). The basic app instantly gives fragmentation and wall damage predictions for a given generic blast design. The advanced app loads the current pit contours, reads the blast design from a spreadsheet, and calculates the expected vibration magnitude and peak frequency. The apps have been developed specifically for the Rio Tinto Borates and Lithium Mine in California and are used to support blast design between the operations group and geotechnical group. The basic app instantly gives fragmentation and vibration predictions given a blast design by combining the Kuz-Ram fragmentation model with the scaled distance model for vibrations. The advanced app uses the signature hole method. Blasting outcomes at the mine have improved since using the apps.

## INTRODUCTION

Rio Tinto owns and operates the Rio Tinto Borates and Lithium Mine in Southern California. The mine is located in the Mojave Desert, a large sedimentary and evaporite basin. The mine has been active for over 100 years and currently provides most of the United States borates demand. Borates are boron-oxygen compounds that have many industrial and domestic uses including in cleaning products, as a hardening agent in smartphone glass, a fire retardant, and an insecticide. Borates are used in many industrial processes including for manufacturing fibreglass and as a metallurgical flux and a neutron absorber in nuclear reactors. Recently, a significant quantity of lithium was found to occur in the tailings, and the mine is projected to become a major producer of lithium in the coming years.

Rock blasting at the mine is unique because of the abundance of soft arkose and the anisotropic and brittle nature of the kernite ore. The arkose unit is a weakly consolidated siltstone that makes up most of the overburden; the mine staff informally refer to the arkose unit as 'potato dirt'. Figures 1 and 2 show the stratigraphic column along with an annotated photograph of the pit wall. Figures 3 and 4 illustrate the reduced catchment bench capacity and wall damage concerns identified at the mine.

The empirical vibration and fragmentation models used by the basic app are described first, followed by a description of the basic app. Subsequently, a description of the advanced app is given, followed by a summary.

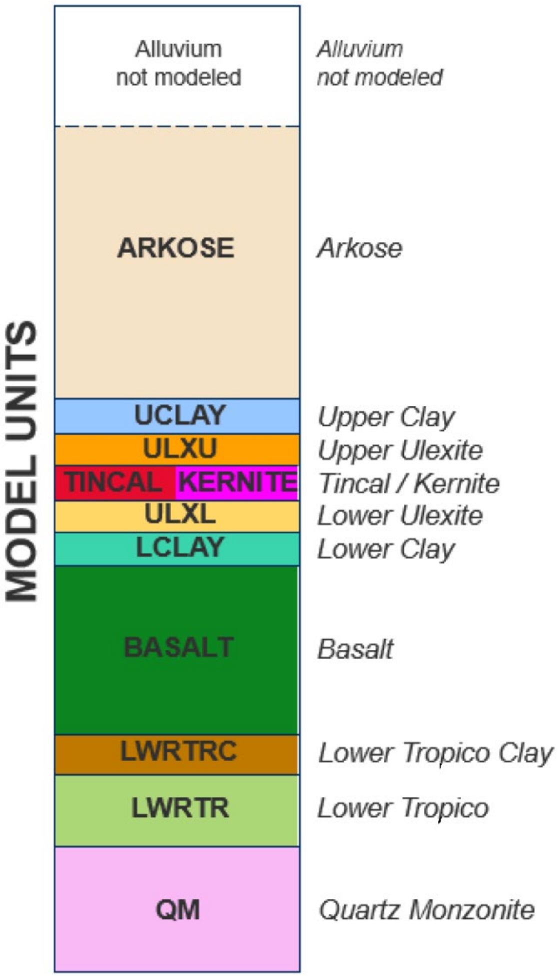


**FIG 1** – Mine stratigraphic column.

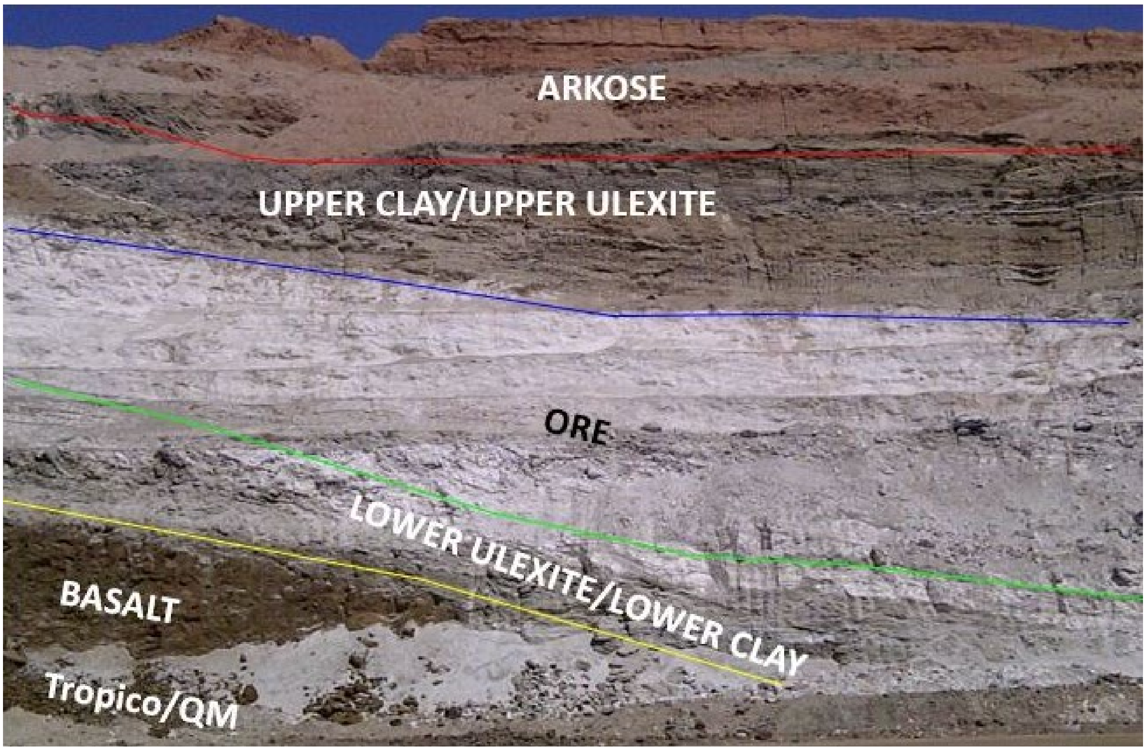


**FIG 2** – Annotated photograph of bench.

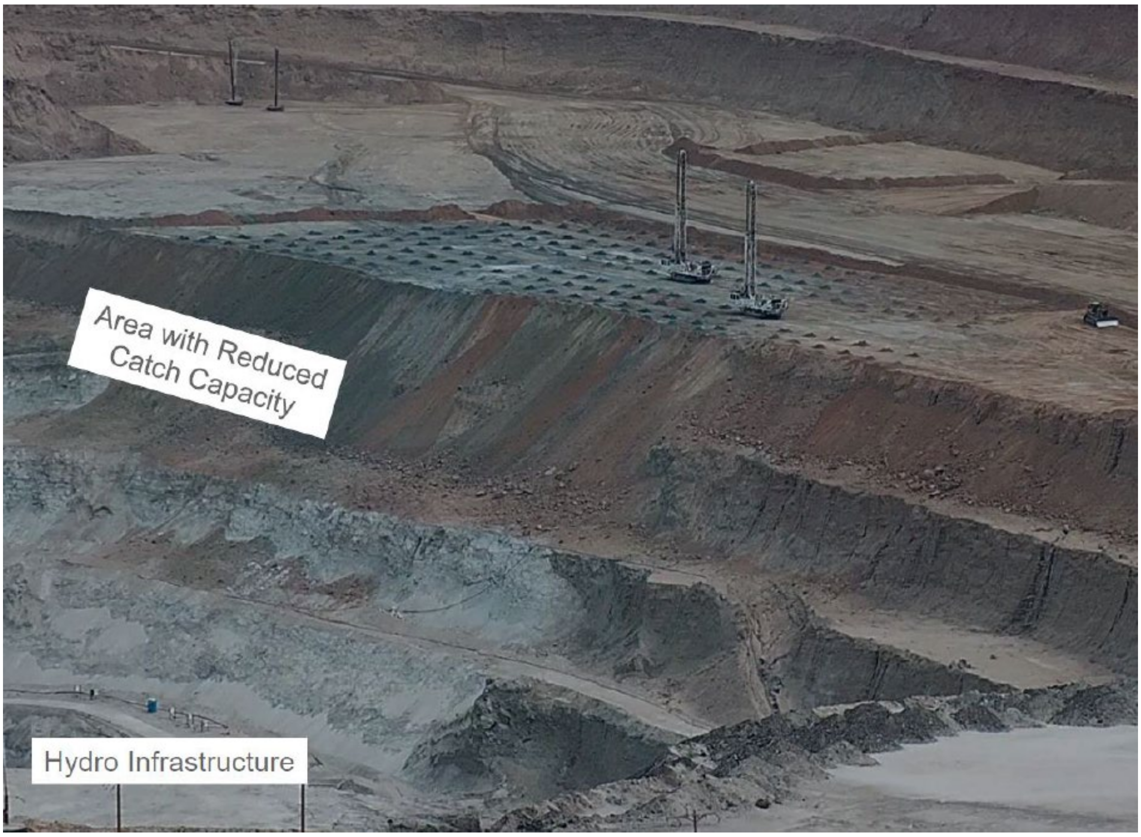


**FIG 3** – Bench spillover as blasting approaches a bench with critical dewatering infrastructure.


**FIG 4** – Blast-induced movement on structures in adjacent benches.

## VIBRATION MONITORING AND MODELLING

The magnitude of blast-induced vibrations is commonly used as an indicator for blast-induced rock damage. At the mine, vibrations from each blast are monitored using the BlastMate hardware platform. The data is grouped by rock type and plotted as Peak Particle Velocity (PPV, the maximum vector magnitude) versus scaled distance. Transforming the distance to a scaled distance is done by dividing the distance from the blast by the square root of the mass of charge per delay. This allows blasts of different designs to be compared. Figure 5 shows a subset of the vibration data gathered at the mine.

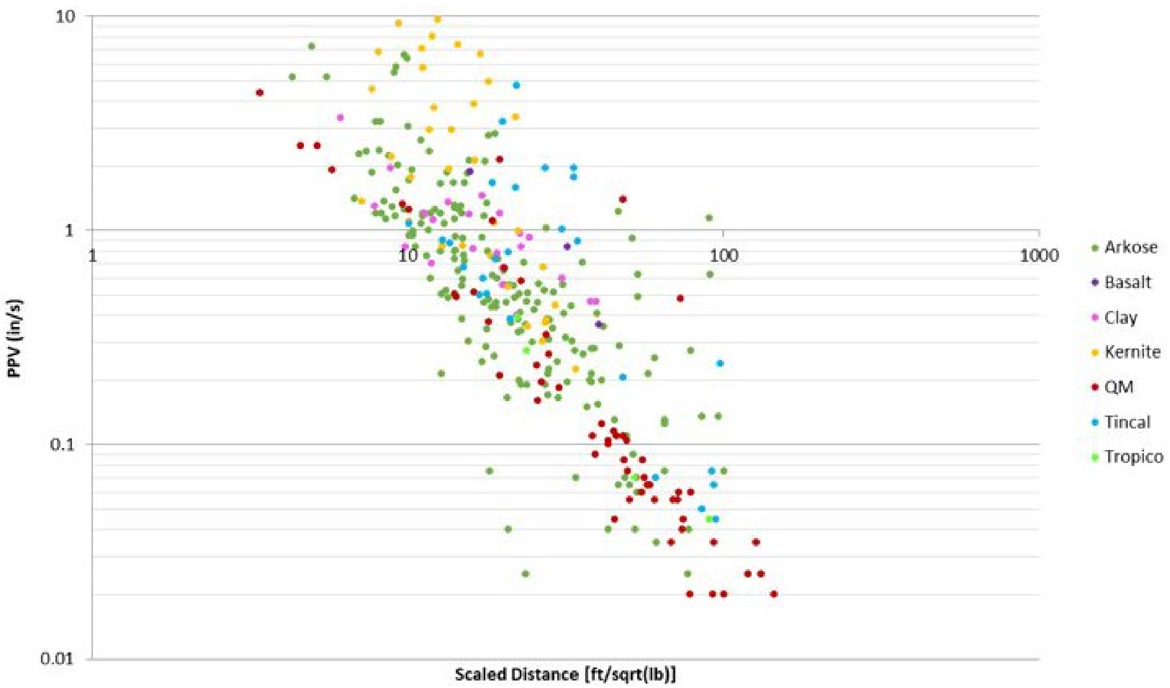


**FIG 5** – Data used to calibrate vibration model.

The scaled empirical distance formula is used to predict the vibration magnitude of future blasts:

$$PPV = A\left(\frac{d}{\sqrt{m}}\right)^{-\beta}$$

where:

| | |
|---|---|
| *d* | is distance from the blast |
| *m* | is the explosive mass |
| *A* and *β* | are constants that are determined from site data (ISEE, 2011) |

In log-log space, this formula gives a straight line that is a best fit to the site data. A separate fit is made using regression analysis for each rock type. Typically, PPV limits are established by rules of thumb or site experience (Persson, Holmberg and Lee, 1994). In the context of rock damage, the stress induced by the vibration can be compared to rock strength to inform a PPV limit:

$$PPV_c = \frac{1}{FoS}\frac{\sigma_t}{\rho C_p}$$

where:

| | |
|---|---|
| $PPV_c$ | is the critical PPV above which rock damage will occur |
| *FoS* | is a factor of safety |
| $\sigma_t$ | is the rock tensile strength |
| $\rho$ | is rock density |
| $C_p$ | is the rock p-wave speed |

The presence of water in the rock has several effects on the resulting blast-induced vibrations. Typically, in wet holes, more energetic emulsion explosives are used. Wet rock has a higher effective density and stiffness, so velocity of detonation (VoD) and explosive performance are higher, and vibration propagation is increased. To account for the effect of water, a separate scaled distance model fit is made for wet and dry rock based on the available data. To account for the effects of blast size and pattern timing on vibration magnitude, groups of holes that detonate within an 8 ms window are considered to occur together. Perfect constructive interference is assumed within an 8 ms window, so the contribution of each hole to the PPV is added together.

## FRAGMENTATION MEASUREMENT AND MODELLING

Adequate fragmentation is essential for cost-effective operation of a large open pit, as insufficient fragmentation can cause delays in mucking and can require oversized reduction before ore is sent to the crusher. A widely accepted approach to predicting fragmentation is the empirical Kuz-Ram model (Cunningham, 2005). Fragmentation is typically determined by photogrammetric analysis, and the results are expressed as cumulative size pass curves. The Kuz-Ram model relates the charge mass, the powder factor, and the explosive strength to a fragmentation curve:

$$R_x = 100\ exp\left[-0.693\left(\frac{x}{x_m}\right)^n\right]$$

$$x_m = AK^{-0.8}Q^{1/6}\left(\frac{115}{RWS}\right)^{19/20}$$

where:

| | |
|---|---|
| $R_x$ | is the per cent of material below size $x$ |
| $K$ | is the powder factor |
| $Q$ | is the charge mass |
| $RWS$ | is explosive Relative Weight Strength |
| $A$ and $n$ | are rock-specific parameters called the rock factor and the uniformity, respectively |

In this work, the uniformity term is taken as unity and the rock factor $A$ is determined for each rock type via regression analysis. Figure 6 shows fragmentation curve data from the mine (orange curve) along with Kuz-Ram model fits (blue curve). The fragmentation can be predicted by using the powder factor, charge mass, and explosive strength from a given blast design in the Kuz-Ram model equation.

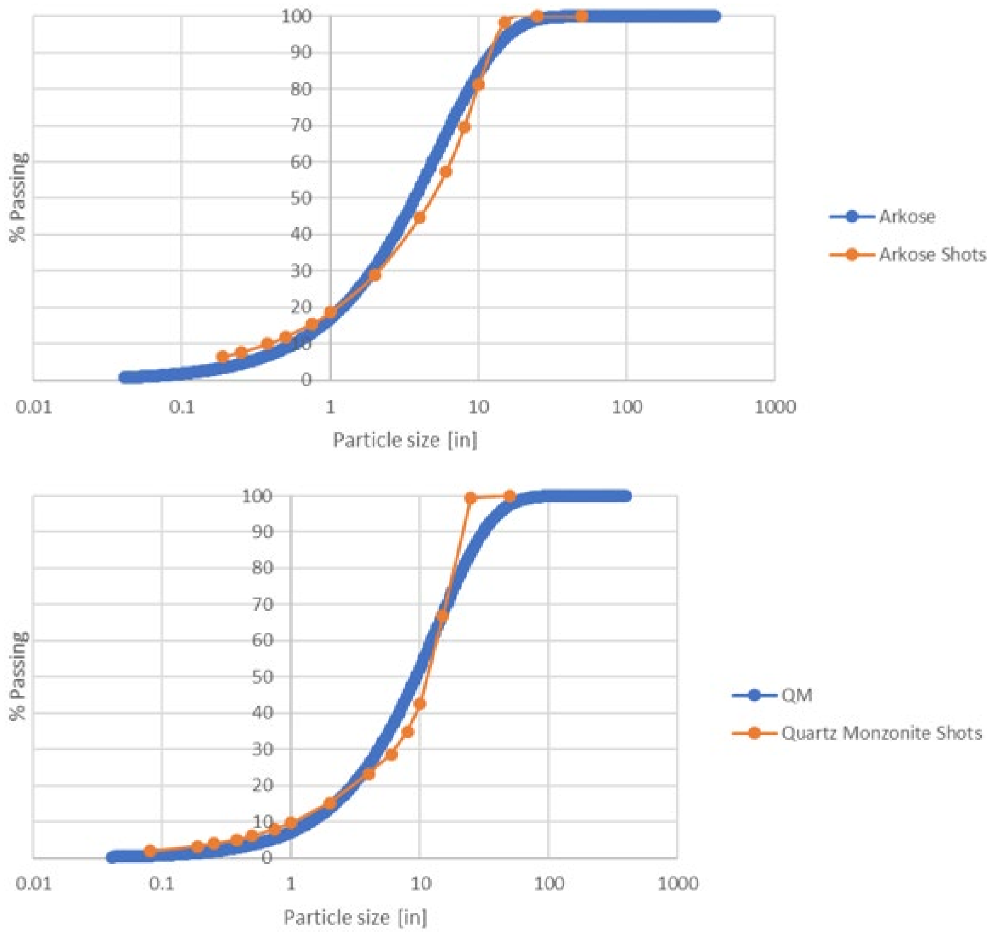


**FIG 6** – Fragmentation data with Kuz-Ram fits.

## BASIC WEB APPLICATION

The vibration and fragmentation model described above are combined into the easy-to-use basic web app. The app takes as input a blast design in terms of explosive type, rock type, hole spacing, charge and stemming length, and a standoff distance. The app instantly predicts fragmentation and vibration, showing the results as charts and 3D plots. Figure 7 shows the app as it appears on a computer or tablet. The blast design is input on the left. The top of the middle column shows the fragmentation curve prediction. Vibration intensity is presented in three ways: as a chart of predicted PPV versus distance, as a chart of PPV versus scaled distance with site data, and in graphical form.

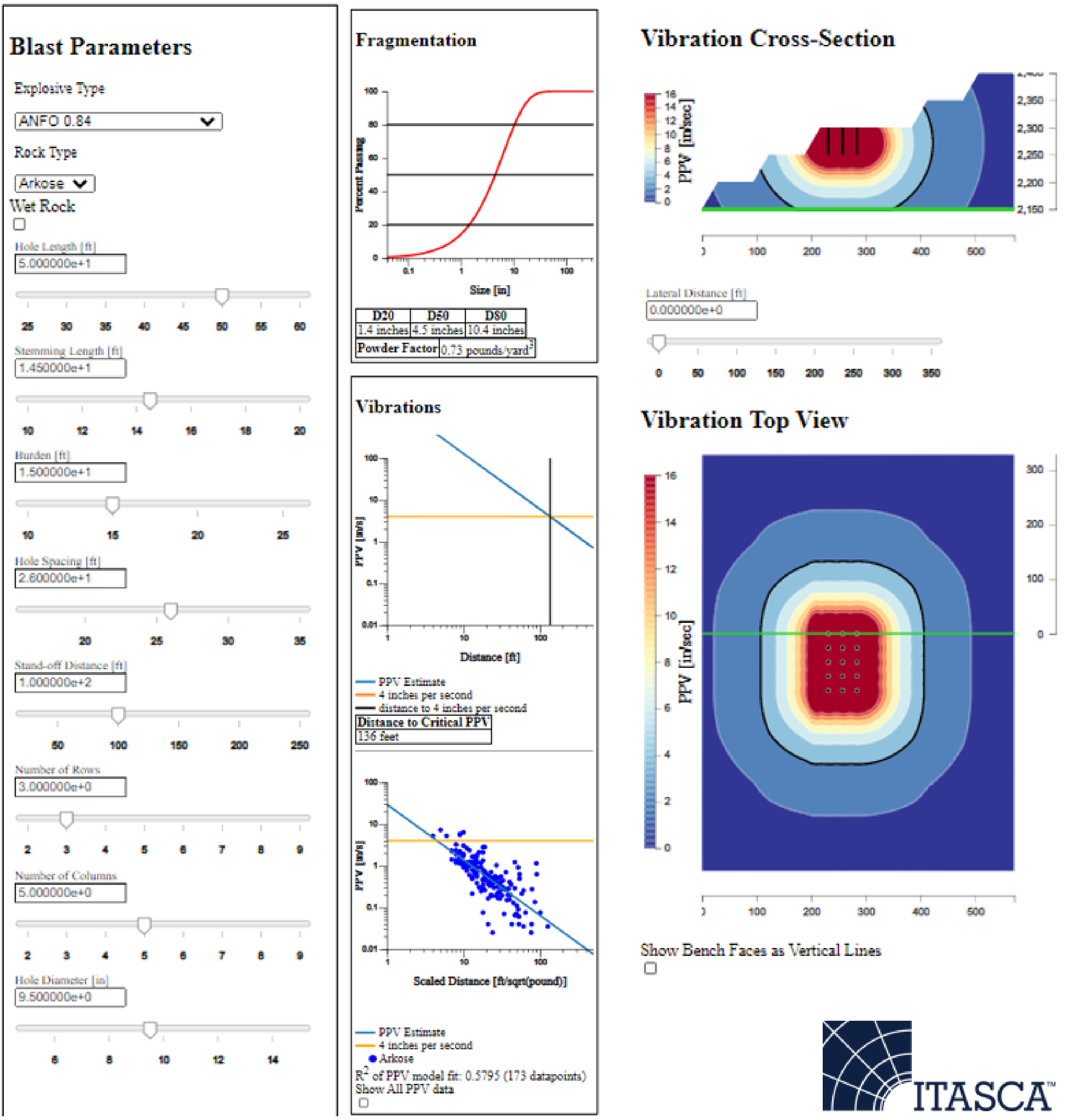


**FIG 7** – The Boron blast app.

A series of familiar and interactive slider inputs are used for the blast design parameters; the plots are updated in real time as the sliders move. This instantaneous feedback leads to a rapid and intuitive understanding of the underlying physical process. The program is written in Javascript and is a standalone web application that runs on computers, tablets, and phones. The D3.js library is used for visualisation and math.js is used for array manipulation. All calculations are performed on the client side, allowing the app to be used when an internet connection is unavailable.

## ADVANCED WEB APPLICATION

The basic web app is limited to a standard bench geometry, a simple rectangular blast, and a simple timing pattern. Blasts at the mine do not always conform to this idealised situation, so an advanced app was developed to consider the effects of pit shape and the true blast designs. The advanced app takes as input the latest pit contour line breaks as a DXF file, the lithology models, and a standard spreadsheet describing the blast design (hole locations, charge mass, timing etc). The pit line contours are transformed into a surface mesh of the pit shell in the region within 1000 feet of the blastholes. The Boron lithology model is used with the blasthole locations to determine the lithology of the blast.

The signature hole method is used to predict the blast-induced waveform of a multi-hole blast by time shifting and superposition of a waveform measured from a single hole (signature) test shot (Yang and Scovira, 2010). Compared to the scaled distance method, the signature hole method more accurately accounts for hole timing and geometric effects. The model was calibrated with signature hole data shot in the Boron mine quartz monzonite unit. The full waveform at each point around the blast is predicted and the peak frequency is determined via a Fourier transform.

Using the advanced web app involves dragging and dropping the required data into a web browser, and all computations are done in the cloud. It takes about five minutes to predict PPV and peak frequency for a typical 100-hole blast. The results are rendered, in the browser, in a 3D view that can be zoomed and rotated (see Figures 8 and 9).

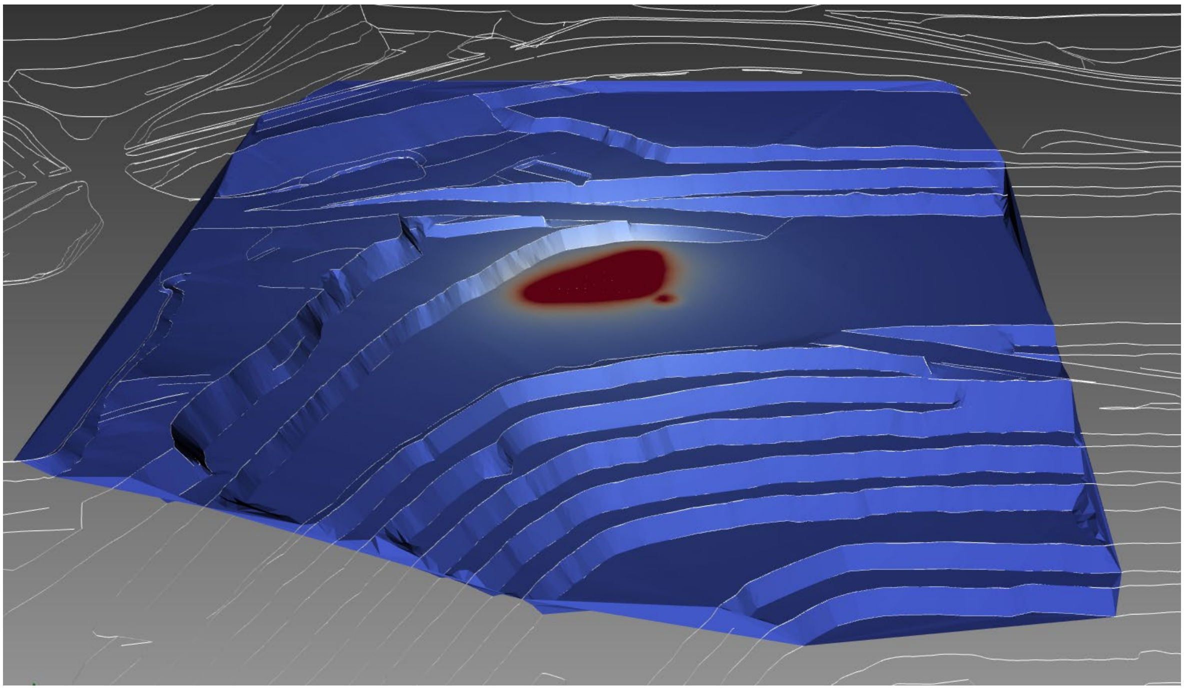

**FIG 8** – Predictions of PPV from the advanced app. Red corresponds to a PPV of 25 inches/second or greater.

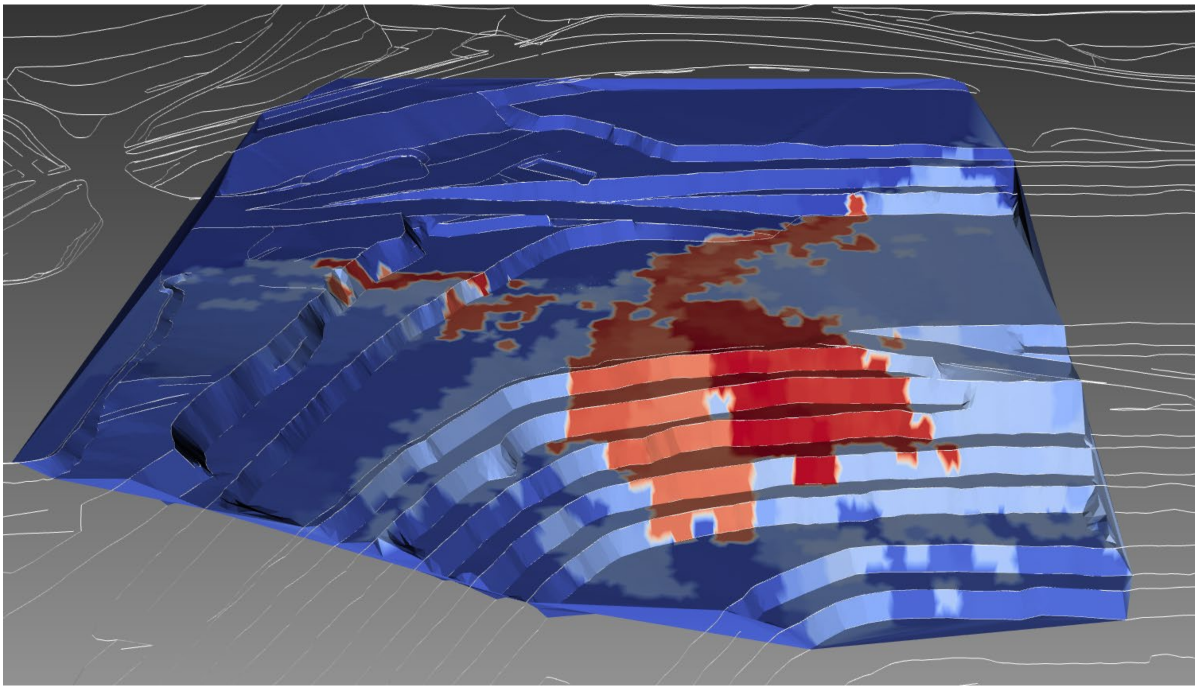

**FIG 9** – Predictions of the peak frequency of the transverse blast-induced waves. Red indicates a peak frequency of 17 Hz.

## SUMMARY

Two blast modelling applications have been created for use at the Boron mine. The basic application gives instant answers for a generic blast, and the advanced application takes longer but gives a more accurate result that considers more of the pit shape and blast specifics. Using the apps has led to a greater understanding of how blast design influences blast outcomes. The app has helped facilitate technical discussion between the operations and geotechnical groups at the mine. With this approach, blasting outcomes have improved. Figure 10 shows the improved final wall and catch berm performance at Boron.

**FIG 10** – Improved blast outcomes at Boron.

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# Mine to mill study – Bond Work index as factor for intrusive and non-intrusive rock type

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## ABSTRACT

This study data has been collected on a three-month period from November 2020 to January 2021 and based on three different pits: Vindaloo Main, Vindaloo Central and Bouere. Both pits having specific geotechnical and geological properties. Vindaloo Main, VIM, and Central, VIC, have the same mineral composition as volcanic tuff with felsic intrusion or monzodiorite, both rock mass is fractured with small joint. Geotechnically the uniaxial compressive strength UCS between 200 to 300 Mpa, Young's Modulus 110 Gpa and Poisson's ratio between 0.1 to 0.2. Bouere pit is basically made of volcanic tuff stock works as intrusive, UCS between 200 to 300 Mpa, Young's Modulus 103 Gpa and Poisson's ratio between 0.2 to 0.3.

The first step of this study was to collect various data from mining operations around the drilling and blasting inputs. This was based around the different scenario of drilling parameters and the outcomes on the blasting, and how they impact load and haul operations and productivities of excavators from the heave to the last flitch. In most cases surface photos of the blast do not represent the overall fragmentation in the blast volume. Photos of the different flitches will not show only the effect of blast, as the loading operation generating more fragmentation by applying weight on the muck pile. Then the record of the different material uses for feeding. The last part of the analysis is the physical data from process plant such as particle size, throughput rate, both for the crusher and the mill, per pit and material and the specific Bond Work index (BWI). Fine particle quantity production is also a key point as is one direct indicator of milling performance. In overall, making link to BWI can influence at each stage of the mining to the processing operation.

Drill and blast constitute a high cost in hard rock mining and trying to reduce these costs without an overview of all the integrated processes and context could end up by cost relocating to other activities in the process. However, the cost associated to size reduction builds exponentially as the particle size becomes smaller.

## INTRODUCTION

The scope of work is analysing data from two different periods in term of drill and blast parameters and how it is impacted the Mining to Processing activities. The results are used to evaluate how efficient the BWI of different material feeding into the processing can be optimised. The essay therefore recommends some sustainable operational changes to optimise mill throughput for an optimal cost reduction of total project costs. This is however with a change in material type be a continuous and dynamic process. By this approach, the aim was to choose the best drill and blast parameters to produce a homogenous blasted material with top size material reduction and fine quantity increasement, which will improve mill throughput. By considering the geology of both pits as well propose different ore bleeding for feeding based on the BWI.

Hounde is located approximately 250 km south-west of Ouagadougou, the capital city of Burkina Faso. Hounde gold mine is centred on Latitude 11°24'57.6"N and Longitude 3°31'51.6"W. The Mineral Resource estimate for the Hounde gold mine as of 31 December 2019 reports:

- Measured Mineral Resources totalling 1.7 Mt grading 1.75 g/t for contained metal of 96 koz.
- Indicated Mineral Resources totalling 58.6 Mt grading 2.07 g/t for contained metal of 3797 kozAu.
- Inferred Mineral Resources totalling 6.9 Mt grading 2.07 g/t for contained metal of 456 kozAu.

Hounde Gold Operation HGO mine is by the Hounde greenstone belt, which comprises andesites, basalts with acidic volcanic intercalations, gabbros, greywacketo argillaceous sediments and banded chert. Gold mineralisation is typically hosted by varying size and orientation, quartz-albite-carbonate-pyrite veins, with associated disseminated pyrite. The belt is located in the western region of Burkina Faso. The belt extends approximately 330 km along strike and is up to 60 km wide. The belt is orientated north-north-east to south-south-west. The Vindaloo zone comprises a group of closely spaced and subparallel mineralised deposits that extend over a strike length of 7 km. It is composed of a series of north-north-east trending and steeply-dipping to subvertical mineralised deposits hosted by intermediate volcanic rocks (coarse andesitic polymictic debris flows, massive flows, and pyroclastic and lapilli tuff), sedimentary rocks (interlayered greywacke and graphitic argillite) intruded by concordant (sill-like) bodies of monzodiorite. Bouere is hosted in a mafic to intermediate volcanic sequence, comprised of fine-grained tuffs and pyroclastic andesitic flows and breccia interlayered with more massive basaltic and andesitic flows. As is common in West Africa, the mineralisation has been weathered significantly. As such, four weathering domains are noted, namely overburden, saprolite, transition, and fresh.

Several studies and metallurgical extraction test work programs have been performed on samples from the Hounde Gold Mine deposits. Comminution test work and current practice indicates that primary ores require moderate grinding energy and have moderate abrasivity; however, the ores are highly competent and display a high resistance to impact breakage. Grind sensitivity test work on the primary composites indicated that lower residue grades, faster leaching rates and higher gold extractions are achieved with increasing fineness of grind (Endeavour Mining Corporation, 2019).

The HGO process plant have fed by front end loader to an apron feeder which deliver ore to a vibrating grizzly screen. Grizzly oversize is crushed in a jaw crusher which is the primary crusher, then grizzly undersize and crushed oversized report to the surge bin feed conveyor. Primary crushing product is conveyed to the surge bin. Excess ore from the surge bin overflow to a dead stockpile. Feed to the SAG mill is drawn from the surge bin by an apron feeder, which is discharge onto the conveyor feeding the SAG mill.

SAG mill discharge screened on a single deck vibrating screen with screen oversize (pebbles) conveyed to the pebble crushing circuit for size reduction, as secondary crushing. Crushed pebbles discharge onto the SAG mill feed conveyor. The SAG mill discharge screen undersize gravitate to the common mill discharge hopper with the ball mill product.

The ball mill operates in closed circuit with hydrocyclones with cyclone underflow feeding the ball mill for size reduction.

A portion of the combined SAG and ball mill discharge streams pumped from the mill discharge hopper to two gravity gold recovery circuits each incorporating a feed preparation screen and a batch centrifugal gravity concentrator. Gravity concentrate processed using intense cyanidation.

Milling circuit product (trash screen undersize) prior to leaching/adsorption, compose of six tanks of carbon-in-leach CIL. Gold recovered by elution process.

## INSTRUMENT AND METHOD

### Instrument

The main tools used for this study are: Surpac, version 6.7 (by GEOVIA, Dassault Systems) for pattern design considering ore dig plan. Random belt cuts for detailed analysis particularly on fine. Bond Work index test made from sampling for different pits. Excel 365, version 2101 (by Microsoft Office, Microsoft Corporation) to generate a database, analyse and graphically interpret results.

### Method

Ore shot pattern design follow procedure to minimise ore dilution and maximise fragmentation. Achieving that requirement, shots are blasted with burden oriented to free face when ore direction permit it.

The Bond Work index is defined as the kilowatt hour per short ton required to break from infinite size to a product size of 80 per cent passing 100 µm. The standard test is done by putting a certain volume sample of material with the balls to the ball mill and ground initially at 100 revolutions. The ground sample was screened with the test sieve and the undersize sample was weighed and fresh unsegregated feed was added to the oversize to bring its weight back to that of original charge. The numbers of revolution required was calculated from the results of the previous period to produce sieve undersize equal to the 1/3.5 of the total charge in the mill. The grinding period cycles were continued until the net grams of sieve undersize produce per mill revolution reaches equilibrium. Then the undersize product and circulating load was screen analysed and the average of the last three net grams per revolution Gbp is the ball mill grindability. The test mesh used was 100 mesh 85 µm. The Bond Work index Wi was calculated from the following equation:

$$Wi = 44.5/(P1)0.23 \times (Gbp)0.82\ [10/(P80)0,5–10/(F80)0.5]$$

Where:

| | |
|---|---|
| F80 | is the size in µm, at which 80 per cent of the new feed to ball mill passes |
| P80 | is the size in µm, at which 80 per cent of the last cycle sieve undersize product passes |
| P1 | is the opening in µm, of the sieve size tested |

Then the data collected was sorted in different excel sheets for analysis. One for the productivity of the excavator per location and material type. Second one mentioning blending record per day by different type of material and percentage with average calculation per month. On the processing side, the daily records of the particle size at 80 per cent passing size P80.

Physical data such as the throughput rate and ball consumption per ton reduce records. Data classified for the primary and secondary crushing as well for the milling. Bond Work index associated to different material has also been captured. Total and average data has been calculated at the end of each month.

At Hounde gold operation it is a modified version of the Bruce and Berry work index procedure. It is a relatively quick method used to develop work index values that approximate the BWI within 10 per cent. The Bruce and Berry technique requires that the reference and unknown samples have approximatively the save particle size distribution and cumulative weight passing 80 per cent. The modified version of this procedure assumes that the minimum relative work index value calculated at four different grinding times or work inputs represents the closest approximation of the Bond Work index. The modified procedure places less emphasis on matching the feed particle size distribution of the reference and unknown samples. Both procedures depend on the reference and unknown samples having work indices in approximately the same range.

The reference is a sample of vindaloo fresh ore with an estimated work index in average of 15.9 KWh per metric ton at 100 mesh. The samples are staged crushed to minus 2 mm and split into 1.070 kg charges. The crushed feed samples are screened and the cumulative passing 80 per cent F80 and modulus are determined. The charges are then grinding in ball mill for 10, 20, 30 and 40 minutes and screened at 53, 75, 106, 150, 212, 300, 600, 1000 and 2000 µm. The cumulative per cent passing is plotted against particle size in microns on a Roslin-Rammler plot to determine the cumulative P80 and the modulus for each power input or grind time.

It is important to mention that oxide from Kari pit compose of quartz veins hosted by meta-andesite ore has been fed during this period.

> *The procedure for the work index test bases the $BW_i$ value on the calculation of* new *fines generated in the test. This means that the fraction of fines in the feed should not influence the test result significantly, if at all. For example, for a sample with 20% of -300 µm material in the feed, if this is not scalped out of the fresh feed, then the mill charge, at 250% circulating load will contain 0.2/3.5 or 5.7% of -300 µm in the mill charge compared with 0% for a scalped fresh feed, at a closing screen of 300 µm. This should not have a great influence on the production of new fines unless the test was carried out in a wet environment and the fines contained a high percentage of clays to affect the viscosity of the grind environment. (Gupta and Yan, 2016)*

## RESULTS AND DISCUSSION

Drill and blast parameters change in December 2020 as shown in Table 1.

**TABLE 1**

Drill and blast parameters.

| | | Bouere | | Vindaloo Main | | Vindaloo Central | |
|---|---|---|---|---|---|---|---|
| | | Fresh ore | | Fresh ore | | Fresh ore | |
| **Items** | **Units** | **Previous** | **Actual** | **Previous** | **Actual** | **Previous** | **Actual** |
| Hole diameter | (mm) | 115 | 115 | 115 | 115 | 115 | 115 |
| Hole dip | (degrees) | 90 | 90 | 90 | 90 | 90 | 90 |
| Bench height | (m) | 9 | 9 | 9 | 9 | 9 | 9 |
| Subdrill | (m) | 1 | 0.8 | 1 | 0.8 | 1 | 0.8 |
| Hole length | (m) | 10 | 10 | 10 | 10 | 10 | 10 |
| Stem length/ uncharged length | (m) | 2.7 | 2.7 | 2.7 | 2.7 | 2.7 | 2.7 |
| Charged length | (m) | 7.30 | 7.30 | 7.3 | 7.30 | 7.3 | 7.30 |
| Burden -design | (m) | 3 | 3.0 | 2.8 | 3.0 | 2.8 | 3.0 |
| Spacing design | (m) | 3 | 3.3 | 2.9 | 3.3 | 2.9 | 3.3 |
| Total charge weight | (kg) | 90.2 | 87.8 | 90.2 | 87.8 | 90.2 | 87.8 |
| Tonnes | (t) | 221.94 | 244.13 | 200.24 | 244.13 | 200.24 | 244.13 |
| Volume | ($m^3$) | 81.00 | 89.1 | 73.08 | 89 | 73.08 | 89 |
| Tonnes/drilled metre | (t/m) | 22.19 | 24.9 | 20.02 | 24.9 | 20.02 | 24.9 |
| Powder factor | ($kg/m^3$) | 1.11 | 0.98 | 1.23 | 0.98 | 1.23 | 0.98 |
| Powder factor | (kg/t) | 0.41 | 0.36 | 0.45 | 0.36 | 0.45 | 0.36 |

The major change was on drilling, the mesh of drilling pattern opens up and the subdrill length reduce. Adjustment led to a cost saving about 10 per cent and 22 per cent respectively for Bouere and both Vindaloo Main and Central pit, overall cost reduces by 16 per cent. The first observation made is that the heave fragmentation contains a little bit more oversize rock. Also, the rock breaker hours increase. A lot of factors apply to the productivity; some of them are sometimes more operational such as truck availability, loading environment and affect the productivity.

HGO fleet of excavators have three Komatsu PC 2000 and four PC 1250 mining three metre flitch on the fresh rock. The productivity of both excavators decreases from change point, nonetheless by analysing the flitch-by-flitch productivity trend remain the same, meaning increasing by going to the last flitch.

Ore blending proportion vary from a month to another but basically stay in the target per material type. Front end loaders feed the crusher bin following the number of buckets per material per cycle. Reconciliation at the end of the month between geology and process plant team give accurate results such as the quantity fed per grade bin, per pit or per material type. In this case, the percentage of material per pit and weathering is the more appropriate data to measure the influence of the rock properties on the processing.

Result for the period is summarised in Table 2.

**TABLE 2**

Blend per pit and weathering.

| Pits | Blend | | Nov-20 | Dec-20 | Jan-21 | BWI (KWh/t) |
|---|---|---|---|---|---|---|
| | | | Actual | Actual | Actual | Lab test |
| VIM | Oxide | % | | | | 15.2 |
| | Transition | % | | | | |
| | Fresh | % | 46% | 4% | 40% | |
| | Pf | kg/t | 0.45 | 0.36 | 0.36 | |
| VIC | Oxide | % | | | | 17.1 |
| | Transition | % | | | | |
| | Fresh | % | 9% | 44% | 15% | |
| | Pf | kg/t | 0.45 | 0.36 | 0.36 | |
| Bouere | Oxide | % | | | | 18.1 |
| | Transition | % | | | | |
| | Fresh | % | 5% | 8% | 12% | |
| | Pf | kg/t | 0.41 | 0.36 | 0.36 | |
| Kari | Oxide | % | 40% | 45% | 34% | 14.8 |
| | Transition | % | | | | |
| | Fresh | % | | | | |

Processing physicals also calculate at the end of the month as shown in Table 3.

**TABLE 3**

Blend per pit and weathering.

| Processing physicals | | Nov-20 | Dec-20 | Jan-21 |
|---|---|---|---|---|
| | | Actual | Actual | Actual |
| Total crushed (wet tons) | t | 391 740 | 397 595 | 428 135 |
| Crushing rate | t/h | 618.20 | 642.61 | 652.88 |
| total crusher power | kWh/t | 86 001 | 84 400 | 88 440 |
| total processed (dry tons) | t | 367 897 | 394 343 | 365 750 |
| plant feed rate | t/h | 555.00 | 555.39 | 555.39 |
| total mill power | kWh/t | 18.66 | 18.68 | 18.68 |
| ball consumption | kg/t | 0.71 | 0.86 | 0.86 |
| P80 | µm | 99.09 | 96.47 | 97.65 |

The crushed and milled tons material increasing during the period. Crusher and surge bin feeder replacement early December could be the main root of this results. So, it is difficult to link this performance to the mining operation. The crusher power consumption drops from November to December but gone up January, this it is explain by the feeding of the month contain a lot of heave material, where there is about 80 per cent of the oversize material. The variance between these months in terms of material fed per pit is 6 per cent. November material has been blasted before the parameters change. This variation translates into 2439 KWh/t or 3 per cent consume more and highest than the base BWI. The lowest power consumption was in December when feeding more

VIC material. The mill power consumption stays relatively the same over the period increase of 2 per cent since the change. The ball consumption also increases for 21 per cent that means that the size of the feed is harder than before. Only the P80 result show decrease trend.

## CONCLUSIONS

Overall, the change in drill and blast save cost in the mining but per the quantity assessment for this period both mining and process key performance measurement are decreasing.

BWI test value is less than the actual energy used by the process. Using more energy to break the rock in the pit will have a positive effect on the mill power consumption. Some trial must be done to confirm that statement as well as a cost study.

Analyse the granulometry at the end of the crushing process could give more precision about the fine generate by the blast.


## ACKNOWLEDGEMENTS

Many thanks to Endeavour Mining Corporation for this opportunity, especially VP mining for the support during the redaction of the paper. Thanks to the process plant team for their support; and the mining engineering team, especially the long-term planner engineer for the availability and the guidance.


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# Blast design and execution drives mine site profits

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## ABSTRACT

Optimisation of Blast Design and Execution must be done right the first time. It is too late after the ore is broken.

Consequences of poor design and execution negatively affect productivity from load and haul through the primary crusher to processing and hence reduce mine site profits.

Harnessing and utilising ‘big data’ from geology, geometallurgy, drill hole location, powder loading, blast timing, blast movement and fragmentation to close the loop in blast design and optimise the blast increases productivity across the entire value chain from Mine to Mill.

As the internet of things and artificial intelligence increases the quantity of data available, being able to capture and utilise this data to advantage is a challenge. Recent advances in software and hardware together with the increased power of cloud computing allows engineers to close the blasting loop to increase mine site productivity.

Numerous case studies over the years detail significant gains from implementing the innovative technologies available to Mine to Mill projects. The opportunity is available for all members of the multi-discipline teams at each mine site to actively ensure productivity improvements are realised and maintained not only in their area, or silo of responsibility but across the entire mine site.

This paper refers to recent case studies which outline how significant improvements in productivity are available to all mining operations to increase mine site profitability and dividends to shareholders.

## INTRODUCTION

Comminution is the process of particle size reduction to liberate valuable minerals from their host ore. The blast in the mine is the first step in the comminution chain which ends with a particle size small enough to liberate the valuable mineral for recovery.

Comminution is generally the single largest consumer of energy on a mine site.

Drill and blast impact the entire mining process, from mining equipment efficiency through crushing and grinding circuit performance, to recoveries and final product quality. Costs and energy usage increase throughout the comminution process. Optimising the blasting process can pay huge dividends downstream, reducing costs and energy consumption.

An optimum blast not only optimises the cost of that blast, but it also improves the total cost profile of the entire Mine to Mill chain.

Optimising blast fragmentation can lead to:

- Increasing dig rates, truckload and utilisation of mobile equipment utilisation by reducing maintenance in the load and haul process.
- Reducing (or better still avoiding) blockages at the primary crusher.
- Increasing throughput and process efficiency of crushing and grinding:
  - For mines with semi-autogenous grinding (SAG), increasing fines in ROM ore can increase SAG mill throughput from 8–30 per cent, Cameron, Drinkwater and Pease (2017).
  - Increase in iron ore revenue by increasing lump to fines ratio.

Improved dilution tracking ensures all ore is sent to the plant versus the waste dump or waste being processed for no economic gain. Not only does this improve ore recovery, but it also impacts processing plants downstream where vital blending can be required to ensure the plant operates efficiently.

Add high-accuracy machine-guidance drills and shovels with state-of-the-art positioning capabilities, and you wield a profoundly profitable trifecta: yield, fragmentation, and dilution.

This trinity improves a mine's entire cost profile: reduced blasting costs, reduced rework, improved digability, increased ore recovery, less crushing, higher t/h through the plant, and improved efficiency for ore processing.

## DRIVERS FOR IMPROVEMENT IN MINE TO MILL PROCESS

Measuring blast movement leads to:

- reduction in misclassification of material which increases ore yield
- improving grades of ore fed to mill.

Measuring and managing fragmentation through Mine to Mill leads to:

- elimination of oversize rocks which block the primary crusher
- increased shovel digability and truckloads
- automatic optimisation of crusher closed side settings to reduce downtime
- early detection of oversize in product streams to indicate screen failure and screen blinding to avoid re-screening batches of product.

Optimum fragmentation is:

Not too coarse… not too fine… just right!!

To determine optimum fragmentation, the Mine General Manager coordinates fragmentation discussions across the operational silos:

- blast design advise impact of geotechnical data and powder factor
- load and haul advise impact on shovel digability and truckload
- primary crusher advise size which blocks crusher
- process use simulation to confirm particle size distribution to optimise each unit operation in crushing and grinding:
    - each unit operation has an optimum feed size and a customer with an optimum feed size
    - SAG mill requires bi-modal particle size distribution: lump for autogenous grinding media and fines smaller than aperture of discharge grate to increase throughput
    - iron ore requires minimum fines to increase lump to fines ratio.

To achieve optimum fragmentation:

- use modern blast design software
- layout blast design for optimum fragmentation
- review design and outcomes for bench above
- predict fragmentation
- blast
- measure fragmentation
- compare measured with predicted fragmentation
- modify blast design to achieve optimum fragmentation

- ideally, automate these steps.

## BLASTING

Comminution is an energy-intensive process that accounts for a significant proportion of mine site costs. Effective blasting improves comminution and reduces mine site costs, CEEC (2016).

Blast Engineers measure fragmentation in ROM ore for feedback to manage energy input in the design of blast patterns to optimise the ROM product size for feed into the primary crusher.

### Blast design

Effective blast design includes an understanding of:

- Rock structure and rock mechanics where unconfined compressive strength (UCS) and Point Load Index data are complemented by measurement of fracture zones.
- Blasthole crushing zone which is a function of hole diameter and explosive energy and produces the fines required to increase SAG mill throughput.
- Blasthole spacing to avoid oversize rocks which block the primary crusher.
- Detonation timing and sequences which impact on ore movement.
- Production tonnage.
- Downstream process requirements.

In modern blast design and simulation software, fragmentation models predict the blast's particle size distribution (PSD) for different rock masses, blasthole geometry, and explosive parameters. Predicted size distributions are used as inputs in the simulation of downstream processes to determine the blast design required to optimise fragmentation in the muck pile to optimise process throughput. Two fragmentation models used extensively worldwide to predict fragmentation in the blast are Kuz-Ram and Swebrec, which are described respectively in detail by Cunningham (2005) and Ouchterlony (2005).

When a blasthole is detonated, rock breakage occurs in two different stress regions, compressive and tensile:

- Compressive stress waves form a 'crushing zone' adjacent to the blasthole, which creates fine particles.
- Tensile stress waves propagate the 'cracked zone', outside the crush zone and between blastholes to create coarse particles. The size of these particles is influenced by the characteristic rock fracture zones and rock mass.

Figure 1 which illustrates the formation of a crushing zone, fracture zone and fragment formation zone around the blasthole was developed by Kabwe (2018).

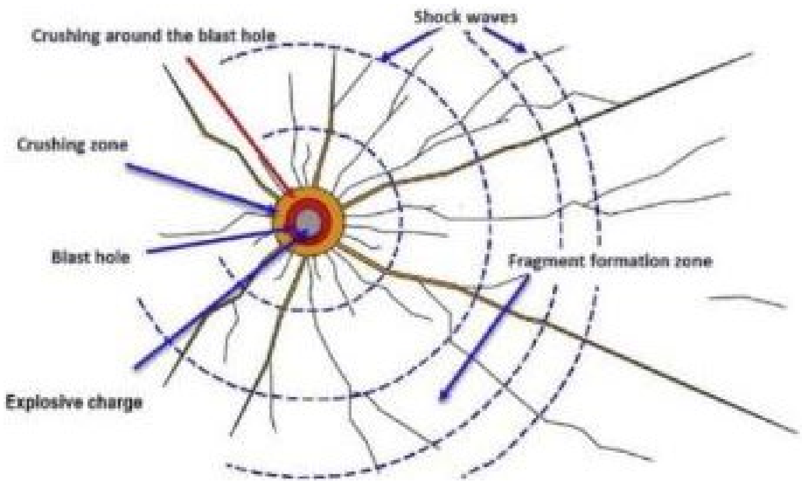


**FIG 1** – Rock breakage mechanisms near the blasthole.

When the actual measure of fragmentation does not match predictions, blast engineers modify their blast designs to manage selected parameters to meet the optimum PSD, therefore tuning the model to their specific rock type and mine parameters.

The blast engineer's options to control the ROM PSD are generally to manage blasthole geometry, explosive type and quantity. It can be almost impossible to change hole diameter in the short-term due to the investment in drilling equipment and explosive supply. Likewise, bench height cannot be changed due to the impacts this would have on mining schedules. Consequently, blast pattern parameters (hole spacing, depth, diameter, hole angle), explosives (type, quantity, and position within the hole), and blast initiation (sequence and timing) are the available variables in blast design.

Conditions in blasting that the blast engineer has control over influence aspects of blast movement in different ways, which at times can be in counterbalance with design targets such as PSD.

Initiation changes directly influence movement geometries and trends, sending the material in favourable directions and subjecting that material to uniformity in its movement geometries.

Explosive changes have direct impacts on movement magnitudes and directly impact the variability of those magnitudes.

Pattern parameters, such as burden and spacing, similarly link into movement geometries as initiation design does, resulting in increased or decreased uniformity of movement.

## HOLISTIC APPROACH TO THE DRILL AND BLAST PROCESS

A holistic approach to drill and blast provides solutions to improve yield, fragmentation and misclassification is shown in Figure 2.


**FIG 2** – Holistic drill and blast process.

Incorporating blast design software, high-precision drilling, sound QA and QC in blast execution, and post-blast analytics empowers blast engineers to optimise fragmentation.

The ability to tailor fragmentation outcomes and to minimise misclassification through blast movement monitoring are highly sought. In effect, tailoring acts as a form of preconcentration by mining the ore delineations in the correct post-blast locations and meeting product chemical specifications.

Realising these benefits needs a focus outside conventional operation silos as a successful Mine to Mill project requires a multi-discipline approach across all the silos in the value chain.

## MEASURING BLAST MOVEMENT

Controlling ore loss and dilution is critical for mining operations. Mining *in situ* dig blocks or relying on a standard template or surface indicators as an input to 'inferred' movement, can result in significant lost revenue.

If ore movement is not accounted for, up to 25 per cent of the mine's total recoverable product could end up as waste, translating into tens of millions of dollars in lost revenue per annum.

A rigorous and proven blast monitoring system directly measures blast movement. Directional transmitters placed within the blast volume prior to blasting are located after the blast with a detector and the data processed by purpose-designed software. The system is suitable for all open cut mines and is designed for routine use by site personnel.

- Each blast movement monitor (BMM) is activated, programmed, and installed before blasting.
- A detector locates each BMM after the blast. Data is downloaded to software that calculates the movement vectors then summarises and archives the results.
- Proprietary algorithms derived from years of research are used to adjust the ore boundaries according to the measured movement of the blast, enabling the most accurate ore control.
- Within the body of the blast, surface movement represents a small portion of the bulk movement which is greatest around mid-bench. The result is a D-shape profile like Figure 3. As a result of the differential movement between the surface and rock at depth, ore loss and dilution can still be significant if surface observations given by blast vector indicators (BVIs), such as polypipe, are used to translate dig boundaries.

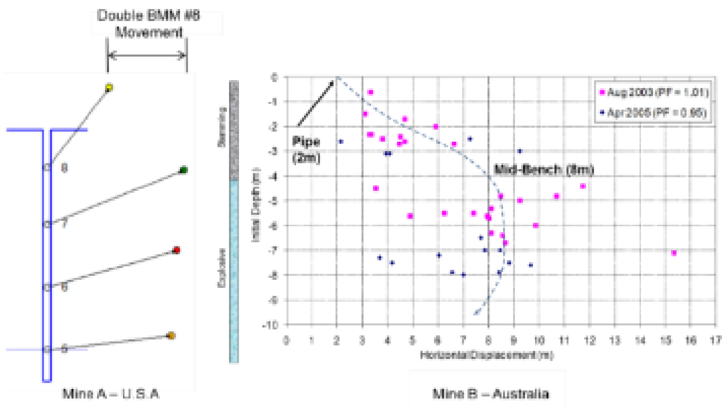


**FIG 3** – Horizontal movement versus installation depth.

In addition to the variance between surface movement and movement at depth, movement measured at similar depths throughout the blast can vary up to 50 per cent around the mean, making modelling or creation of a template far less successful than measuring when trying to maximise ore recovery.

## MEASURING FRAGMENTATION AFTER THE BLAST

Image-based measurement systems which automatically measure particle size in the mine and throughout the plant, are shown in Figure 4. These can be installed on the loading equipment and at the primary crusher to understand fragmentation in the pit and at the primary crusher. Knowledge of particle size of the blasted muck pile provides feedback to the drill and blast design and execution process. It also detects oversize to reduce or avoid blockages of the primary crusher.

The use of cameras and image processing on conveyors in the crushing and screening process provides measured data on particle size and material flow. This data can be analysed to optimise throughput, proactively manage crusher closed side settings and provide early indications of blockages and screen damage problems.

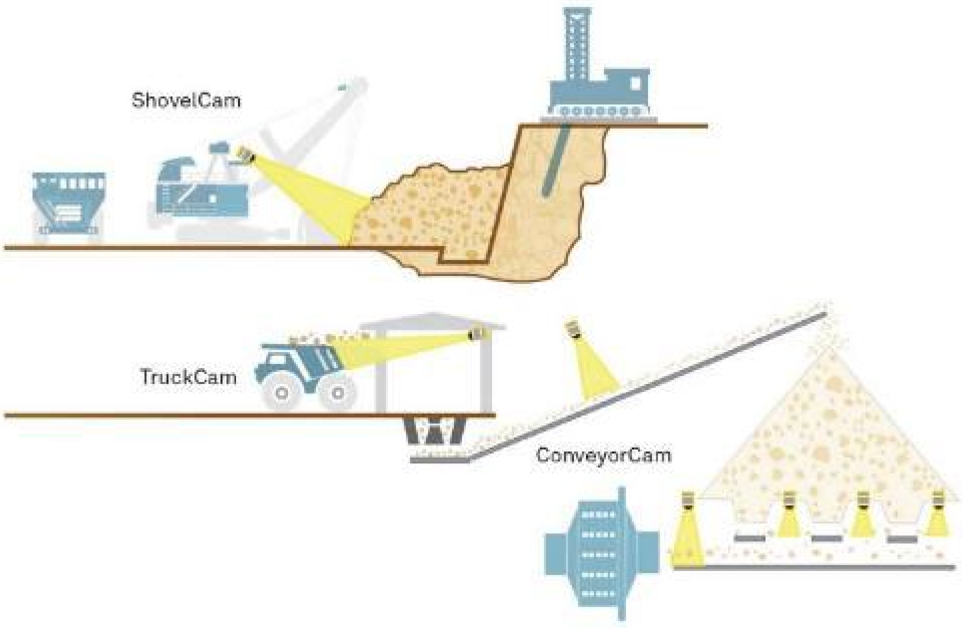


**FIG 4** – Image processing technology is highly versatile, monitoring shovels, excavators, loaders, haul trucks, crushers, conveyor belts, mill feed and screen decks.

## IMPACT OF FRAGMENTATION ON LOAD AND HAUL PERFORMANCE

Many papers confirm blast fragmentation impacts digability and truckload factors, and several field studies have endeavoured to quantify the impact.

Data from fieldwork at Granny Smith gold mine in Western Australia was subjected to rigorous statistical and simulation studies, Brunton *et al* (2003). This study concluded that $P_{80}$ was the best fragmentation distribution parameter to estimate dig time. For $P_{80}$ values greater than 800 mm, the relationship with dig time was expected to increase, resulting in a more significant change in dig time for any increase in $P_{80}$.

## CASE STUDIES

Business cases for measuring blast movement and PSD at the shovel and truck dump locations can be based on:

- Preventing oversize rocks from entering the primary crusher to decrease the number of bridging incidents and hence downtime of the crusher.
- Feedback to Blast Design Engineers as part of a Mine to Mill optimisation project:
  - For mines with a SAG mill in the grinding circuit, blasting finer producing rocks smaller than the aperture of the SAG mill discharge grate reduces the specific energy in comminution and increases SAG mill throughput.
  - For hematite iron ore mines, minimising fines in the blast can increase lump to fines ratio, thus increasing revenue.
- Increase in shovel or loader digability and truck fill factors.
- Decrease in the maintenance of Ground Engaging Tools and truck bodies.
- Minimise misclassification of material, and optimise blending as a direct result of blast movement.

The following case studies outline how some mines measure PSD to manage blast design to optimise fragmentation, measure blast movement to reduce misclassification and improve mine site productivity and profits.

## Morenci Copper Mine, Arizona

PSD data from cameras on shovels was used to increase fragmentation of the blast by tightening the drill pattern. As a result, top size in the ROM ore muck pile reduced from 1300 to 840 mm while 80 per cent passing size ($P_{80}$) reduced from 700 to 400 mm.

In addition to improving productivity in the downstream value chain, a significant side-benefit from reducing top size and $P_{80}$ in the blast came from an increase in digability with shovel productivity increasing from 30 000 to 70 000 t/day, Lowery, Kemeny and Girdner (2000).

After a year of data analysis of measured fragmentation overall rock types and mining areas, the site employed a zone-specific blast fragmentation optimisation program to identify a target PSD to optimise the next stage in the process, either heap leach (for oxide ore) or feed to the mill (for sulfide ore).

Once targets were identified, Morenci measured fragmentation from shovel mounted cameras to manage fragmentation in the blast to further improve productivity. The outcome is summarised in Figure 5.

Blast pattern 9 × 9 m with 2 m man

Blast pattern 8 × 8 m with 250 mm ball

| Drill pattern (metres) | $P_{80}$ (mm) | Top size (mm) | Shovel tons per day |
|---|---|---|---|
| 9.1 × 9.1 | 700 | 1270 | **30 000** |
| 8.2 × 8.2 | 400 | 840 | **70 000** |

**FIG 5** – Different blasting parameters yielded better results, person and object for scale in blue circle.

## African copper mine

A large copper mine in Africa reports daily downtime at the primary crusher due to oversize rock bridging events. This information is used to track the downtime, where the truck came from, the time at which that truck was loaded and the PSD data from the shovel. Fragmentation is monitored and reported as shift and daily PSD data and is a crucial metric in their hour-by-hour and day-to-day operations.

## Porgera Gold Mine, Papua New Guinea

By increasing fineness of ROM fragmentation at Porgera through blast design and properties of explosives used, loader dig time decreased by 36 per cent and truck payload increased by 10 per cent. Both positive production increases led to an overall productivity increase in the loading and hauling phases of mining of 12 per cent.

## Peru copper mines

Two open pit copper mines recognised the need to invest in drill and blast operations to improve rock fragmentation. Key objectives identified included:

- optimise resources, technologies and processes
- evaluate blast design to improve fragmentation from pit to plant
- improve shape of the muck pile
- improve SAG milling energy efficiency
- increase equipment utilisation efficiency
- increase SAG mill throughput.

ShovelCam, TruckCam and ConveyorCam were implemented to measure fragmentation from post-blast product to SAG mill feed. Blast designs were adjusted to optimise post blast fragmentation to increase equipment efficiency and SAG mill throughput leading to:

- 50 per cent increase in material less than 25 mm
- 20 per cent increase in mining equipment productivity
- 12 per cent increase in SAG mill throughput
- 13 per cent reduction in SAG mill specific energy consumption
- USD7.5 million per month increase in revenue, realising USD90 million a year.

## Highland Valley Copper, Canada

Low-tech visual identifiers on the muck pile indicated blast movement but the amount of displacement was difficult to quantify. The mine sought to further optimise the kriged model to include less inherent dilution:

- blast movement monitors (BMMs) were installed in monitoring holes throughout each shot
- installation and detection of the BMMs carried out as per site standard operating procedures
- integrated software calculated new dig lines, and areas of ore loss and dilution that would have occurred without monitoring.

An example blast demonstrates the extent of movement and recovered value. Movement occurs within all blasts, and horizontal movement variation ±50 per cent from the mean is common.

In this one blast, measured horizontal movement ranged from 2 to 11 m and vertical movement was up to 6.4 m. By accurately accounting for blast movement per Figure 6, Highland Valley Copper:

- **Avoided 35 per cent dilution:** 22 700 t of low-grade waste were diverted from the mill.
- **Increased ore yield 24 per cent:** 15 400 t of higher-grade ore were recovered.
- **Increased revenue:** reducing dilution and ore loss in the one blast increased recovered value by USD80 000.

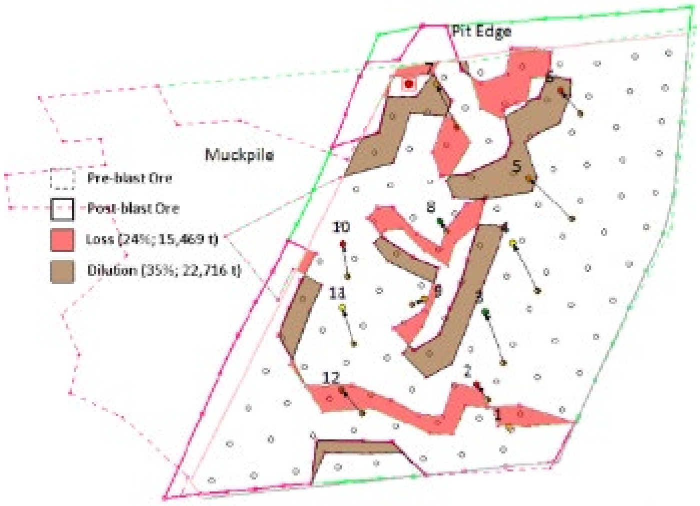


**FIG 6** – Blast movement at Teck's Highland Valley Copper operation.

# SIZE OF THE PRIZE OR THE VALUE OPPORTUNITY

Modifying blast design can avoid production of oversize rocks which block the primary crusher:

- For a 40 Mt/a operation delivering 5000 t/h. to the primary crusher at a value of $30 per ton, one hour of downtime reduces revenue by $150000.

Modifying blast design to decrease the PSD in the blast can increase digability and truckload factors. Decreasing particle size in the blast can reduce load and haul costs by 8 per cent:

- Typically load and haul costs in an open pit mine are ~20 per cent of total costs. If the total mine cost is ~$12 per ton, load and haul would be ~$2.40 per ton. If a 40 Mt/a operation achieved a decrease in load and haul costs of 8 per cent or 20 cents per ton, the reduction in costs would be $8 000 000 per annum.
- For mines with semi-autogenous grinding (SAG), increasing fines in ROM ore can increase SAG mill throughput from 8–30 per cent, Cameron, Drinkwater and Pease (2017). Increase in revenue can vary from significant to astronomical.

Blast design can be managed to increase iron ore lump to fines ratio, Kojovic *et al* (1998):

- If a 40 Mt/a iron ore operation increases lump to fines ratio by 5 per cent from 20 million to 22 million t/a lump, with a $10 per ton premium for lump, revenue could increase by $20 000 000.

Measuring PSD on crusher discharge conveyors allows automatic adjustment of the closed side setting (CSS) which avoids downtime for manual measurement and increases safety by removing people from this task.

Measuring PSD on screen discharge conveyors can detect oversize from screen panel wear or holes in screen panels. If an oversize particle is detected three times in one minute, the control system can shut the process down to repair the offending hole in a screen panel:

- The cost of re-screening is in the order of $5 per t. If two re-screening exercises of 100 000 t each are encountered each year, costs increase by $1 000 000 per annum.

## Cost of automatically measuring PSD by image analysis

For a typical 40 Mt/a mine with six shovels, one primary crusher, and five conveyors in the grinding circuit, the capital for a 12-camera image analysis system would be in the order of USD800 000, delivering an impressive return on investment.

## Benefits of monitoring blast movement

Many mines using blast movement technology report accurately identifying ore movement delivers savings of several hundred thousand dollars per blast.

## CONCLUSIONS

Technologies and methodologies to measure and optimise fragmentation and blast movement are readily available and can be automated to integrate into the mine operation's workflow to deliver significant increases in yield, productivity, and mine site profits.


## ACKNOWLEDGEMENTS

The authors acknowledge support from Hexagon to submit this paper for publication.


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# Scaled depth of burial application to increase blast efficiency in PT. Indo Muro Kencana

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## ABSTRACT

PT. Indo Muro Kencana (PT.IMK) is a gold mine that is located in Central Kalimantan, Indonesia. Drilling and blasting activity is applied in order to support mining production. Drilling and Blasting (D&B) planning is conducted by its engineers. A drilling and blasting contractor conducts drilling and blasting activity. The objective of this paper is to evaluate the implementation of the scaled depth of burial (Chiappetta and Treleven, 1997) in explosive charging sheet application to increase blast efficiency. The function of the scaled depth of burial is to maintain stemming height above explosive charge in order to prevent the 'over energy' condition, to reduce the excessive fly rock possibility, to maintain rock fragmentation (digger's productivity) and to increase blast efficiency.

The focus of this paper is to implement the scaled depth of burial as a reference in D&B planning and practice, especially in designing stemming height. The purpose of this paper is to validate that the scaled depth of burial can be used as an engineering tool in D&B practice in PT.IMK. However, scaled depth of burial cannot applied as an engineering tool to determine stemming height above explosive charge for trim blastholes and secondary blasting area. In general, the implementation of the scaled depth of burial in PT.IMK is applicable for production blastholes (ore and waste blasting).

In conclusion, it is clear that scaled depth of burial can be used as an engineering tool to create explosive charging sheet and also to increase blast efficiency (ore and waste blasting) in PT.IMK.

## INTRODUCTION

PT. Indo Muro Kencana (PT.IMK) is a gold mine that is located in Murung Raya region, Central Kalimantan province, Indonesia. This mining area (previously known as Mount Muro Gold mine) can be reached by using land transport at approximately 7–7.5 hours from city of Palangka Raya (the capital city of Central Kalimantan province). Alternatively, it can be reached by using air transport, approximately one hour flight from Tjilik Riwut airport (airport of Palangka Raya).

The mining operation at PT.IMK is operated in multiple open pits using the surface mining method (combination excavator and haul truck). The mining operation is performed in two shifts and from 2021 to early 2022, the mine operation is conducted in Bantian area and Serujan pit. In addition, rock blasting (drilling and blasting) is applied in order to support mining production. Moreover, the planning for drilling and blasting (D&B planning such as: drill design, blast design and explosive charging sheet) is conducted by its D&B engineers. At this stage, for blasthole drilling with 127 mm hole diameter is conducted a contractor, namely PT. Dahana as drilling service company. Blasting activities (such as: explosives supply, explosives transport, priming explosives, charging explosives, stemming blastholes and then firing/blasting) is also done by PT. Dahana as blasting service for PT.IMK. In addition, the height of mining benches is 15 m and 18 m.

Blasting of ore and waste rock is applied in PT.IMK to fulfill the mining production target and ore fragmentation resulted from blasting is hauled to mineral processing plant.

The scaled depth of burial is applied as an engineering tool to design stemming height in order to prevent over energy, to reduce excessive fly rock, to maintain rock fragmentation and to increase blast efficiency. On the blast location, explosive charging practice for waste area and ore area are identified and divided by the ore line (this guide is provided by grade control geologist). Explosive charging practice is conducted by refer to explosive charging sheet. The objective of this paper is to evaluate the implementation of scaled depth of burial (Chiappetta and Treleven, 1997) to increase blast efficiency. The writer will only evaluate the implementation of scaled depth of burial in designing stemming height.

The focus of this paper is the implementation of scaled depth of burial at D&B practice in PT.IMK, especially during D&B planning and during explosive charging activity on bench. The purpose of this paper is to validate that scaled depth of burial can be used as engineering tool and engineering standard for D&B engineering current practice in PT.IMK. Furthermore, it seems that scaled depth of burial cannot be utilised as reference and engineering tool to determine stemming height above explosive charge in trim blastholes (blasting near final wall) and secondary blasting area (boulder blasting). It is important to note that engineering adjustment need to be made by D&B Engineer for charging explosives in trim blasting holes and secondary blasting (boulder blasting).

In general, the implementation of the scaled depth of burial in PT.IMK is applicable for production blastholes (ore and waste blasting). In fact, it can be seen that the scaled depth of burial application can be used as a reference and as an engineering tool for explosive charging sheet application to increase blast efficiency in PT.IMK.

## MINING ACTIVITIES AT PT.IMK

Currently, mining activities in PT.IMK is conducted by applying surface mining method and using combination of excavator and haul trucks (dump truck). There are several excavators with various capacities used to achieve the production target. Likewise, there also several dump trucks with the capacity of 39 t to 55 t used for the transportation of the ore to the crushing plant and overburden material to the waste dump. The list of mining equipment in PT.IMK is shown in Table 1.

In simple terms, a diagrammatic in Figure 1 is illustrating the typical transportation system in the mine area to the crushing plant (ore) and waste dump (waste). At this point, ADT is the acronym for Articulated Dump Truck (Volvo A40G/F, with capacity of 39 t). Next, HD is acronym for Rigid Dump Truck (HD Caterpillar 773 and HD Terex TR60, with capacity of 55 t). Next, waste dump is acronym to describe area to dump waste material (barren rock). In addition, rom pad stockpile is acronym to define ore stockpile at processing plant area. It is located at crusher area (mineral processing plant).

There is nothing special in the transportation system except a requirement that the ore production must be met to the target of the crushed rock for the next step of mineral processing.

**TABLE 1**

Mining equipment.

| Unit | Type | Average capacities (payload) |
|---|---|---|
| Excavator | Excavator Caterpillar Cat 349 | 3.6 bcm |
| Excavator | Excavator Volvo 750BL | 4.5 bcm |
| Excavator | Excavator Volvo 480D | 3.5 bcm |
| Excavator | Excavator PC 400 | 2.6 bcm |
| HD Hauler | HD (Rigid Dump Truck) Terex TR60 | 55 t |
| HD Hauler | HD (Rigid Dump Truck) Caterpillar 773E | 55.5 t |
| ADT Hauler | ADT Volvo A40G/F | 39 t |

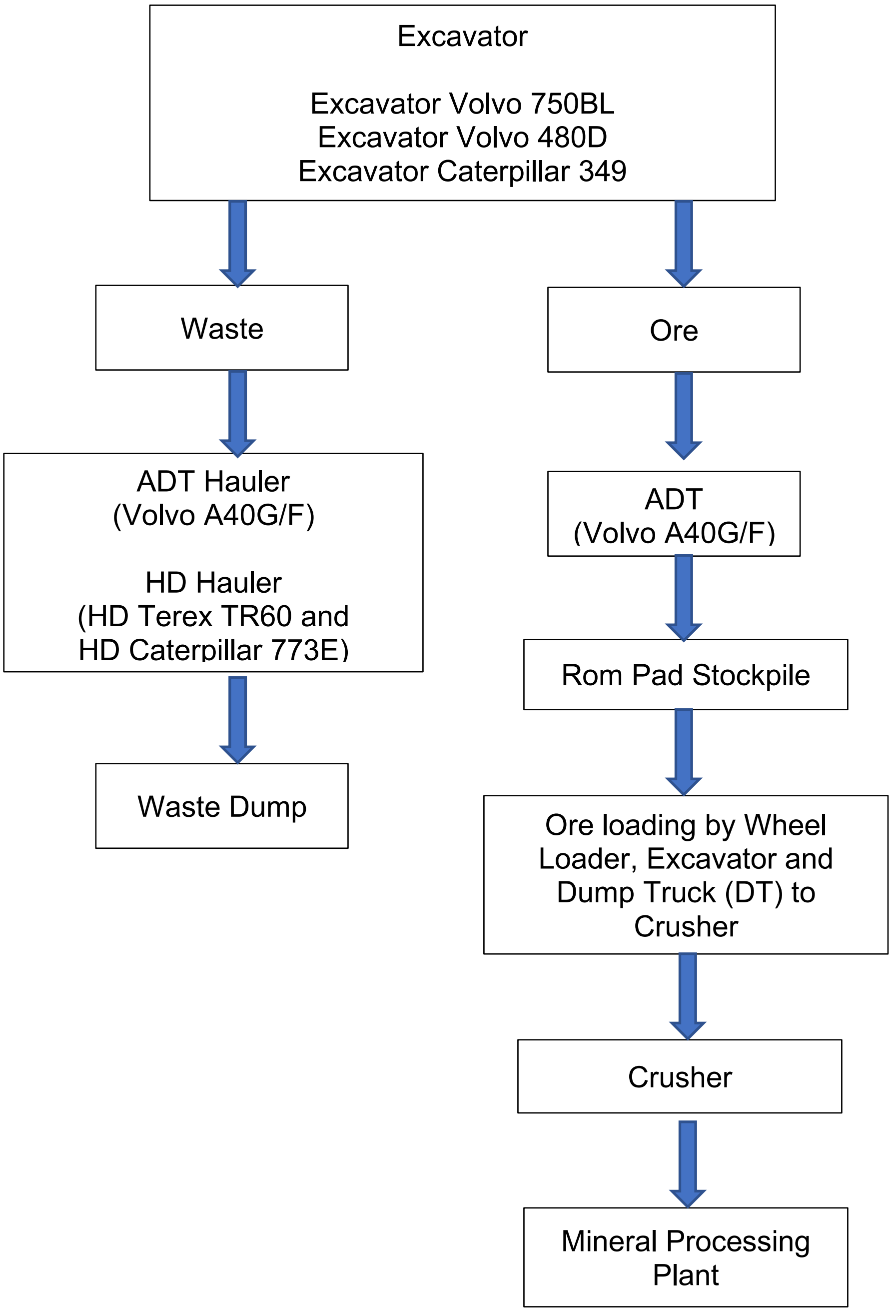



**FIG 1** – Mining processes and system in PT. Indo Muro Kencana.

## BLAST EFFICIENCY AND OPTIMUM BLASTING

Current D&B engineering applied in PT.IMK is directed to break rock mass to become rock fragmentation of both valuable ore and waste rock. Ore and waste blasting is the term where the ore and waste rock are blasted in the same blasting process. Suwandhi (2012) states that drilling and blasting or the rock breakage process is the first step in a series of production processes in mining and quarrying and the cost of this initial step is normally only 8–12 per cent of total cost production. It is recognised that the use of commercial explosives to break the rock is favourable due to a lowest cost option when the rock cannot be free-dug or continuously mined because of its geological characteristics. Then, rock blasting is also considered to be more optimum if compared to mining operation using combination of ripping (dozer ripping) and excavator. At this stage, it is expected that rock fragmentation by blasting is smaller than the size of the excavator's bucket and also smaller than the crusher's opening size.

To provide a good distribution of rock fragmentation consistency the blasting process should also be considering a drilling and blasting cost, mining cost in order to reduce repair and maintenance costs (R&M costs) and fuel cost. Furthermore, managing the drilling and blasting cost will maintain electricity cost in crushing plant and mineral processing plant. To get an optimum blast operation and increase blast efficiency, the concept of the scaled depth of burial is implemented.

The optimum blasting is a condition where the fragmentation of the blasting results can be utilised for further processes with guaranteed safely, controlled blast negative impact to surrounding area and blasting cost – effective.

## DRILL AND BLAST ENGINEERING

There are several active pits at PT.IMK with the routine activities of drilling, blasting, loading and hauling. For the multiple pits operation, the company has a blasting target of around 2000 kilotonne (kt) per month of both ore and waste rock.

The rock mass is classified as hard rock with Unconfined Compressive Strength (UCS) varying from 40–195 MPa and consists of small portion of transition soft rock material of oxide and alluvial.

The rock conditions has the potential for reactive ground, therefore to bulk explosive is added with inhibitors in order to anticipate the presence of reactive ground.

The type of explosive used is an emulsion blend, normally blended 70 per cent emulsion matrix and 30 per cent ANFO, called Dabex FRG produced by PT. Dahana, Indonesia, and initiated by conventional Nonel system. This type of explosive has enough power to break rock mass of ore and waste rock when having a density of 1.15 g/cc.

Details of explosives type and initiation system can be seen in Table 2 and Figure 2.

**TABLE 2**

Types of explosives in use at PT.IMK.

| Explosives | Type | Notes |
|---|---|---|
| Bulk Explosives | Dabex FRG series.<br>Emulsion blend,<br>70% emulsion matrix and 30% ANFO | Explosive diameter 5 inches (127 mm)<br>Density 1.15 g/cc.<br>Explosive loading density 14.57 kg/m.<br>Emulsion explosives have high water resistance in order to anticipation of wet holes. |
| Cast Booster | Dayaprime 200 g | 200 g booster for trim blasting and secondary blasting (boulder blasting) |
| Cast Booster | Dayaprime 400 g booster | 400 g booster (for production holes blasting; ore and waste blasting) |
| Package Booster | Dayagel Magnum 1000 g | 1000 g package booster (for trim blasting, secondary and production blasting) |
| In hole detonator | Dayadet in-hole length | 6.0 m; 9.0 m; 12.0 m |
| | Dayadet in-hole delay | 500 ms; 3000 ms |
| Surface detonator | Dayadet surface delay | 6.0 m |
| | Dayadet surface delay | 17 ms; 25 ms; 42 ms; 67 ms; 109 ms |
| Lead-in Line | Dayadet Lead-in Line | 500 m |

**FIG 2** – Explosives in use at PT.IMK.

D&B Engineers design blast geometry on the basic of blasthole diameter of 127 mm and loading density of explosive of that diameter is 14.57 kg/m. Other parameter of blast geometry with variation of bench height is shown at Table 3. Table 3 also shows the powder factor (PF) that is the ratio of explosive to tonnage of ore or waste rock for each bench height, where the density of rock is 2.5 t/bcm. If burden and spacing are always the same for any different bench height, it can be found that the higher the bench height, than the lower PF will be obtained. The same thing will happen when burden and spacing are increased.

Excess blast energy sometimes occurs due to poor of the amount of stemming material and the overcharging of explosives. The result of 'over energy' is uncontrolled energy that can lead to violent fly rock, excessive air blast noise and dust. In addition, good crater and oversized material can be produced. Oversize will negatively influence excavator productivity.

**TABLE 3**

Blast geometry with variation of bench height (H).

| **Blast geometry** | | **Waste H=5.0 m** | **Ore H=5.0 m** | **Waste H=6.0 m** | **Ore H=6.0 m** |
|---|---|---|---|---|---|
| Blasthole diameter, d | (mm) | 127 | 127 | 127 | 127 |
| Burden, B | (m) | 3.5 | 3.5 | 3.5 | 3.5 |
| Spacing, S | (m) | 4.0 | 4.0 | 4.0 | 4.0 |
| Stemming, T | (m) | 2.7 | 3.0 | 3.2 | 3.5 |
| Bench Height, H | (m) | 5.0 | 5.0 | 6.0 | 6.0 |
| Hole depth, L | (m) | 5.5 | 5.5 | 6.5 | 6.5 |
| Sub drill, J | (m) | 0.5 | 0.5 | 0.5 | 0.5 |
| Powder Column, PC | (m) | 2.8 | 2.5 | 3.3 | 3.0 |
| Explosive density, $\partial_E$ | (g/cc) | 1.15 | 1.15 | 1.15 | 1.15 |
| Explosive Loading Density, $L_d$ | (kg/m) | 14.57 | 14.57 | 14.57 | 14.57 |
| Explosive weight, $W_E$ | kg/hole) | 40.80 | 36.43 | 48.08 | 43.71 |
| Powder Factor, PF | (kg/tonne) | 0.23 | 0.21 | 0.23 | 0.21 |
| Scaled Depth of Burial, SDoB | (kg/m$^{1/3}$) | 1.26 | 1.37 | 1.45 | 1.56 |

| Blast geometry | | Waste H=7.5 m | Ore H=7.5 m | Waste H=9.0 m | Ore H=9.0 m |
|---|---|---|---|---|---|
| Blasthole diameter, d | (mm) | 127 | 127 | 127 | 127 |
| Burden, B | (m) | 4.0 | 4.0 | 5.0 | 5.0 |
| Spacing, S | (m) | 4.5 | 4.5 | 6.0 | 6.0 |
| Stemming, T | (m) | 2.6 | 3.2 | 2.7 | 3.2 |
| Bench Height, H | (m) | 7.5 | 7.5 | 9.0 | 9.0 |
| Hole depth, L | (m) | 8.0 | 8.0 | 9.5 | 9.5 |
| Sub drill, J | (m) | 0.5 | 0.5 | 0.5 | 0.5 |
| Powder Column, PC | (m) | 5.4 | 4.8 | 6.8 | 6.3 |
| Explosive density, $\partial_E$ | (g/cc) | 1.15 | 1.15 | 1.15 | 1.15 |
| Explosive Loading Density, $L_d$ | (kg/m) | 14.57 | 14.57 | 14.57 | 14.57 |
| Explosive weight, $W_E$ | kg/hole) | 78.68 | 69.94 | 99.08 | 91.79 |
| Powder Factor, PF | (kg/tonne) | 0.23 | 0.21 | 0.15 | 0.14 |
| Scaled Depth of Burial, SDoB | (kg/m$^{1/3}$) | 1.22 | 1.45 | 1.26 | 1.45 |

The Drill and Blast Section of PT.IMK is tasked to manage D&B planning, blasting operation, evaluation and reporting of drilling and blasting activities. In this section there are D&B Superintendent, D&B Engineers and D&B supervisors. The D&B Engineers have to carry out and responsible a daily drill and blast proposal that covering planning and designing drill and blast works, assigning explosives charging sheet, predicting distribution of blast fragmentation, and monitoring the blast effects to the surrounding area. Their duty will be supervised by D&B Superintendent.

The company of PT.IMK as a mine owner cooperates with PT. Dahana as a contractor who has a task to carry out drilling and blasting in accordance with the plans and designs made by D&B Engineers of PT. IMK. PT. Dahana performs a job of blasting activity covering priming, explosives charging and pouring stemming material into blasthole, tie up and firing. Supply, delivery and permit of the explosives is also conducted by PT. Dahana, as per Indonesia Police regulation and the Ministry of Mines regulation, and then is stored in the magazine (explosives storage) provided by PT.IMK.

## DRILL and BLAST PROPOSAL

The D&B proposal is created by D&B Engineer as a detailed plan of weekly mine planning (production schedule). Information in D&B proposal is a polygon of the blast area, blast geometry, recommendation (from mine planning engineer, geotechnical engineer, grade control geologist, mine survey), drilled metre required, explosives requirement, blast volume, blast tonnage and powder factor. Next, it is reviewed and then approved by D&B superintendent. A sample of D&B proposal in PT.IMK can be seen in Figure 3.

## Drill design

Initially, a drill design is created by D&B Engineer as a detailed version of the D&B proposal and as per mine plan engineer schedule (weekly plan). The information in the drill design is a polygon of the blasting area, blasting geometry (burden, spacing, drill depth), recommendation/instruction from D&B Engineer. A drill design application at PT.IMK can be seen on Figure 4.

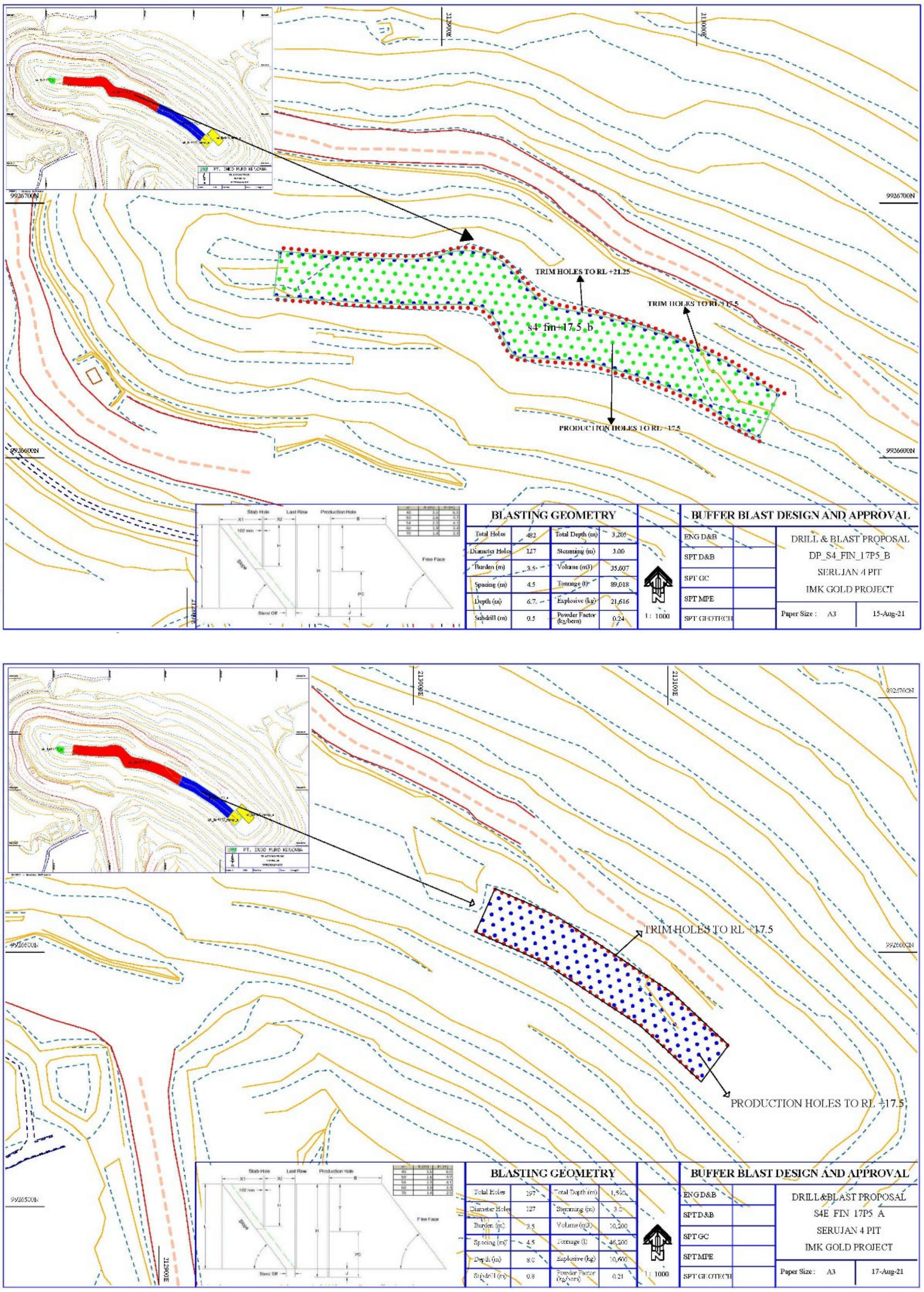



**FIG 3** – Drill and blast proposal.

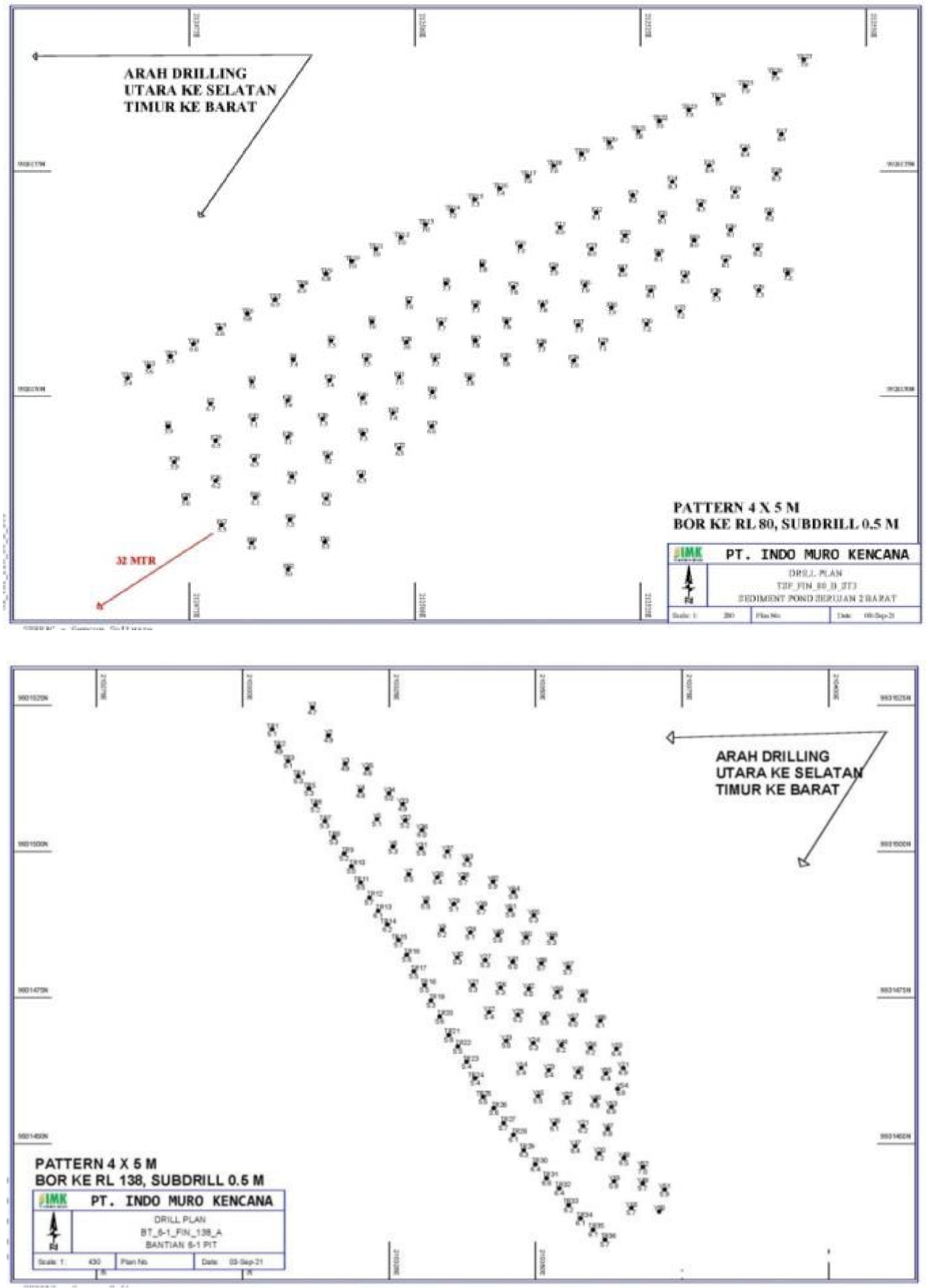



**FIG 4** – Drill design.

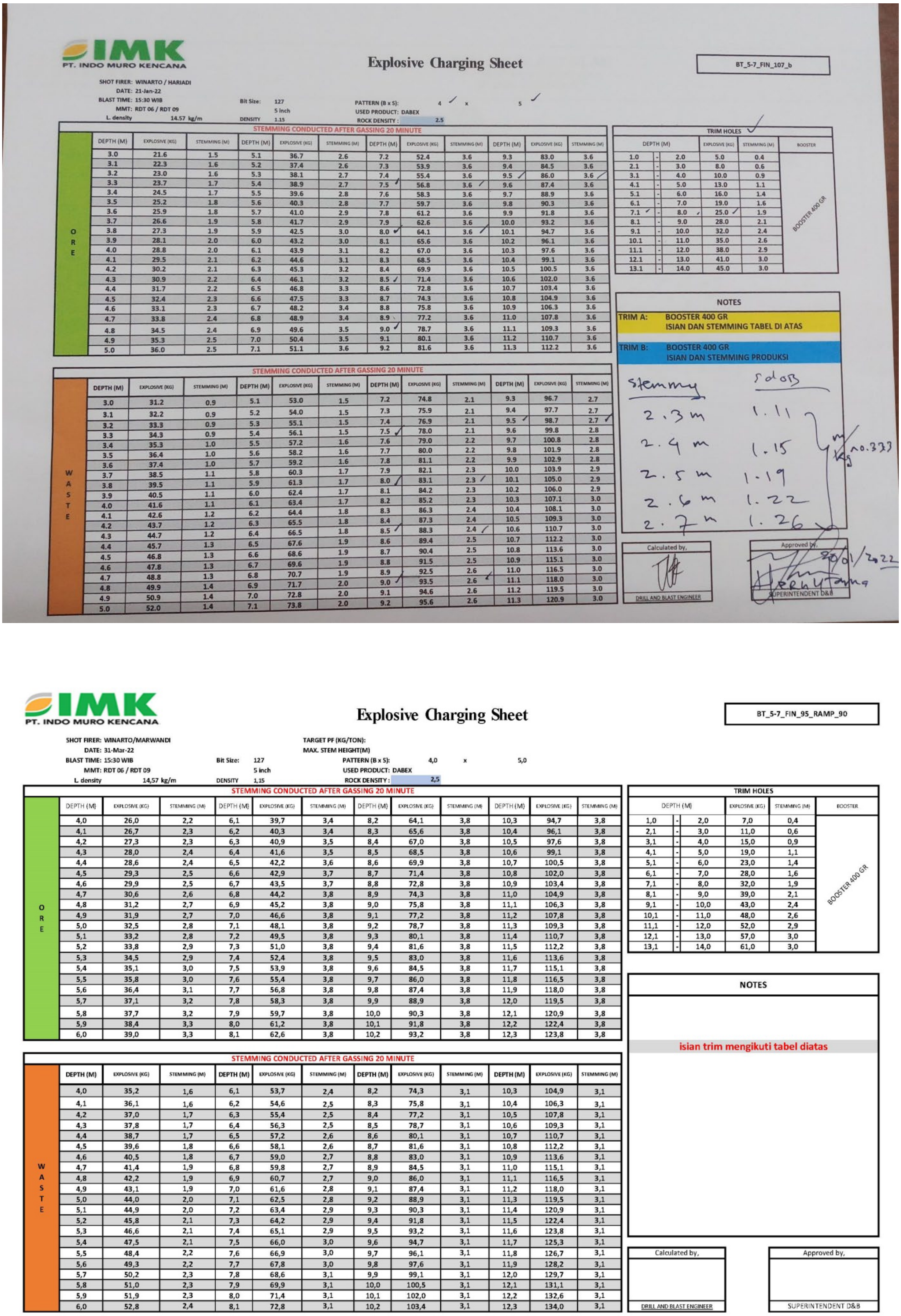
## Explosive charging sheet

The explosive charging sheet is a charging standard (guide) for blaster and blast crew for loading explosives on bench. The scaled depth of burial is an engineering tool to determine stemming height in explosive charging sheet in PT.IMK.

The scaled depth of burial is applied by D&B Engineer in order to reduce over energy, to reduce excessive fly rock, to maintain rock fragmentation (digger's productivity) and finally to increase blast efficiency.

The scaled depth of burial application in explosive charging sheet in PT.IMK is shown in Figure 5.

At this stage, it is clear that scaled depth of burial is consistently applied and implemented by D&B Engineer at PT. IMK as an engineering tool to design stemming height in explosive charging sheet.



IMK
PT. INDO MURO KENCANA

**Explosive Charging Sheet**

BT_5-7_FIN_107_b

SHOT FIRER: WINARTO / HARIADI
DATE: 21-Jan-22
BLAST TIME: 15:30 WIB
MMT: RDT 06 / RDT 09
L density 14.57 kg/m
Bit Size: 127
5 inch
DENSITY 1.15
PATTERN (B x S): 4 x 5
USED PRODUCT: DABEX
ROCK DENSITY : 2.5

STEMMING CONDUCTED AFTER GASSING 20 MINUTE

ORE

| DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 3.0 | 21.6 | 1.5 | 5.1 | 36.7 | 2.6 | 7.2 | 52.4 | 3.6 | 9.3 | 83.0 | 3.6 |
| 3.1 | 22.3 | 1.6 | 5.2 | 37.4 | 2.6 | 7.3 | 53.9 | 3.6 | 9.4 | 84.5 | 3.6 |
| 3.2 | 23.0 | 1.6 | 5.3 | 38.1 | 2.7 | 7.4 | 55.4 | 3.6 | 9.5 | 86.0 | 3.6 |
| 3.3 | 23.7 | 1.7 | 5.4 | 38.9 | 2.7 | 7.5 | 56.8 | 3.6 | 9.6 | 87.4 | 3.6 |
| 3.4 | 24.5 | 1.7 | 5.5 | 39.6 | 2.8 | 7.6 | 58.3 | 3.6 | 9.7 | 88.9 | 3.6 |
| 3.5 | 25.2 | 1.8 | 5.6 | 40.3 | 2.8 | 7.7 | 59.7 | 3.6 | 9.8 | 90.3 | 3.6 |
| 3.6 | 25.9 | 1.8 | 5.7 | 41.0 | 2.9 | 7.8 | 61.2 | 3.6 | 9.9 | 91.8 | 3.6 |
| 3.7 | 26.6 | 1.9 | 5.8 | 41.7 | 2.9 | 7.9 | 62.6 | 3.6 | 10.0 | 93.2 | 3.6 |
| 3.8 | 27.3 | 1.9 | 5.9 | 42.5 | 3.0 | 8.0 | 64.1 | 3.6 | 10.1 | 94.7 | 3.6 |
| 3.9 | 28.1 | 2.0 | 6.0 | 43.2 | 3.0 | 8.1 | 65.6 | 3.6 | 10.2 | 96.1 | 3.6 |
| 4.0 | 28.8 | 2.0 | 6.1 | 43.9 | 3.1 | 8.2 | 67.0 | 3.6 | 10.3 | 97.6 | 3.6 |
| 4.1 | 29.5 | 2.1 | 6.2 | 44.6 | 3.1 | 8.3 | 68.5 | 3.6 | 10.4 | 99.1 | 3.6 |
| 4.2 | 30.2 | 2.1 | 6.3 | 45.3 | 3.2 | 8.4 | 69.9 | 3.6 | 10.5 | 100.5 | 3.6 |
| 4.3 | 30.9 | 2.2 | 6.4 | 46.1 | 3.2 | 8.5 | 71.4 | 3.6 | 10.6 | 102.0 | 3.6 |
| 4.4 | 31.7 | 2.2 | 6.5 | 46.8 | 3.3 | 8.6 | 72.8 | 3.6 | 10.7 | 103.4 | 3.6 |
| 4.5 | 32.4 | 2.3 | 6.6 | 47.5 | 3.3 | 8.7 | 74.3 | 3.6 | 10.8 | 104.9 | 3.6 |
| 4.6 | 33.1 | 2.3 | 6.7 | 48.2 | 3.4 | 8.8 | 75.8 | 3.6 | 10.9 | 106.3 | 3.6 |
| 4.7 | 33.8 | 2.4 | 6.8 | 48.9 | 3.4 | 8.9 | 77.2 | 3.6 | 11.0 | 107.8 | 3.6 |
| 4.8 | 34.5 | 2.4 | 6.9 | 49.6 | 3.5 | 9.0 | 78.7 | 3.6 | 11.1 | 109.3 | 3.6 |
| 4.9 | 35.3 | 2.5 | 7.0 | 50.4 | 3.5 | 9.1 | 80.1 | 3.6 | 11.2 | 110.7 | 3.6 |
| 5.0 | 36.0 | 2.5 | 7.1 | 51.1 | 3.6 | 9.2 | 81.6 | 3.6 | 11.3 | 112.2 | 3.6 |

STEMMING CONDUCTED AFTER GASSING 20 MINUTE

WASTE

| DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 3.0 | 31.2 | 0.9 | 5.1 | 53.0 | 1.5 | 7.2 | 74.8 | 2.1 | 9.3 | 96.7 | 2.7 |
| 3.1 | 32.2 | 0.9 | 5.2 | 54.0 | 1.5 | 7.3 | 75.9 | 2.1 | 9.4 | 97.7 | 2.7 |
| 3.2 | 33.3 | 0.9 | 5.3 | 55.1 | 1.5 | 7.4 | 76.9 | 2.1 | 9.5 | 98.7 | 2.7 |
| 3.3 | 34.3 | 0.9 | 5.4 | 56.1 | 1.5 | 7.5 | 78.0 | 2.1 | 9.6 | 99.8 | 2.8 |
| 3.4 | 35.3 | 1.0 | 5.5 | 57.2 | 1.6 | 7.6 | 79.0 | 2.2 | 9.7 | 100.8 | 2.8 |
| 3.5 | 36.4 | 1.0 | 5.6 | 58.2 | 1.6 | 7.7 | 80.0 | 2.2 | 9.8 | 101.9 | 2.8 |
| 3.6 | 37.4 | 1.0 | 5.7 | 59.2 | 1.6 | 7.8 | 81.1 | 2.2 | 9.9 | 102.9 | 2.8 |
| 3.7 | 38.5 | 1.1 | 5.8 | 60.3 | 1.7 | 7.9 | 82.1 | 2.3 | 10.0 | 103.9 | 2.9 |
| 3.8 | 39.5 | 1.1 | 5.9 | 61.3 | 1.7 | 8.0 | 83.1 | 2.3 | 10.1 | 105.0 | 2.9 |
| 3.9 | 40.5 | 1.1 | 6.0 | 62.4 | 1.7 | 8.1 | 84.2 | 2.3 | 10.2 | 106.0 | 2.9 |
| 4.0 | 41.6 | 1.1 | 6.1 | 63.4 | 1.7 | 8.2 | 85.2 | 2.3 | 10.3 | 107.1 | 3.0 |
| 4.1 | 42.6 | 1.2 | 6.2 | 64.4 | 1.8 | 8.3 | 86.3 | 2.4 | 10.4 | 108.1 | 3.0 |
| 4.2 | 43.7 | 1.2 | 6.3 | 65.5 | 1.8 | 8.4 | 87.3 | 2.4 | 10.5 | 109.3 | 3.0 |
| 4.3 | 44.7 | 1.2 | 6.4 | 66.5 | 1.8 | 8.5 | 88.3 | 2.4 | 10.6 | 110.7 | 3.0 |
| 4.4 | 45.7 | 1.3 | 6.5 | 67.6 | 1.9 | 8.6 | 89.4 | 2.5 | 10.7 | 112.2 | 3.0 |
| 4.5 | 46.8 | 1.3 | 6.6 | 68.6 | 1.9 | 8.7 | 90.4 | 2.5 | 10.8 | 113.6 | 3.0 |
| 4.6 | 47.8 | 1.3 | 6.7 | 69.6 | 1.9 | 8.8 | 91.5 | 2.5 | 10.9 | 115.1 | 3.0 |
| 4.7 | 48.8 | 1.3 | 6.8 | 70.7 | 1.9 | 8.9 | 92.5 | 2.6 | 11.0 | 116.5 | 3.0 |
| 4.8 | 49.9 | 1.4 | 6.9 | 71.7 | 2.0 | 9.0 | 93.5 | 2.6 | 11.1 | 118.0 | 3.0 |
| 4.9 | 50.9 | 1.4 | 7.0 | 72.8 | 2.0 | 9.1 | 94.6 | 2.6 | 11.2 | 119.5 | 3.0 |
| 5.0 | 52.0 | 1.4 | 7.1 | 73.8 | 2.0 | 9.2 | 95.6 | 2.6 | 11.3 | 120.9 | 3.0 |

TRIM HOLES

| DEPTH (M) | | | EXPLOSIVE (KG) | STEMMING (M) | BOOSTER |
|---|---|---|---|---|---|
| 1.0 | - | 2.0 | 5.0 | 0.4 | BOOSTER 400 GR |
| 2.1 | - | 3.0 | 8.0 | 0.6 | |
| 3.1 | - | 4.0 | 10.0 | 0.9 | |
| 4.1 | - | 5.0 | 13.0 | 1.1 | |
| 5.1 | - | 6.0 | 16.0 | 1.4 | |
| 6.1 | - | 7.0 | 19.0 | 1.6 | |
| 7.1 | - | 8.0 | 25.0 | 1.9 | |
| 8.1 | - | 9.0 | 28.0 | 2.1 | |
| 9.1 | - | 10.0 | 32.0 | 2.4 | |
| 10.1 | - | 11.0 | 35.0 | 2.6 | |
| 11.1 | - | 12.0 | 38.0 | 2.9 | |
| 12.1 | - | 13.0 | 41.0 | 3.0 | |
| 13.1 | - | 14.0 | 45.0 | 3.0 | |

NOTES

TRIM A: BOOSTER 400 GR
ISIAN DAN STEMMING TABEL DI ATAS

TRIM B: BOOSTER 400 GR
ISIAN DAN STEMMING PRODUKSI

| stemmy | Sdob |
|---|---|
| 2.3 m | 1.11 |
| 2.4 m | 1.15 |
| 2.5 m | 1.19 |
| 2.6 m | 1.22 |
| 2.7 m | 1.26 |

m/kg^0.333

Calculated by,
DRILL AND BLAST ENGINEER

Approved by,
20/01/2022
Herry Tama
SUPERINTENDENT D&B

IMK
PT. INDO MURO KENCANA

**Explosive Charging Sheet**

BT_5-7_FIN_95_RAMP_90

SHOT FIRER: WINARTO/MARWANDI
DATE: 31-Mar-22
BLAST TIME: 15:30 WIB
MMT: RDT 06 / RDT 09
L density 14,57 kg/m
Bit Size: 127
5 inch
DENSITY 1,15
TARGET PF (KG/TON):
MAX. STEM HEIGHT(M)
PATTERN (B x S): 4,0 x 5,0
USED PRODUCT: DABEX
ROCK DENSITY : 2,5

STEMMING CONDUCTED AFTER GASSING 20 MINUTE

ORE

| DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 4,0 | 26,0 | 2,2 | 6,1 | 39,7 | 3,4 | 8,2 | 64,1 | 3,8 | 10,3 | 94,7 | 3,8 |
| 4,1 | 26,7 | 2,3 | 6,2 | 40,3 | 3,4 | 8,3 | 65,6 | 3,8 | 10,4 | 96,1 | 3,8 |
| 4,2 | 27,3 | 2,3 | 6,3 | 40,9 | 3,5 | 8,4 | 67,0 | 3,8 | 10,5 | 97,6 | 3,8 |
| 4,3 | 28,0 | 2,4 | 6,4 | 41,6 | 3,5 | 8,5 | 68,5 | 3,8 | 10,6 | 99,1 | 3,8 |
| 4,4 | 28,6 | 2,4 | 6,5 | 42,2 | 3,6 | 8,6 | 69,9 | 3,8 | 10,7 | 100,5 | 3,8 |
| 4,5 | 29,3 | 2,5 | 6,6 | 42,9 | 3,7 | 8,7 | 71,4 | 3,8 | 10,8 | 102,0 | 3,8 |
| 4,6 | 29,9 | 2,5 | 6,7 | 43,5 | 3,7 | 8,8 | 72,8 | 3,8 | 10,9 | 103,4 | 3,8 |
| 4,7 | 30,6 | 2,6 | 6,8 | 44,2 | 3,8 | 8,9 | 74,3 | 3,8 | 11,0 | 104,9 | 3,8 |
| 4,8 | 31,2 | 2,7 | 6,9 | 45,2 | 3,8 | 9,0 | 75,8 | 3,8 | 11,1 | 106,3 | 3,8 |
| 4,9 | 31,9 | 2,7 | 7,0 | 46,6 | 3,8 | 9,1 | 77,2 | 3,8 | 11,2 | 107,8 | 3,8 |
| 5,0 | 32,5 | 2,8 | 7,1 | 48,1 | 3,8 | 9,2 | 78,7 | 3,8 | 11,3 | 109,3 | 3,8 |
| 5,1 | 33,2 | 2,8 | 7,2 | 49,5 | 3,8 | 9,3 | 80,1 | 3,8 | 11,4 | 110,7 | 3,8 |
| 5,2 | 33,8 | 2,9 | 7,3 | 51,0 | 3,8 | 9,4 | 81,6 | 3,8 | 11,5 | 112,2 | 3,8 |
| 5,3 | 34,5 | 2,9 | 7,4 | 52,4 | 3,8 | 9,5 | 83,0 | 3,8 | 11,6 | 113,6 | 3,8 |
| 5,4 | 35,1 | 3,0 | 7,5 | 53,9 | 3,8 | 9,6 | 84,5 | 3,8 | 11,7 | 115,1 | 3,8 |
| 5,5 | 35,8 | 3,0 | 7,6 | 55,4 | 3,8 | 9,7 | 86,0 | 3,8 | 11,8 | 116,5 | 3,8 |
| 5,6 | 36,4 | 3,1 | 7,7 | 56,8 | 3,8 | 9,8 | 87,4 | 3,8 | 11,9 | 118,0 | 3,8 |
| 5,7 | 37,1 | 3,2 | 7,8 | 58,3 | 3,8 | 9,9 | 88,9 | 3,8 | 12,0 | 119,5 | 3,8 |
| 5,8 | 37,7 | 3,2 | 7,9 | 59,7 | 3,8 | 10,0 | 90,3 | 3,8 | 12,1 | 120,9 | 3,8 |
| 5,9 | 38,4 | 3,3 | 8,0 | 61,2 | 3,8 | 10,1 | 91,8 | 3,8 | 12,2 | 122,4 | 3,8 |
| 6,0 | 39,0 | 3,3 | 8,1 | 62,6 | 3,8 | 10,2 | 93,2 | 3,8 | 12,3 | 123,8 | 3,8 |

STEMMING CONDUCTED AFTER GASSING 20 MINUTE

WASTE

| DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) | DEPTH (M) | EXPLOSIVE (KG) | STEMMING (M) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 4,0 | 35,2 | 1,6 | 6,1 | 53,7 | 2,4 | 8,2 | 74,3 | 3,1 | 10,3 | 104,9 | 3,1 |
| 4,1 | 36,1 | 1,6 | 6,2 | 54,6 | 2,5 | 8,3 | 75,8 | 3,1 | 10,4 | 106,3 | 3,1 |
| 4,2 | 37,0 | 1,7 | 6,3 | 55,4 | 2,5 | 8,4 | 77,2 | 3,1 | 10,5 | 107,8 | 3,1 |
| 4,3 | 37,8 | 1,7 | 6,4 | 56,3 | 2,5 | 8,5 | 78,7 | 3,1 | 10,6 | 109,3 | 3,1 |
| 4,4 | 38,7 | 1,7 | 6,5 | 57,2 | 2,6 | 8,6 | 80,1 | 3,1 | 10,7 | 110,7 | 3,1 |
| 4,5 | 39,6 | 1,8 | 6,6 | 58,1 | 2,6 | 8,7 | 81,6 | 3,1 | 10,8 | 112,2 | 3,1 |
| 4,6 | 40,5 | 1,8 | 6,7 | 59,0 | 2,7 | 8,8 | 83,0 | 3,1 | 10,9 | 113,6 | 3,1 |
| 4,7 | 41,4 | 1,9 | 6,8 | 59,8 | 2,7 | 8,9 | 84,5 | 3,1 | 11,0 | 115,1 | 3,1 |
| 4,8 | 42,2 | 1,9 | 6,9 | 60,7 | 2,7 | 9,0 | 86,0 | 3,1 | 11,1 | 116,5 | 3,1 |
| 4,9 | 43,1 | 1,9 | 7,0 | 61,6 | 2,8 | 9,1 | 87,4 | 3,1 | 11,2 | 118,0 | 3,1 |
| 5,0 | 44,0 | 2,0 | 7,1 | 62,5 | 2,8 | 9,2 | 88,9 | 3,1 | 11,3 | 119,5 | 3,1 |
| 5,1 | 44,9 | 2,0 | 7,2 | 63,4 | 2,9 | 9,3 | 90,3 | 3,1 | 11,4 | 120,9 | 3,1 |
| 5,2 | 45,8 | 2,1 | 7,3 | 64,2 | 2,9 | 9,4 | 91,8 | 3,1 | 11,5 | 122,4 | 3,1 |
| 5,3 | 46,6 | 2,1 | 7,4 | 65,1 | 2,9 | 9,5 | 93,2 | 3,1 | 11,6 | 123,8 | 3,1 |
| 5,4 | 47,5 | 2,1 | 7,5 | 66,0 | 3,0 | 9,6 | 94,7 | 3,1 | 11,7 | 125,3 | 3,1 |
| 5,5 | 48,4 | 2,2 | 7,6 | 66,9 | 3,0 | 9,7 | 96,1 | 3,1 | 11,8 | 126,7 | 3,1 |
| 5,6 | 49,3 | 2,2 | 7,7 | 67,8 | 3,0 | 9,8 | 97,6 | 3,1 | 11,9 | 128,2 | 3,1 |
| 5,7 | 50,2 | 2,3 | 7,8 | 68,6 | 3,1 | 9,9 | 99,1 | 3,1 | 12,0 | 129,7 | 3,1 |
| 5,8 | 51,0 | 2,3 | 7,9 | 69,9 | 3,1 | 10,0 | 100,5 | 3,1 | 12,1 | 131,1 | 3,1 |
| 5,9 | 51,9 | 2,3 | 8,0 | 71,4 | 3,1 | 10,1 | 102,0 | 3,1 | 12,2 | 132,6 | 3,1 |
| 6,0 | 52,8 | 2,4 | 8,1 | 72,8 | 3,1 | 10,2 | 103,4 | 3,1 | 12,3 | 134,0 | 3,1 |

TRIM HOLES

| DEPTH (M) | | | EXPLOSIVE (KG) | STEMMING (M) | BOOSTER |
|---|---|---|---|---|---|
| 1,0 | - | 2,0 | 7,0 | 0,4 | BOOSTER 400 GR |
| 2,1 | - | 3,0 | 11,0 | 0,6 | |
| 3,1 | - | 4,0 | 15,0 | 0,9 | |
| 4,1 | - | 5,0 | 19,0 | 1,1 | |
| 5,1 | - | 6,0 | 23,0 | 1,4 | |
| 6,1 | - | 7,0 | 28,0 | 1,6 | |
| 7,1 | - | 8,0 | 32,0 | 1,9 | |
| 8,1 | - | 9,0 | 39,0 | 2,1 | |
| 9,1 | - | 10,0 | 43,0 | 2,4 | |
| 10,1 | - | 11,0 | 48,0 | 2,6 | |
| 11,1 | - | 12,0 | 52,0 | 2,9 | |
| 12,1 | - | 13,0 | 57,0 | 3,0 | |
| 13,1 | - | 14,0 | 61,0 | 3,0 | |

NOTES

isian trim mengikuti tabel diatas

Calculated by,
DRILL AND BLAST ENGINEER

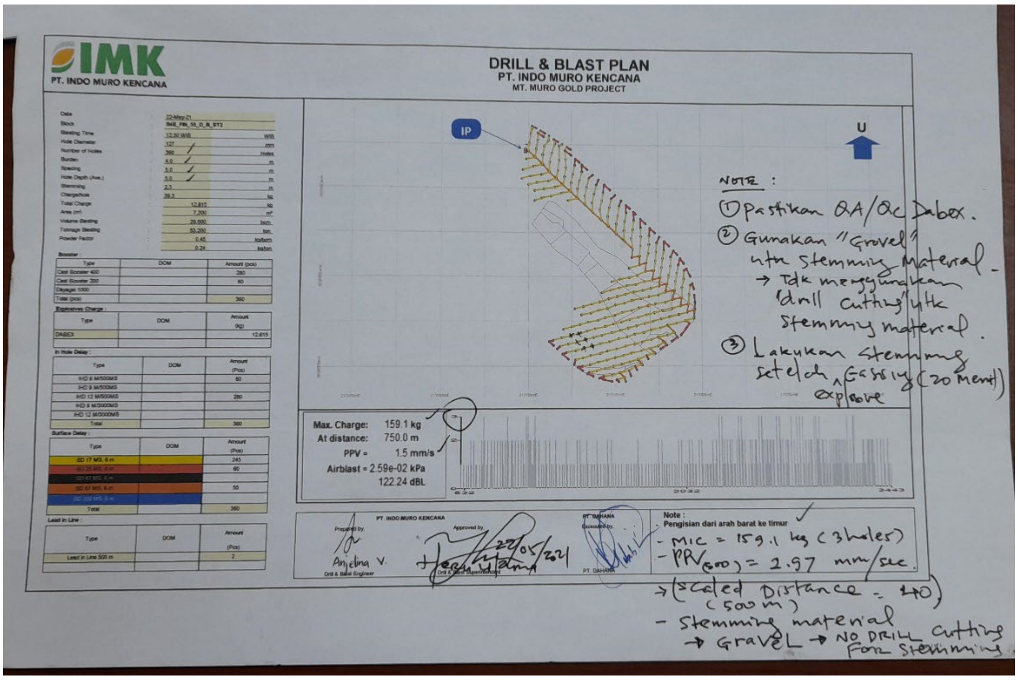
Approved by,
SUPERINTENDENT D&B

**FIG 5** – Explosive charging sheet.

However, the scaled depth of burial application cannot be used for an engineering tool to design stemming height for trim blasting (blasting near final wall), shallow blasthole (less than 3.0 m depth) and secondary blasting. At this point, for trim blasting application and secondary blasting (boulder blasting), it is clear that technical adjustments based on actual rock conditions must be made by D&B engineer in order to control stemming height and to anticipate the over energy condition and to prevent excessive fly rock.

## Blasting design

Daily production blasting in PT.IMK is conducted at 12.30 pm (rest time for day shift). A blast design (drill and blast plan) is made by D&B Engineer for daily production blasting (Figure 6). This blasting design is created by the D&B engineer using blast design software (JKSimBlast software). Additionally, it is used as a guide for blaster and blasting coordinator. The purpose of this blasting design is to achieve optimum blasting condition. The optimum blasting is a condition where the fragmentation of the blasting results can be utilised for further processes with guaranteed safely, controlled blast negative impact to surrounding area and blasting cost-effective.



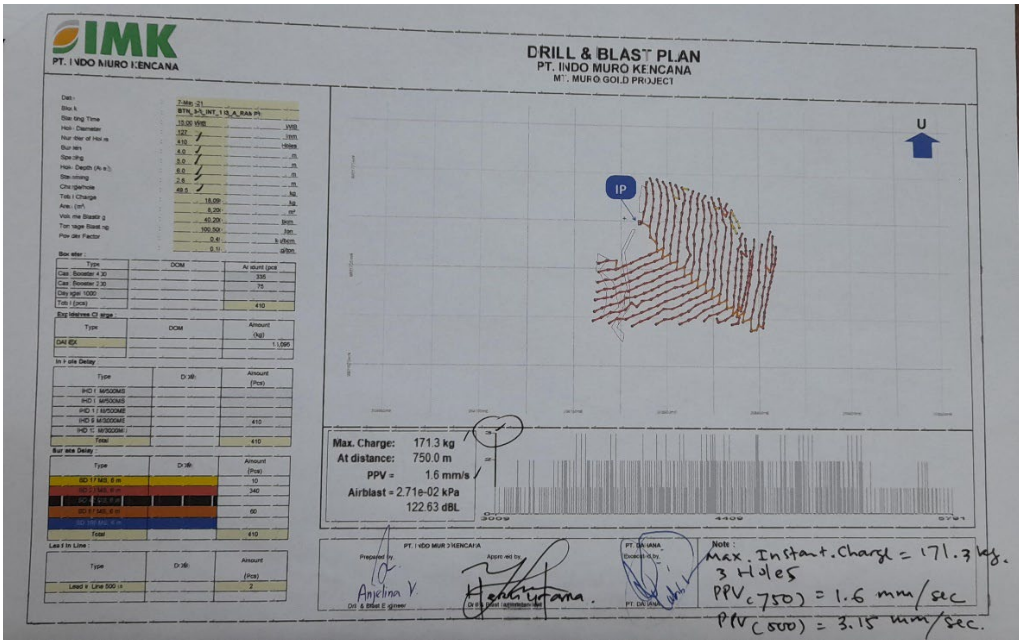




**FIG 6** – Blast design.

## Blast Design Prediction

In PT.IMK, the blast practices are predicted and monitored by D&B Engineer, such as prediction of blasting fragmentation distribution (by using Kuz-Ram method, Cunningham, 1987) and ground vibration monitoring. Prediction of the blast fragmentation distribution, rock blasting geometry and explosive parameter can be adjusted by D&B engineer based on current rock mass condition.

Blast geometry can be adjusted based on the actual rock mass condition and engineering consideration. Rock Factor value that is used in the prediction of fragmentation distribution is obtained from 0.12 times blastability index.

Engineering tool (rock mass classification) that is used by D&B engineer to make adjustments to blast design is the blastability index (Lilly, 1986).

The scaled depth of burial also can be used for fly rock prediction (estimated maximum rock projection and estimated blast clearance radius).

Ground vibration prediction and air blast prediction (scaled distance, maximum instantaneous charge and holes blasted in same delay) are also predicted by D&B engineer. Furthermore, estimated blast volume, blast tonnage, powder factor and blasting economics (estimated drilling and blasting costs) are also predicted by D&B engineer.

Prediction of D&B cost consists of the cost of blasthole drilling (meter drilled), cost of bulk explosives usage, and cost of accessories usage (surface delay detonator, in delay hole detonator, booster, Lead in Line).

# DRILL and BLAST OPERATIONAL

## Drilling

Ore and waste blasting by using conventional blasting method (with Nonel initiation system) and sleep blasting method (with Nonel initiation system).

Blasthole drilling is conducted drilling service company (PT. Dahana) as per design and instruction from the D&B engineer.

Drilling practices in PT.IMK are based on a blasthole diameter 127 mm with 5.5 m to 9 m average depth by using various drill units (top hammer: Furukawa HCR 1500, Sandvik Pantera 1100 and Atlas Copco T45). Figure 7 shows various drill rigs.


**FIG 7** – Drill unit.

## Blasting

Current blasting practice at PT.IMK is ore and waste blasting by using conventional blasting method (Nonel initiation system) and sleep blasting method (Nonel initiation system).

Daily drill and blast proposal, drill design, explosive charging sheet, blast design and blast monitoring by IMK D&B Engineer and supervised by D&B Superintendent.

Daily blasting activity (priming, charging explosives, stemming, tie up and firing) is performed by PT. Dahana (blasting contractor). The bulk explosives in used is emulsion blend type, with 30 per cent ANFO and 70 per cent emulsion matrix.

The bulk explosive density is 1.15 g/cc, explosive loading density 14.57 kg/m (hole diameter 127 mm) and its commercial name is Dabex FRG series (For Reactive Ground). This emulsion blend is a product from PT. Dahana (Persero).

Figures 8 to 11 show priming activity, charging activity, final blast checks and blasted product.


**FIG 8** – Priming activity.


**FIG 9** – Charging explosives and stemming process.


**FIG 10** – Final check by D&B Engineer and blaster.

**FIG 11** – Rock fragmentation by blasting.

# SCALED DEPTH OF BURIAL APPLICATION

## Stemming and scaled depth of burial

In surface mine blasting, stemming is the key to an optimum blasting. It is clear that stemming acts as an energy confinement. In addition, stemming is an important concern in blast design in PT.IMK, especially for ore and waste blasting.

Drill cuttings are not recommended to be used as stemming material for ore and waste blasting in PT.IMK.

Crushed rock (gravel) is recommended to be used for stemming material at PT.IMK in order to increase blasting performance. It is clear that stemming is played important role to increase energy confinement.

The scaled depth of burial is an engineering tool for D&B Engineer to create an explosive charging sheet with proper stemming height in order to raise optimum blasting condition. In addition, the main role of scaled depth of burial application in explosive charging sheet is to design and to maintain stemming height to prevent over energy and to reduce excessive fly rock than can cause property damage to mine equipment.

It is expected that rock fragmentation by blasting with scaled depth of burial application is smaller than the size of the excavator's bucket and also smaller than the crusher's opening size in process plant area.

At this point, the writer will only evaluate the implementation of scaled depth of burial as an engineering tool for designing stemming height in explosive charging sheet. Finally, it seems that scaled depth of burial is applied in daily practice by D&B Engineer, especially in explosive charging sheet. Figure 12 shows the Scaled Depth of Burial ranges.

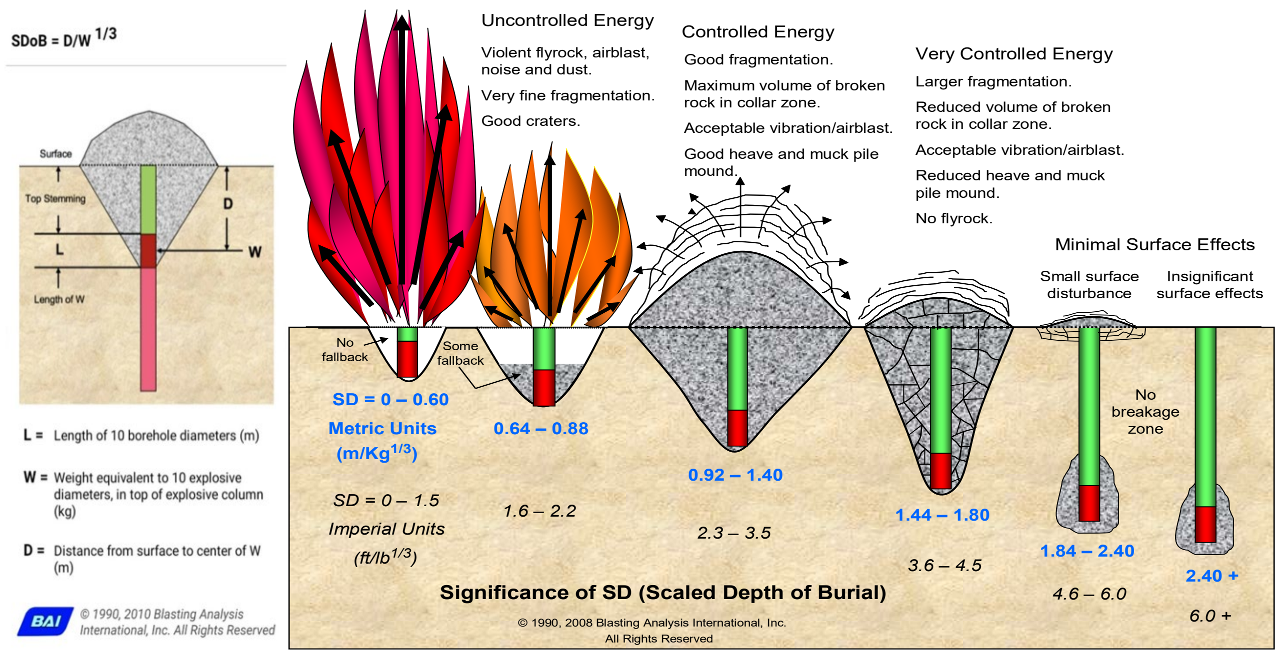



**FIG 12** – Scaled depth of burial (Chiappetta and Treleven, 1997).

## Evaluation of scaled depth of burial application to increased blast efficiency

Currently, blast efficiency in surface mine blasting can defined as the optimum blasting. At this point, to get an optimum blast operation and increase blast efficiency in PT.IMK, the concept of scaled depth of burial is implemented. It is expected that rock fragmentation by blasting that is blasted with scaled depth of burial application will increase blast efficiency.

The concept of scaled depth of burial can be used as reference for explosive charging sheet at ore and waste blasting in D&B engineering practice in PT.IMK. Additionally, it is claimed that the concept of scaled depth of burial is applicable for stemming height design in ore and waste blasting. At this point, the implementation of scaled depth of burial as an engineering tool to design stemming height has been consistently applied by PT.IMK's D&B Engineer.

Explosive charging sheet for daily blasting practice is created by D&B Engineer. In addition, stemming height (m) and scaled depth of burial are used in explosive charging sheet for waste blasting is stemming height$_{waste}$ = 2.5 m - 3.0 m, SDoB$_{waste}$ = 0.92 - 1.4 kg/m$^{1/3}$ (controlled energy).

The value for scaled depth of burial is 1.2 - 1.3 kg/m$^{1/3}$ for waste blast design at PT.IMK.

Stemming height (m) and scaled depth of burial are used in explosive charging sheet for ore blasting is stemming height $_{ore}$ = 3.2 m - 4.0 m, SDoB$_{ore}$ = 1.44 - 1.68 kg/m$^{1/3}$ (very controlled energy). The value for scaled depth of burial is 1.44 - 1.5 kg/m$^{1/3}$ for ore blasting at PT.IMK. Details are listed in Table 4.

**TABLE 4** – Scaled depth of burial application in PT.IMK.

| Blast design parameter | Waste blasting | Ore blasting |
|---|---|---|
| Hole diameter (mm) | 127 | 127 |
| Stemming rule of thumb | 20 – 24 × hole diameter | 25 – 30 × hole diameter |
| Stemming (m) | 2.54 | 3.18 |
| Explosive Density (g/cc) | 1.15 | 1.15 |
| Explosive Loading density (kg/m) | 14.57 | 14.57 |
| L (Length of 10 × Borehole diameter) in m | 1.27 | 1.27 |
| D (Distance from surface to centre of W) (m) | 3.18 | 3.82 |
| W (Weight equivalent to 10 × Borehole diameter) (kg) | 18.5 | 18.5 |
| Scaled Depth of Burial ($m/kg^{1/3}$) | 1.2 | 1.44 |
| Scaled Depth of Burial ($m/kg^{1/3}$) | Controlled energy, good fragmentation | Very controlled energy, larger fragmentation |
| |  |  |

The scaled depth of burial also can be used as an engineering tool for estimated max rock projection and estimated blast clearance radius. The formula for estimated maximum rock projection is:

$$\text{Range Max} = 11 \times \text{SDoB}^{-2.167} \times D^{0.667}$$

The formula for estimated blast clearance radius is:

$$\text{BCR or Blast Clearance Radius} = \text{FoS} \times 11 \times \text{SDoB}^{-2.167} \times D^{0.667}$$

Table 5 shows the Maximum Rock Projection and the Blast Clearance Radius calculation.

**TABLE 5**

Scaled depth of burial application in PT.IMK.

| Blast design parameter | Estimated max rock projection (Range Max) | Estimated blast clearance radius (BCR = Blast Clearance Radius) |
|---|---|---|
| Hole diameter (mm) | 127 | 127 |
| Formula | Range max = 11 × SDoB $^{-2.167}$ × D $^{0.667}$ | Blast clearance radius = FoS × 11 × SDoB $^{-2.167}$ × D $^{0.667}$ |
| Scaled Depth of Burial ($m/kg^{1/3}$) | 1.2<br>Controlled Energy | 1.2<br>Controlled Energy |

| | | |
|---|---|---|
| Range max | 11 × SDoB $^{-2.167}$ × D $^{0.667}$<br>11 × (1.2) $^{-2.167}$ × (127) $^{0.667}$= 187.5 m<br>blast radius for mining equipment (m) = 300 m | FoS × 11 × SDoB $^{-2.167}$ × D $^{0.667}$<br>Blast clearance radius = (2) × 11 × (1.2) $^{-2.167}$ × (127) $^{0.667}$= 375 m<br>blast radius for human (m) = 500 m |

In general, the scaled depth of burial is applied by D&B Engineer in daily practice to maintain stemming height in order to prevent over energy and to maintain rock fragmentation by blasting. It is expected that rock fragmentation by blasting is smaller than size of excavator's bucket and also smaller that crusher's opening size.

Table 6 shows the excavator productivity at PT.IMK. It can be seen that rock fragmentation that is blasted by using scaled depth of burial application is applicable to mine. However, it is important to note that the scaled depth of burial cannot apply as engineering tool to design stemming height for trim blasting holes (final wall), shallow blastholes and secondary blasting (boulder blasting).

**TABLE 6**

Excavator productivity in PT.IMK.

| | Excavator | Target productivity (t/hr) | Average monthly productivity (t/hr) | Productivity variance (t/hr) | Variance |
|---|---|---|---|---|---|
| Jan-22 | Exc Caterpillar 349 | 360 | 387 | 27 | 8% |
| Jan-22 | Exc Volvo 480 | 360 | 359 | -1 | 0% |
| Jan-22 | Exc Volvo 750 | 600 | 607 | 7 | 1% |
| Feb-22 | Exc Caterpillar 349 | 360 | 415 | 55 | 15% |
| Feb-22 | Exc Volvo 480 | 360 | 410 | 50 | 14% |
| Feb-22 | Exc Volvo 750 | 600 | 563 | -37 | -6% |
| Mar-22 | Exc Caterpillar 349 | 360 | 421 | 61 | 17% |
| Mar-22 | Exc Volvo 480 | 360 | 386 | 26 | 7% |
| Mar-22 | Exc Volvo 750 | 600 | 676 | 76 | 13% |
| Apr-22 | Exc Caterpillar 349 | 360 | 432 | 72 | 20% |
| Apr-22 | Exc Volvo 480 | 360 | 420 | 60 | 17% |
| Apr-22 | Exc Volvo 750 | 600 | 694 | 94 | 16% |

## CONCLUSIONS

To get an optimum blast operation and increase blast efficiency in PT.IMK, the concept of the scaled depth of burial is implemented.

The scaled depth of burial can be used as an engineering tool to determine stemming height for explosive charging sheet at ore and waste blasting in D&B engineering practice at PT.IMK.

Based on current practice, it seems that the concept of scaled depth burial is applicable for stemming height design in ore and waste blasting in PT.IMK. Additionally, it is expected that rock fragmentation by blasting that produced from blasting with scaled depth of burial application is fulfilled the requirement for optimum blasting condition. The optimum blasting is a condition where the fragmentation of the blasting results can be utilised for further processes with guaranteed safely, controlled blast negative impact to surrounding area and blasting cost – effective. Also, the formula for estimated maximum projection range and estimated safe distance radius (by using scaled depth of burial) can be used as an engineering tool to reduce over energy, excessive fly rock. It seems that safety issue due to blasting, such as excessive fly rock, will be maintained.

However, it is important to note that the concept of scaled depth of burial cannot be utilised as an engineering tool to determine stemming height in trim blasting holes (blasting near final wall), shallow blasthole (less than 3 m depth) and secondary blasting area (boulder blasting).

At this point, it is recommended that technical adjustment by D&B Engineer is required to validate stemming height and explosive charging sheet for trim blasting and secondary blasting (boulder blasting).

Generally, scaled depth of burial application is applicable for designed stemming height for production blastholes (ore and waste blasting) in PT.IMK. In conclusion, it is clear that scaled depth of burial can be applied in order to prevent excessive fly rock, to reduce over energy, to maintain rock fragmentation by blasting (digger's productivity) and also to increase blast efficiency in PT. Indo Muro Kencana.

## ACKNOWLEDGEMENTS

The authors greatly acknowledge the managements of PT.IMK who gives permission to utilise the data used in this paper.

The authors also would like to express appreciation to PT. Dahana (Persero) for job site Indo Muro Kencana as a partner to perform drilling and blasting practices.

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# Dispatch systems and operational efficiency

# Quantifying iron ore 'handleability' to reduce processing plant delays

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## ABSTRACT

When an iron orebody nears its end of life, the extracted material can become increasingly problematic to handle. Higher moisture content occurs as larger portions of originally-below-water-table material is mined, which in some deposits is accompanied by further changes in physical and/or chemical properties present along the edges of the orebody. Plant delays related to problematic material have affected all of BHP's mining hubs to various degrees.

Following the recent successful work in Yandi, where the development of an index to quantify the handleability of channel iron ore deposits resulted in an increase in extracted volumes and an extension to the expected life-of-mine, the value of implementing similar indices for bedrock iron ore deposits has been recognised.

A materials handling index has been developed by combining observed empirical relationships between material properties and processing plant performance. Indices for each deposit were further refined and optimised by implementing a back-testing strategy which facilitated an understanding of what weighting or scaling each of the contributing components should carry to yield the highest success rate of predicting actual delays at a given plant.

Currently materials handling indices are deployed to all mine sites and are made available in block models as well as calculated for individual blasthole and production samples. Values in block models are obtained by predicting the required properties using machine learning models, whereas hyperspectral data is used to 'measure' some of those properties on a sample level.

This paper discusses the work that has been undertaken to obtain materials handling indices, the testing that has been done to optimise them, and what processes have been developed to ensure the best metrics are being used in the different planning and scheduling horizons to optimise extraction.

## INTRODUCTION

A 2017 BHP internal investigation – triggered by a 3.5 Mt ore-for-rail shortfall from its Yandi mine during that year – identified that there was insufficient early knowledge of how material will handle in processing plants and how this impacts its throughput (Gilroy *et al*, 2018). This led to the development and implementation of an index at Yandi that provided better insight as to how material handles in the value chain (eg Gilroy *et al*, 2018; Ker, 2021). Yandi is BHP's only detrital (Channel Iron Deposit or CID) mining operation and was experiencing handling problems as progressively lower parts of the Yandi channel were being mined.

The implementation of a materials handling index at Yandi allowed not only for improved scheduling of problematic material by blending with higher quality ores, but also unlocked areas of the orebody that had previously been considered uneconomic using other simpler 'handleability' selection criteria.

The problems observed at Yandi are typical for operations that near their end of life, as progressively more of the material that is located below the pre-mining water table is extracted. Plant delays related to problematic material have affected all of BHP's mining hubs to various degrees.

In response to the success of the materials handling index in reducing the amount of delays in the processing plants at Yandi and allowing BHP to extend the life of the mine by five years (Ker, 2021), similar indices are being developed for all its bedrock operations.

This paper discusses the work that has been undertaken to obtain materials handling indices, the testing that has been done to optimise them, and what processes have been developed to ensure the best metrics are being used in the different planning and scheduling horizons to optimise extraction.

## MATERIALS HANDLING INDEX

Materials handling problems in the supply chain are understood to result from the interplay of three main contributors which, when suboptimal, causes the problem. Those contributors are the rock properties of the material, the way this material is presented at the plant and the specifications of the plant. A simplified representation of this interaction is the 'problematic ore triangle' shown in Figure 1. An individual contributor (unless extreme) does not need to cause an issue, however, ores are likely to become problematic if multiple factors are combined. An example of this could be a sample containing large amounts of ochreous goethite (*rock*) that when dry (*feed presentation*) handles well, but when affected by too much moisture, causes a problem.

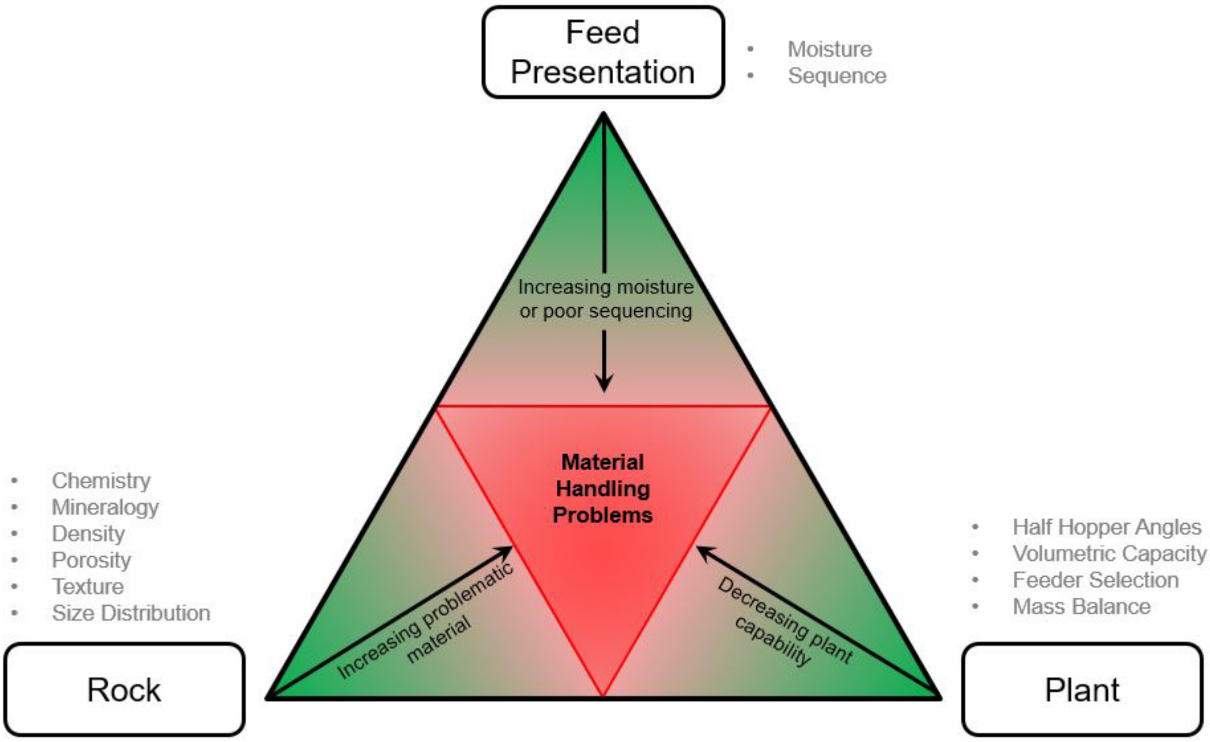


**FIG 1** – The problematic ore triangle, a simple model to illustrate how the interplay between three main contributors (rock properties, feed presentation and plant characteristics) result in materials handling difficulties.

The Materials Handling Index (MHI) is a single index that combines information from the three contributors outlined in Figure 1 and provides an indication of whether that material will handle well downstream. The index is plant specific, and if multiple locations in the supply chain are prone to handling issues, those should all be considered.

### Materials Handling Index in Yandi

The original Yandi MHI accounts for the interplay of several common minerals in channel iron deposits with the effects of moisture. Those components were empirically determined and the problematic range of each of the components was carefully evaluated for the Yandi processing plants and subsequently encoded in the MHI equation by means of scaling factors (Haest, 2018).

To obtain the quantities of each of the contributing components for problematic ore two options are available:

1. For each sample that is processed in the Yandi laboratory, infrared spectroscopic data is collected which covers the visible near infrared and short wavelength infrared parts of the spectrum. Algorithms have been developed in-house to convert absorption features from those spectra into quantities of the problematic ore mineral components and moisture.

2. Several machine learning models have been trained and optimised to predict the quantity of the components from assay data (or grade estimates) and other proxies.

The first option, which yields the best results, is the preferred methodology to calculate an MHI in blasthole samples (and grade control models derived from those) for short-term scheduling and in production samples for reconciliation. However, the second option is used to generate estimates of an MHI for a longer time horizon (eg block models or exploration models), albeit with a lower confidence, where spectral data is not available.

The development and implementation of the Yandi materials handling index has been described in detail by Haest (2018) and has been presented previously by Gilroy *et al* (2018).

## Materials Handling Index for bedrock deposits

For the MHI in bedrock deposits a similar approach was followed as described above for Yandi, however, Yandi's MHI could not simply be replicated across all sites because handleability problems vary for different ore types and this added additional complexities to the bedrock MHI.

In the case of bedrock iron deposits, there are two end members: hypogene martite-microplaty hematite and supergene martite-goethite. These deposits have contrasting mineralogies and textures and, in some instances, the younger supergene ore-forming process has overprinted a pre-existing hypogene deposit thereby introducing further complexity.

Constructing a MHI for bedrock deposits involves the following steps:

- Identify the components that contribute to problematic ore.
- Identify how the combination of those components leads to materials handling problems.
- Ensure components are calculated in drill samples and block models:
  - Implement algorithms to extract component information from hyperspectral data.
  - Train and manage machine learning models to predict components where required. The initial versions of these models are trained using mineralogy from exploration drilling in the deposit.
- Capture the true range over which components are problematic for a given plant (optimisation of index).
- Ensure a robust versioning system exists to not only manage the machine learning model workflow for different deposits, but also the scaling factors per component and the equation that needs to be applied to obtain the MHI.
- Have a monitoring system to track the performance of calculated MHI values.

### *Identifying problematic ore components*

The authors engaged different site teams to understand what material was causing delays in processing plants. This information was often anecdotal and descriptive but could be related to the underlying mineralogy and geo-metallurgical properties of the problematic ore.

From the information obtained from site teams we further established how the different components interact to cause the handling difficulties. This included trying to understand which components, when present together, amplified the problem and which components would cancel each other out. Those relationships were captured in a generic MHI equation for each of our sites.

To be consistent across different BHP mine sites and align to the Yandi MHI, the bedrock MHI is a continuous value between zero and one. Lower values indicate that material is problematic whereas higher values are indicative of material that handles well.

### *Understanding the true problematic range of components*

Similar to the MHI at Yandi, components are simply represented as an index in the MHI equation. The translation of a component quantity into its respective index value has been adopted from Haest

(2018) and is an elegant way to capture the true problematic ranges of each component for a given plant (Figure 2).

To work out the true problematic ranges of the different components in the MHI equation, we tested various realistic scaling ranges for each (positions of the dashed lines in Figure 2) and checked how successful the resulting ‘test’ MHI would have been if it had been in operation for an elapsed period of time (eg last 12 or 24 months). The measure of success for an MHI would be its ability to predict true delay events that had occurred during the test period. To achieve this, we estimated the quantities of components in block models and linked that information to the trucks that sourced their material from the pit to take it to a particular crusher. Different ‘test’ MHI values were calculated for each of those trucks, by applying various scaling ranges to the component quantities present in each truck.

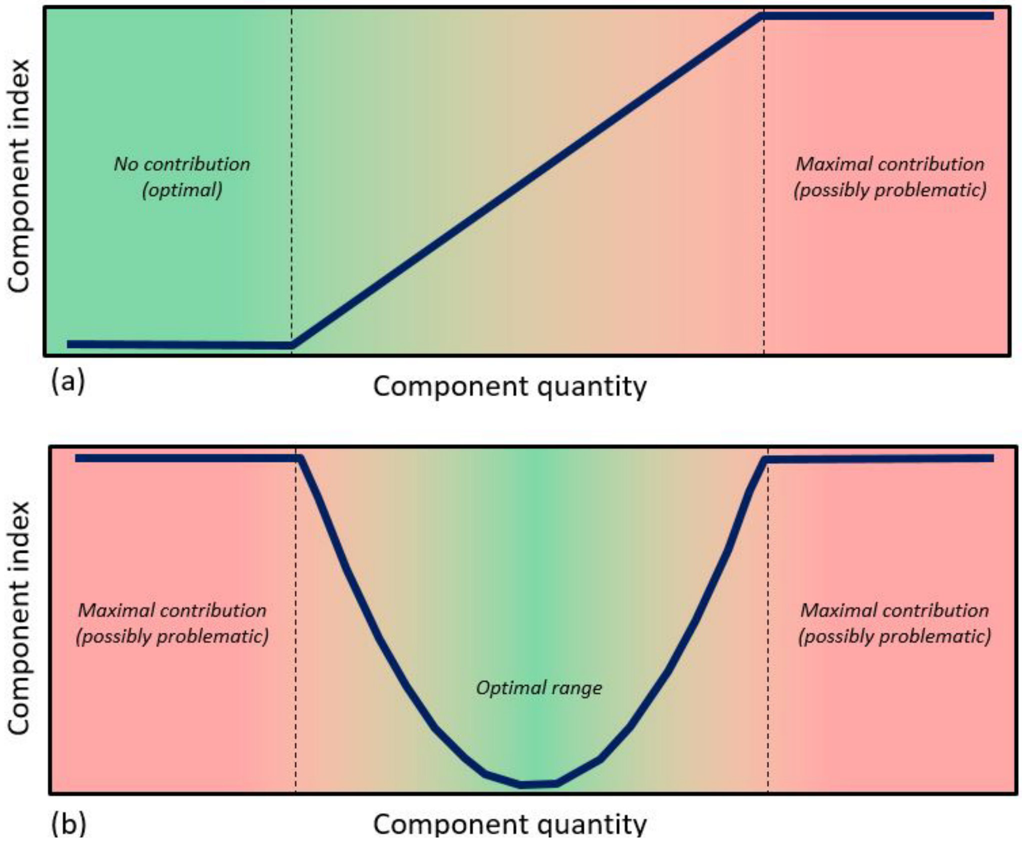


**FIG 2** – Diagrams to show the principle of converting quantities of a given component to an index to capture its true contribution to handleability issues. (a) Example where a component doesn’t contribute until a certain threshold is reached, after which its contribution increases linearly until an upper limit is reached and the contribution is deemed to be maximal. (b) Example of converting a component to an index where both low and high values of the component are considered problematic, and the optimal value lies somewhere in between.

Using a high-resolution ore tracking system developed within BHP’s Technical Service and Data Integration team, to track material through pre-crusher stockpiles, allowed us to further apply the above methodology to trucks that contained rehandled material sourced from stockpiles and not solely those that went directly from pit to crusher (ie direct-feed) trucks. This increased the amount of test points significantly for a given site.

For the back-testing of a MHI candidate version we marked a continuous sequence of trucks with values below a certain threshold (eg 2 or 3 trucks) that dumped in the crusher as an ‘MHI event’ and then compared those events to actual delays recorded for the plant in BHP’s Delay Accounting System (DAS) (Figure 3). A classification metric that accounts for true positives, false positives and false negatives (F-score) was calculated for each of the test MHI values and the best performing MHI per plant was retained and deployed into production systems.

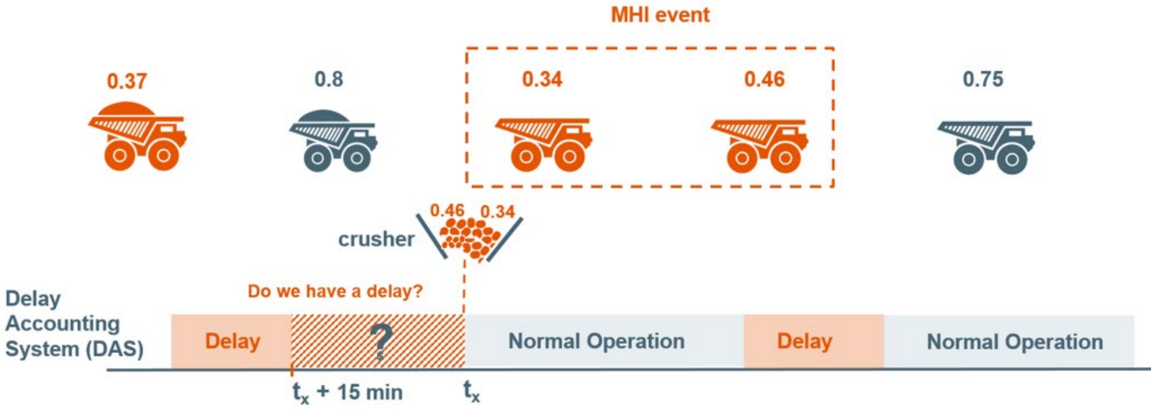


**FIG 3** – Evaluation of MHI performance. The diagram shows the concept of comparing expected MHI 'events' from trucks with actual recorded plant delays. An expected MHI event in this case is observed when a sequence of two trucks with low MHI (eg less than 0.5) unload material in a crusher. If a true delay was recorded in the next 15 minutes, the event would be classified as a *true positive.* If no delay event is observed, the event was a *false positive.* If a delay event occurred but no MHI event was identified, the occurrence was counted as a *false negative.*

## USAGE OF MHI IN BHP

Information about the handleability of ores needs to be available at different time horizons within the business. Currently, MHI values derived from predicted components are populated in the short-term geological models for all pits. Those models can be used by the long-term planning teams.

BHP has implemented the automatic collection of infrared spectra on all the samples that are processed at its Newman laboratory, including blasthole samples and production samples. The component estimates from this data are used in the grade blocking process to provide a higher confidence MHI in the grade control model for use by mine geology and short-term schedulers.

Spectral data collected on production samples (where available) provide a feedback loop between expected values from blastholes and values observed in the sample stations. The results from those samples can further inform handleability issues downstream (eg rail, port or customer).

## MONITORING MHI PERFORMANCE

Monitoring the performance of the Materials Handling Index encompasses two key steps.

1. Ensuring that the underlying prediction models for the components are performing as expected, and not affected by feature creep. This typically occurs when the training data for the prediction model did not contain similar data to what it is trying to predict (ie what currently is being mined).
2. Monitoring the scaling of the contributing components in the MHI equation and adjusting their values if they have become unrepresentative.

Monitoring the performance of the current prediction models is achieved by tracking their performance against more accurate observations from infrared hyperspectral data collected subsequently on blasthole samples. Additional data collected from new blastholes gets regularly added to the training data of the prediction models to keep them up to date.

Adjusting of scaling factors (refer to Figure 2) might be required if parameters in the processing plants (eg screen sizes) have changed, which could impact the performance of the existing MHI and would hence warrant updating the version. The same standardised back-testing framework as outlined in a previous section is used for this purpose. It constantly monitors the ongoing performance of an MHI and compares its performance to alternative MHI versions that are being tracked as possible release candidates. If one of the release candidates consistently outperforms the current MHI, the necessary site engagements to roll out a new version will be started.

## CONCLUSIONS

Materials handling difficulties for both detrital and bedrock deposits can be encoded in a numeric index that captures the interplay of key factors that contribute to problematic ore (corners of the problematic ore triangle).

Different materials handling indices for BHP's bedrock deposits have been constructed and are now routinely deployed in block models and blasthole samples. A back-testing framework was developed to facilitate the optimisation of different MHI versions for various pits and plants. The same framework is now used in a platform for the continuous monitoring of existing MHI performance.

From the back-testing that we have performed prior to implementation, we estimate that the amount of delay at the primary crushers can be reduced by roughly 30–50 per cent, provided the materials handling information is fully embedded in the scheduling processes. Adapting the indices for other downstream processing plants is work that remains outstanding.


## ACKNOWLEDGEMENTS

The authors wish to thank Irish Palma, Hugh McCall and Shane Milligan for their efforts to roll-out the Materials Handling Index to different sites and engagements with downstream customers.

The authors have greatly benefited from discussions with Paul Gilroy and Andrew Hacket who were fundamental in the development and deployment of the original Yandi MHI in 2017. We also acknowledge the tremendous amount of work by Maarten Haest to not only construct the Yandi MHI, but also document in detail the entire process and all the underlying data analysis work.

The ore tracking application in Geoscience is developed and maintained by the Metallurgy team. We are thankful to Mick Lane, Hadi Ariakia and Tim Carpenter for making this available. In addition, Hadi and Tim have been fundamental in setting up the continued automatic deployment of MHI values in block models.

The authors are grateful to all mine geologists and scheduling teams across different sites that have provided assistance with the development of, or provided feedback about, the new MHI's.


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# The role of fleet management in reducing carbon emissions from haul trucks

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## ABSTRACT

The Australian mining industry is a significant contributor to the nation's greenhouse gas (GHG) emissions, and like any industry, it must make efforts to reduce these. Medium- and long-term strategies will include the replacement of diesel engines with alternative energy sources, but what can be done in the short-term to reduce diesel consumption and hence $CO_2$ emissions?

Haul trucks are identified as the major contributors of $CO_2$ emissions in the mining fleet and this paper focuses on firstly identifying what drives haul truck emissions and secondly, discussing what is the role of the Fleet Management System (FMS) in reducing these. A key enabler of this is having a robust fuel consumption model within the FMS, and I describe how this may be done.

Besides novel approaches to incorporating fuel consumption ($CO_2$) management into truck assignment algorithms, this paper also describes how the FMS can be used to implement and manage procedures and policies to reduce haul truck emissions.

## INTRODUCTION

The success of a company's Environment, Social and Governance (ESG) platform is becoming increasingly important. Not only is the sustainability of miners increasingly the focus for capital markets, but also customers and end-users are exerting pressure. The cost of capital can be up to 25 per cent higher for miners with the lowest ESG scores (Legge *et al*, 2021). One of the key takeaways from the AusIMM Social Licence to Operate Forum (AusIMM, 2019) was that social licence is a major source of concern for CEOs globally.

Globally, mining contributes 4–7 per cent of global greenhouse gas (GHG) emissions, and Scope 1 and Scope 2 emissions (emissions from diesel usage and power generation, respectively) amount to 1 per cent of global emissions (Delevingne *et al*, 2020). According to data provided by Australia's Clean Energy Regulator (2019), the country's mining industry contributed 14 per cent of total Scope 1 GHG emissions in 2015–2016, with six mining companies listed in the top 20 corporate contributors.

Whilst many of the largest miners including BHP, Vale and Rio Tinto have set step-change reduction targets ranging from 15–30 per cent by 2030, to achieve a 1.5°C climate change target by 2050, the mining industry will need to reduce direct $CO_2$ emissions to zero (Legge *et al*, 2021). There will be much work to be done to achieve these targets over the coming decades.

Electronic FMSs (Fleet Management System) have been used in open pit mining for over 40 years, and whilst their technology has evolved during that time, their essential function has not. Any FMS aims to record production and time utilisation data digitally and in near real-time. By combining this data with some extra meta-data including the equipment locations, mining blocks, dump boundaries, haul routes and so on, the FMS provides a near-real-time overview of the mine. This overview allows tactical decisions to be made to potentially increase equipment utilisation and productivity. For example, the mine controller or production supervisor can see (or be alerted to) excessive truck queueing and re-allocate the fleet accordingly. The modern FMS is therefore a tool for assisting in-shift operational decision-making, and a data-collection system.

Many current FMSs include mathematical algorithms which allocate trucks to destinations (shovels or dumps) to maximise haulage output. The FMS evaluates potential assignments by considering the haul routes, the current and near-future state of the fleet (delays, dig rates, etc), and any operational goals or constraints, before assigning the truck to the best destination. To the author's knowledge, current FMSs with this capability are focused exclusively on maximising truck productivity.

In the last decade or so, the increasing ubiquity of high-speed wireless communication networks in the operational areas of the mine has allowed the FMS to expand into the asset health space by relaying and recording sensor data from the machines, again in near real-time.

The scope of this paper is to focus on direct GHG emissions from the current diesel-powered mobile mining equipment, and the role that Fleet Management Systems might play in measuring and reducing them.

# MOBILE FLEET $CO_2$ EMISSIONS

## Calculation of $CO_2$ emissions

The $CO_2$ emission from combusted fuel can be determined by on-site metering. However, on-site metering units that continuously monitor equipment emission can be expensive and require permanent maintenance (Mining Environmental Management, 2008). The other alternative is to determine $CO_2$ emission by using mathematical equations.

The $CO_2$ emission from diesel fuels in t/hr can be written as:

$$CO_2 = FC \times CF$$

Where FC is diesel fuel consumption (L/h), and CF is the conversion factor, 0.00268 (Kecojevic and Komljenovic, 2010). Note the linear relationship between $CO_2$ and fuel consumption.

## Sources of emissions

Every open pit mine today uses diesel engines in its day-to-day operation. Legge *et al* (2021) estimated that in an Australian iron ore mine, the diesel-powered mining fleet accounts for over half of the mine's total emissions. Within the mobile fleet, haul trucks accounted for over half of the diesel emissions (55 per cent), followed by loading at 25 per cent. Drilling, blasting, and other equipment contributed to the remaining 20 per cent of mobile fleet emissions.

This is good news for Fleet Management Systems since load and haul are historically their areas of expertise and focus.

## Factors affecting haul truck $CO_2$ emissions

The primary factors which affect haul truck fuel consumption are shown in Figure 1. These factors are grouped into four categories, namely:

- truck characteristics
- haul route characteristics
- truck allocation
- operator behaviour.

Each of these categories are discussed in the following sections.

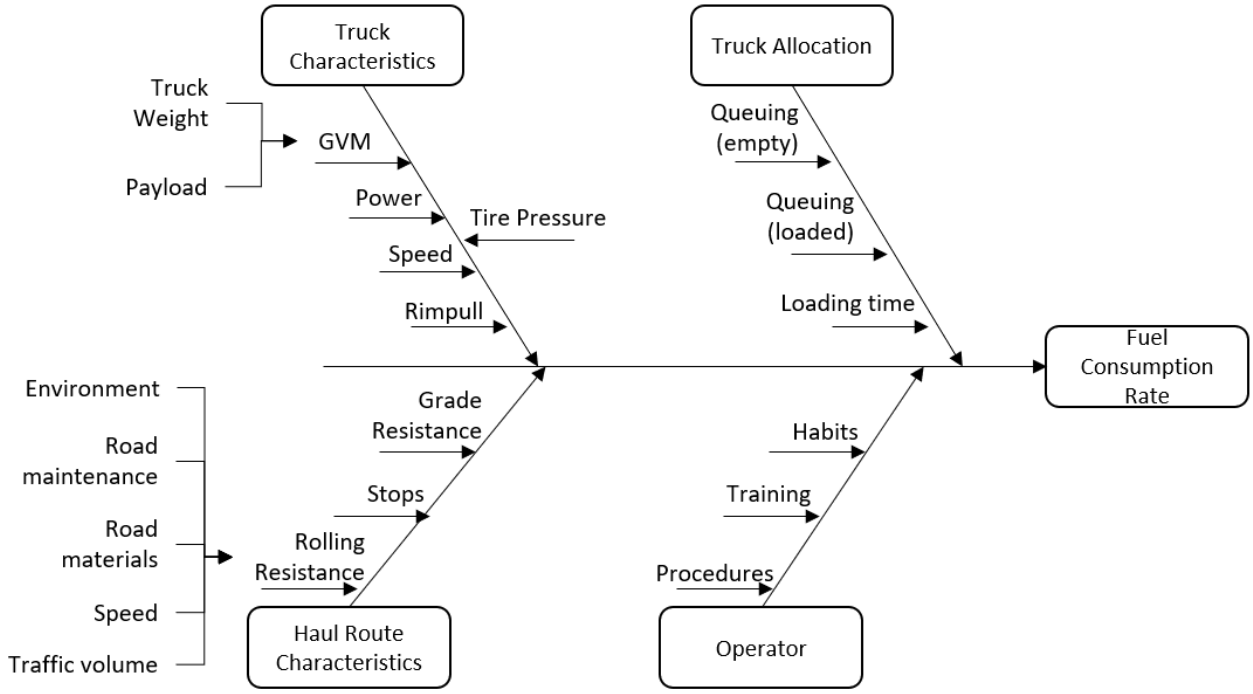


**FIG 1** – Factors affecting truck fuel consumption rate.

## *Truck allocation*

Matching the correct number of haul trucks (haulage capacity) to each shovel (loading capacity) is critical if maximum system productivity is desired. If under-trucked, the shovel will be under-utilised and spend excessive time idling. If there are too many trucks (shovel is over-trucked), the trucks will spend excessive time idling in a queue with multiple acceleration events from stopped as the truck moves forwards in the queue.

As previously discussed, many FMSs can allocate trucks to shovels dynamically to maximise truck productivity and hence total system output. In terms of fuel efficiency, however, this may not be the optimal solution. Awuah-Offei, Osei and Askari-Nasab (2012) concluded that fuel efficiency in truck-shovel systems should be viewed as multi-criteria decisions, since focusing on fuel efficiency alone will lead to recommendations that are bound to be unacceptable to mine operators. In their simulated model, they showed that optimal truck matching that maximises production and shovel utilisation might be in direct conflict with running fuel-efficient operations under certain conditions. This suggests that the existing production-focused optimisation algorithms would need modification to specifically maximise fuel efficiency and that this may produce less than optimal productivity.

Matching the size of the truck to the size of the shovel is also important since loading time is effectively truck idle time. A large truck being loaded by a small shovel will result in long loading times. A poor truck/shovel match will also drive greater variability in payload which will negatively impact fuel usage.

The haulage capacity also needs to be matched to any restrictions on dump capacity. If haulage capacity exceeds the dump capacity (either processing or physical space for trucks), the trucks will spend time queuing at the dump, leading to excessive idling and unnecessary acceleration from stopped whilst loaded. As with queueing at the loading point, this is an area where many FMSs can reduce truck idle time by assigning trucks to an alternative dump.

## *Truck characteristics*

The fuel consumption for haul trucks is determined based on the following parameters (Figure 2):

- The Gross Vehicle Weight (GVW), which is the sum of the weight of an empty truck and the payload.
- The Haul Truck Velocity (V).

- The Total Resistance (TR), which is equal to the sum of Rolling Resistance (RR) and The Grade Resistance (GR) when the truck is moving against the grade of the haul road.
- The Rimpull Force (RF), which is the force available between the tyre and the ground to propel the truck.

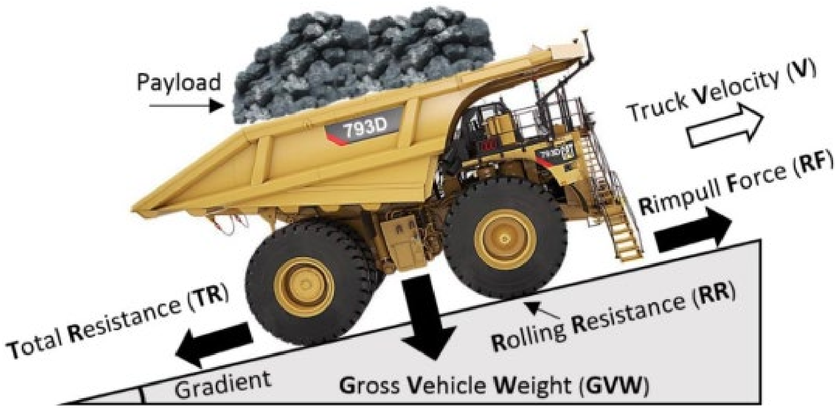


**FIG 2** – Haul road and truck parameters affecting fuel consumption (from Soofastaei *et al*, 2016a).

Figure 3 shows the relationship between the haulage parameters and truck fuel consumption.

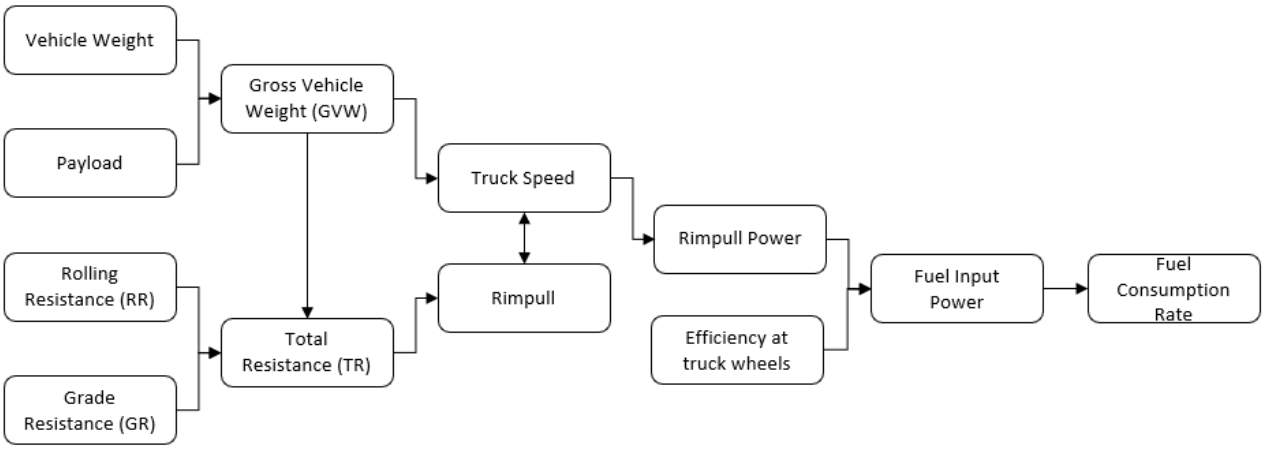


**FIG 3** – Truck characteristics affecting fuel consumption (from Soofastaei *et al*, 2016a).

### Truck size

A study by Leslie (2000) indicates that there is a decrease in fuel consumption when we move to larger and more productive trucks that have more efficient engines. This work was expanded by Schafer-Frentz, Joseph and Curley (2019) to include smaller and larger truck sizes (Table 1 and Figure 4).

**TABLE 1**
Haul truck fuel consumption and $CO_2$ emissions (adapted from Schafer-Frentz, Joseph and Curley (2019).

| Payload (t) | Gross vehicle weight (t) | Effective engine power (kW) | Mean fuel consumption (L/h) | $CO_2$ emissions (t/h) | Change in $CO_2$ (%) |
|---|---|---|---|---|---|
| 70.3 | 108.4 | 590 | 212.3 | 0.57 | - |
| 90.7 | 161.0 | 700 | 261.9 | 0.70 | 22.8 |
| 108.9 | 189.9 | 870 | 301.5 | 0.81 | 15.7 |
| 136.1 | 231.3 | 960 | 352.8 | 0.95 | 17.3 |
| 154.2 | 249.4 | 960 | 389.5 | 1.04 | 9.5 |
| 176.9 | 317.5 | 1320 | 411.8 | 1.10 | 5.8 |
| 217.7 | 376.5 | 1615 | 429.1 | 1.15 | 4.5 |
| 290.3 | 521.6 | 2200 | 410.9 | 1.10 | -4.3 |
| 326.6 | 556.0 | 2400 | 355.0 | 0.95 | -13.6 |
| 362.9 | 623.7 | 2510 | 217.3 | 0.58 | -38.9 |

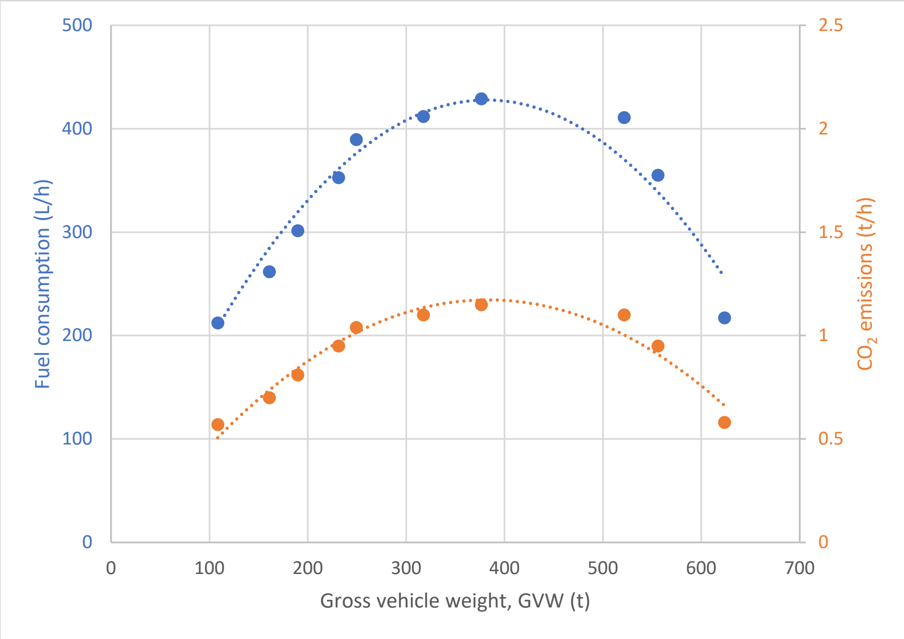


**FIG 4** – Fuel consumption and $CO_2$ emissions by truck gross vehicle weight (GVW) (from Schafer-Frentz, Joseph and Curley, 2019).

The fuel consumption and $CO_2$ emissions increased with GVW up to 218 t ('ultra-class') and then declined thereafter. Schafer-Frentz, Joseph and Curley (2019) concluded that this was due to the significant efforts made by truck original equipment manufacturers (OEMs) with the advent and development of the ultra-class truck after 1997 when engine development primarily targeted this class of truck. Around the same time, the US EPA implemented tiered emission regulations, and technological advances resulted in more efficient fuel control and consumption. Consequently, the trend of reduced emissions with increasing truck sizes since the mid-1990s is understandable (Schafer-Frentz, Joseph and Curley, 2019).

The improvement in emissions is particularly evident when looked at based on truck payload. As payload increases, the rate of emissions per tonne of payload decreases rapidly (Figure 5). Figure 6 shows the percentage increase between truck sizes. For trucks with a payload of 176.9 t or less the rate of change of emissions rate per tonne of a payload is relatively constant (around 5 per cent). As truck payload increases to 217 t and above, the rate of improvement of emission rate per tonne increases dramatically. A 217 t payload truck is approximately 15 per cent lower than a 176 t truck ($CO_2$ per hour per tonne of payload), and a 290 t truck is almost 30 per cent better than the 217 t truck.

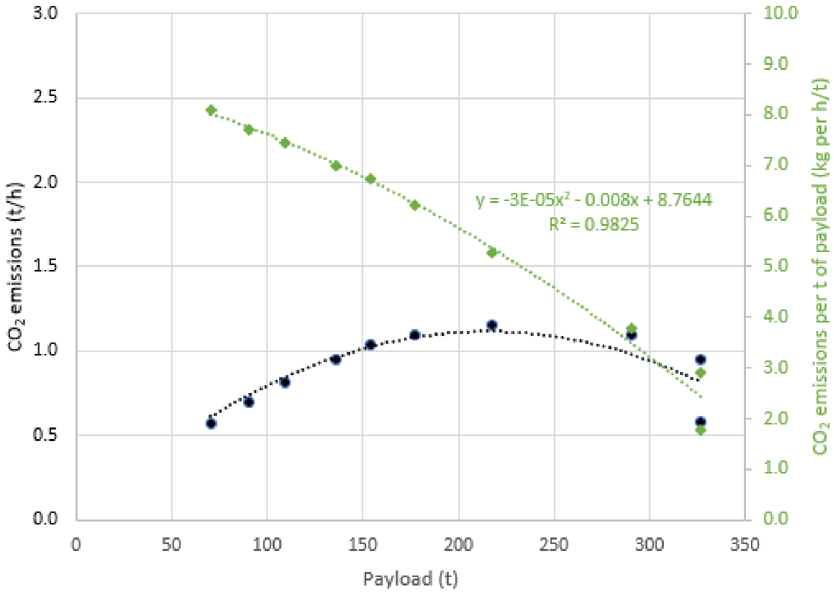


**FIG 5** – $CO_2$ emission rate and $CO_2$ emissions rate per tonne of payload (after Schafer-Frentz, Joseph and Curley, 2019).

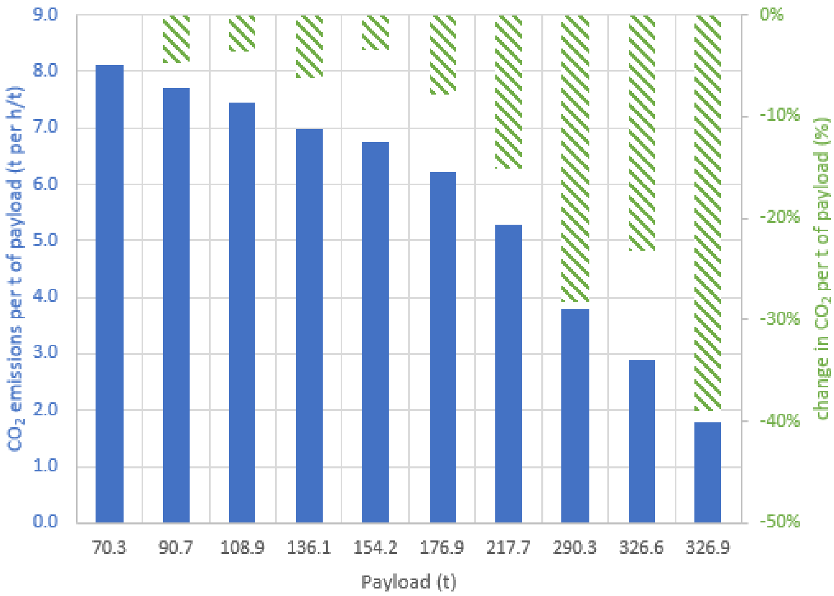


**FIG 6** – $CO_2$ emission rate per tonne of payload and percent change of emission rate per tonne of payload (after Schafer-Frentz, Joseph and Curley, 2019).

Figure 7 shows the annual $CO_2$ emissions assuming 500 truck hours per annum. The highest emitting truck (217.7 t payload) produces 575 t of $CO_2$ each year.

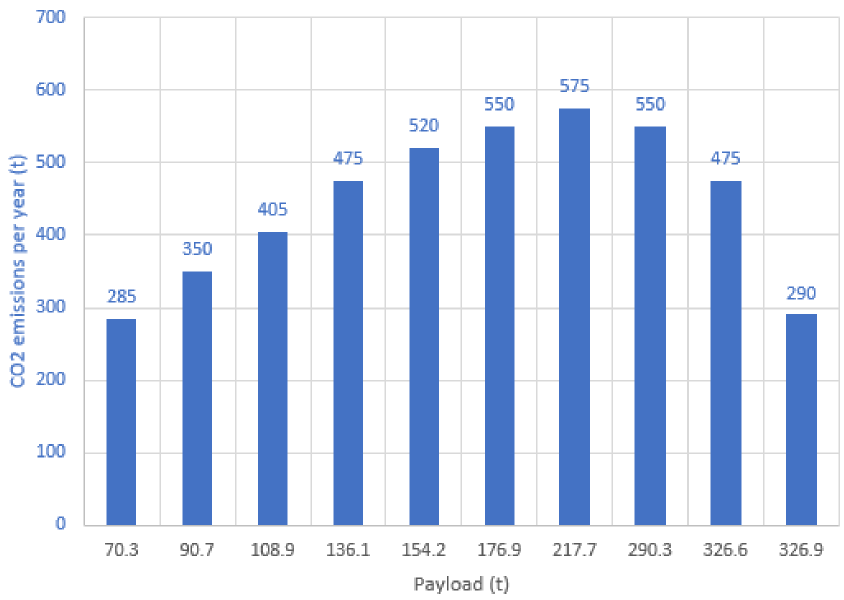


**FIG 7** – Annual $CO_2$ emissions versus truck payload based on 500 hours per annum.

Australia's average $CO_2$ emissions per capita (tonnes per person per annum) is 17.1, and the global average is 4.8 (Worldometer, 2022). So, at 575 t per annum, a 217 t payload truck emits the equivalent of 33 Australians or 120 average world citizens.

### Engine power and engine load factor

The engine load factor is defined as a portion of the rated engine power that is utilised during the work process. It is specific to the equipment type and application/operating conditions, but independent of the equipment size and the rated engine power.

Engine load factor can be estimated empirically. Caterpillar gives values of engine load factors for its trucks as follows:

- Low: 20–30 per cent (continuous operation at an average gross weight less than the recommended. Excellent haul roads. No overloading).
- Medium: 30–40 per cent (continuous operation at an average gross weight approaching the recommended. Minimal overloading. Good haul roads).
- High: 40–50 per cent (continuous operation at or above maximum recommended gross weight. Overloading. Poor haul roads).

From the qualitative descriptions above, we can surmise that the engine load factor is largely driven by payload (GVM) and road conditions.

During acceleration, the engine usually operates at full power with a load factor of 100 per cent. While idling, a truck engine operates at about 10 per cent of full power (Hays, 1990).

Many modern trucks also measure engine load, and this data is available from the OEM (Original Equipment Manufacturer) vehicle management system (for example the VIMS™ system by Caterpillar).

As a measure of 'how hard the engine is working, the engine load factor is, therefore, a derived measure rather than a driver of fuel consumption'. Kecojevic and Komljenovic (2010) found that fuel consumption is a linear function of the load factor, and that fuel consumption increases faster in absolute values for larger trucks (Figure 8). They also calculated the $CO_2$ emission rates (t/hr) of Caterpillar trucks for various engine load factors (Figure 9). The value of $CO_2$ emission ranges from 0.0547 t/hr to 0.1367 t/hr for load factors of 20 per cent and 50 per cent, respectively, for the smallest truck (Cat 770), and from 0.3578 t/hr to 0.8940 t/hr for load factors of 20 per cent and 50 per cent, respectively, for the largest truck (Cat 797B).

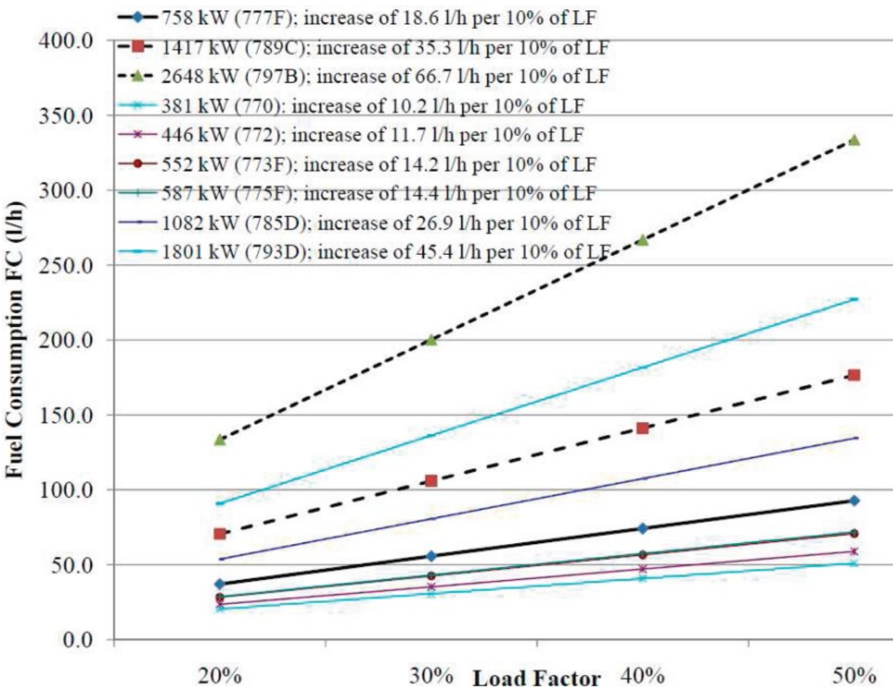


**FIG 8** – Change in fuel consumption as a function of engine load factor (after Kecojevic and Komljenovic, 2010).

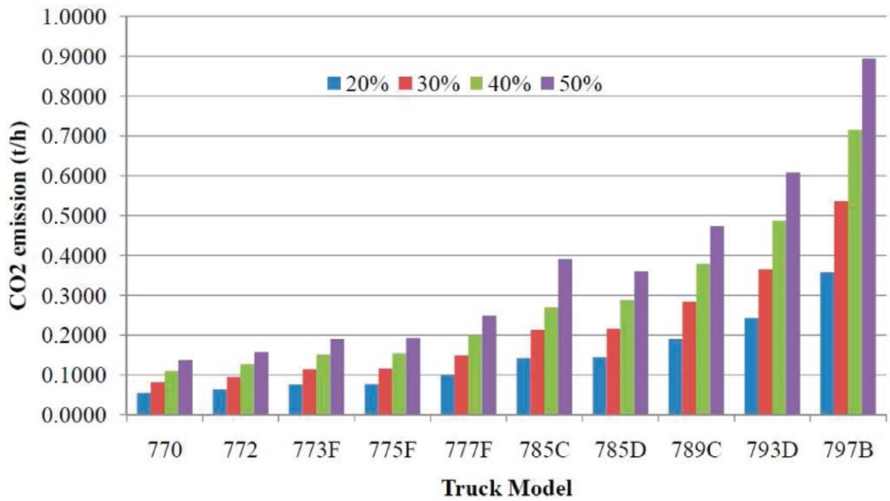


**FIG 9** – $CO_2$ emission rates for Caterpillar trucks under varying load conditions (from Kecojevic and Komljenovic, 2010).

### Tyre inflation

Under-inflated tyres increase fuel consumption by creating a greater tyre-ground footprint and increasing rolling resistance, however, this effect is not expected to be significant for most ultra-class haul trucks if the tyre pressures are maintained within operational limits (600 kPa +/– 100 kPa) (Schafer-Frentz, Joseph and Curley, 2019).

### Payload

Truck payload affects GVW which in turn affects fuel consumption rate (Figure 3). For all values of total resistance, as GVW increases the maximum speed of the truck decreases and fuel consumption increases (Soofastaei *et al*, 2016a). Payload will naturally vary depending on material characteristics, operational variance, and operator. Reducing the variance in payload is desirable not only from a productivity and truck maintenance point of view, but also to reduce fuel consumption (Soofastaei *et al*, 2016a).

Increasing variance (standard deviation) of payload causes fuel consumption to increase for all values of TR for a particular truck (Figure 10) (Soofastaei *et al*, 2016a).

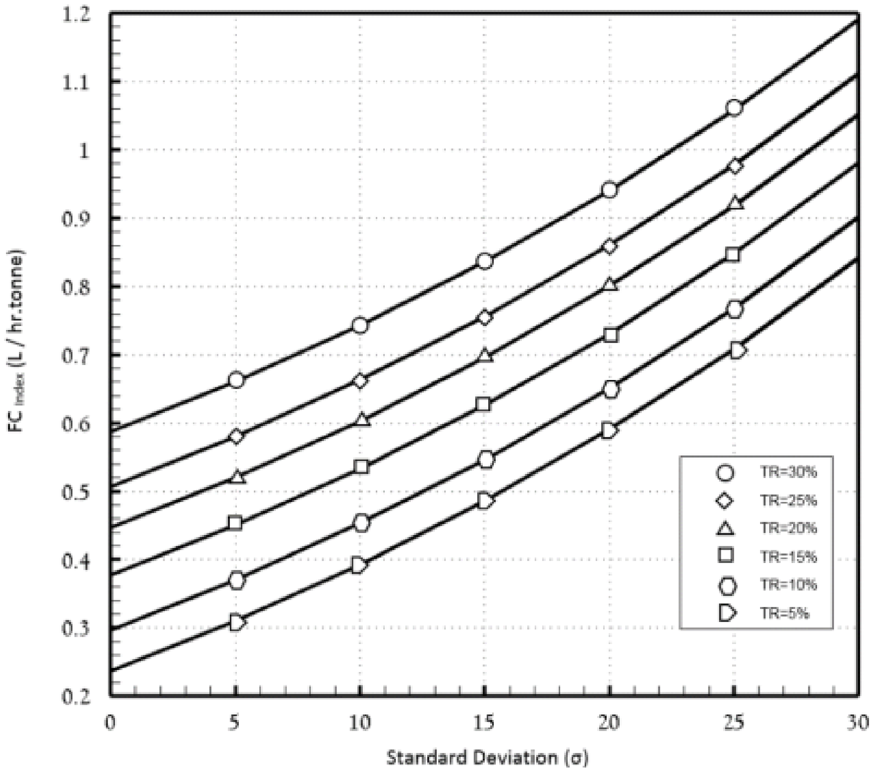


**FIG 10** – The variation of FC with a standard deviation of payload for CAT 793D (from Soofastaei *et al*, 2016a).

With increasing payload variance, fuel consumption index (L/hr.tonne) increases as truck size increases (Figure 11) (Soofastaei *et al*, 2016a). The largest truck had the lowest fuel consumption, however, the difference in fuel consumption between this truck and the next largest decreased as the payload variance increased due to the payload variance (eg 15 t) being a smaller proportion of the payload for the larger truck.

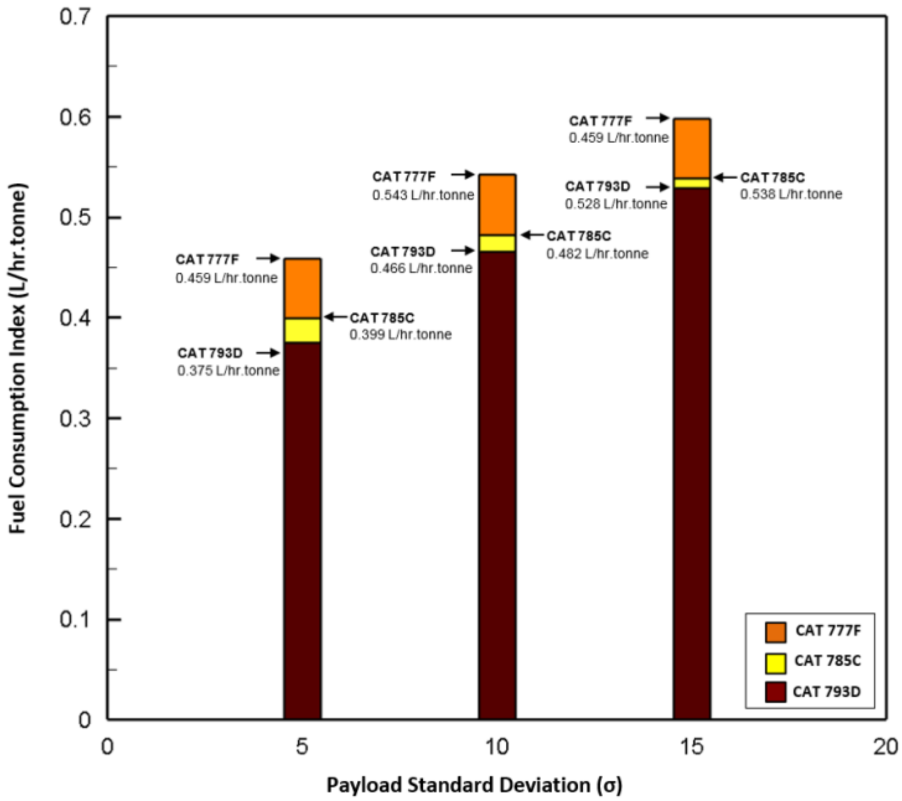


**FIG 11** – Fuel consumption index for three models of haul trucks, TR=10 per cent (from Soofastaei *et al*, 2016b).

Payload variation should therefore be minimised to reduce fuel consumption. Most FMSs have tools to manage payload variation by providing the shovel operator with a real-time payload during the loading activity. Post-analysis of truck payload data from the FMS is also useful for identifying sources of variability and addressing them.

Figure 12 shows the distribution of payload values for a fleet of CAT 797 trucks. The payload values, in this case, were measured by the truck OEM system and recorded in the FMS. The standard deviation of this data set is 21 t with a mean of 333 t.

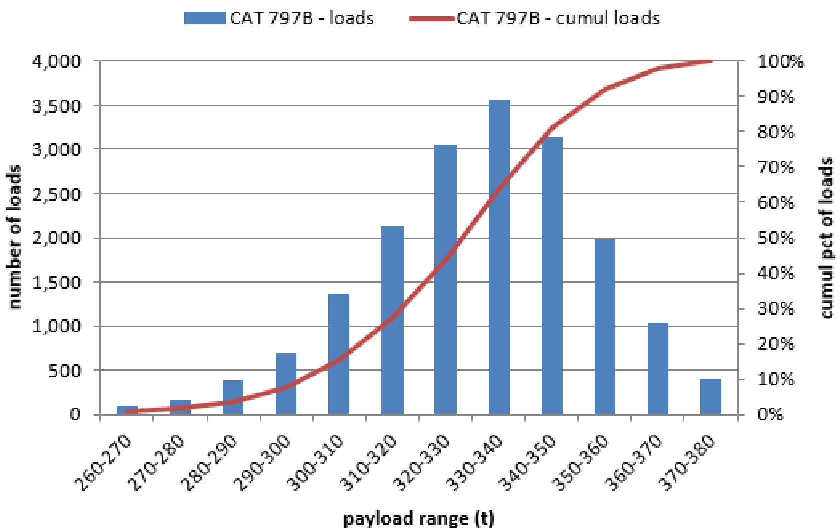


**FIG 12** – CAT 797B payload distribution.

### *Haul route and road conditions*

Haul road Rolling Resistance (RR) is a principal driver for fuel consumption in surface mining operations. RR is a measure of the force that must be overcome to roll or pull a wheel over the ground and is expressed as a percentage of the truck mass. Ground conditions and load affect it – the deeper a wheel sinks into the ground, the higher the rolling resistance (Poke and Lehlean, 1998). As RR increases, greater engine load is required which results in poorer fuel economy.

The environment influences RR. RR will be higher on wet roads and lower on frozen roads for example.

The trucks will experience a range of RR values throughout a typical haul circuit as they travel from in-pit, ramps and main haul roads and onto working dump surfaces. Schafer-Frentz, Joseph and Curley (2019) measured RR in an oil sands mine as follows:

- main haul roads 5.5 per cent
- ramps 7.5 per cent
- dump and in-pit surfaces 8.5 per cent – 11 per cent.

These values are higher than those used by Soofastaei *et al* (2016a) who give the typical values shown in Table 2. According to their study, for typical haul roads, the RR is 2 per cent if the road is hard and well-maintained; on the bench and close to the dump end, the road quality deteriorates and the RR is expected to increase to 3 per cent; during wet periods when the road conditions are worsened, the RR might increase to 4 per cent; finally, under very poor conditions, the RR may rise to 10–16 per cent, however, this would only be over very small sections of the haul road and for short periods of truck operations. In their study, the haul road is considered to have the same conditions as the dirt-dry, but not firmly packed, road and, therefore, a RR of 3 per cent is used in the analysis.

**TABLE 2**

Typical values for Rolling Resistance (RR).

| **Road condition** | **Rolling Resistance (RR) (%)** |
|---|---|
| Bitumen, concrete | 1.5 |
| Dirt-smooth, hard, dry, and well maintained | 2.0 |
| Gravel-well compacted, dry, and free of loose | 2.0 |
| Dirt-dry not firmly compacted | 3.0 |
| Gravel-dry but not firmly compacted | 3.0 |
| Mud-with firm base | 4.0 |
| Gravel or sand-loose | 10.0 |
| Mud-with soft spongy base | 16.0 |

Thompson and Visser (2003) found that RR at a particular point in time is a function of the type of wearing course material (top layer) used, its engineering properties and the traffic speed and volume on the road. Road maintenance is, therefore, an important driver of RR and therefore fuel efficiency.

Schafer-Frentz, Joseph and Curley (2019) investigated the factors related to haul truck fuel consumption and found that Total Resistance (TR) directly affects fuel consumption and emissions, through a mathematically predictable relationship. TR is the sum of Rolling Resistance (RR) and Grade Resistance (GR). They found that even though rolling resistance and haul truck size drive fuel consumption, truck speed also had a strong influence. The practical effect of this was that well maintained (low RR) sections such as main haul roads generated equally high emissions as the high RR in-pit sections due to the high truck speed.

Besides the physical profile of the haul route (distance, lift etc), the number of stops (particularly whilst loaded) along the haul is important, since there is a significant fuel cost associated with stopping and then accelerating a loaded truck. The fuel efficiency gains of reducing the number of stops in a truck cycle were quantified in a case study published by the Australian Department of Resources, Energy and Tourism (2010) which showed that removing an unnecessary stop sign from each truck cycle could reduce fuel consumption across a fleet by 361 kL per annum. This finding also has implications for loaded trucks queuing at a dumping destination, eg a crusher.

### *Truck operator behaviour*

Whilst fuel consumption increases with truck speed, the effect appears less than for light vehicles and motor cars (Thompson and Visser, 2003). Where TR = 0 per cent, a slightly increased rate of fuel consumption with speed was seen due to dynamic rolling resistance effects. At higher levels of TR, this effect was largely obscured by the approximately linear increase in fuel consumption with speed.

By Monitoring and modifying truck operator driving styles, a 7 per cent reduction in fuel consumption was demonstrated over a twelve-month period by Renstrom (2007) who found that each operator influences the result in at least two ways, by driving economically and by the planning of other activities such as queuing, fuelling and breaks. Trucks in a queue often tend to move forward stepwise whenever the truck in front moves. This can result in travelling maybe 100 m with five or more stops and accelerations before reaching the shovel. It is more economic to stand idle until the truck can reach the shovel in one run.

Operators of haul trucks driving on low RR or GR surfaces (eg main haul roads) tend to drive faster than in-pit or dump areas where either RR or GR is much higher (Schafer-Frentz, Joseph and Curley 2019).

## Fuel consumption modelling

To improve fuel consumption, the FMS needs to be able to both measure and model fuel consumption. Measurement is important, because it will allow feedback of performance (integral to any improvement program), and modelling is important because it will allow predictive decisions to be made.

An ideal fuel consumption model would consider all the factors described in Figure 1. These factors can be grouped into four categories, namely truck characteristics, haul route characteristics, truck allocation and operator.

Integral to all the FMSs in the market today is a model of the haul road network. The road network can be quite detailed and is broken into segments that are terminated at three-dimensional nodes, thereby allowing the distance and gradient to be calculated for each segment. An estimate of truck travel time can be calculated for each segment from a model of historical truck travel speeds which is being constantly updated by the FMS.

Having a road network with segments of known grades, distances and historical truck travel speeds provides the basis of the haul road fuel efficiency model.

A literature review has revealed several different approaches to modelling FC, and these are described in the following sections. In common with each of these approaches are the inputs of vehicle mass, total resistance, and vehicle speed. FMS models of fuel consumption should take these variables into account.

Payload is known within the FMS, as is truck speed (derived from moving averages on each segment). Total resistance is not known; however, grade resistance is, and an estimate could be made for rolling resistance for each segment to give total resistance. Some work has been done to estimate rolling resistance using truck suspension data (Schafer-Frentz, Joseph and Curley, 2019) and this could be an area of FMS enhancement.

### *Current FMS models*

FMSs available today have extremely simplistic models to calculate fuel usage. These models use only GR of the road segment, and a corresponding average FC rate (static values added per truck type).

To the author's knowledge, none of the FMSs in the market takes truck acceleration or deceleration into account in modelling fuel consumption – for example accelerating away from a shovel, stopping and starting at a stop sign etc. To be fair, these models were created to estimate fuel tank level (fuel remaining) before OEM fuel tank level sensors were available.

FMSs need to model fuel consumption more accurately to have any chance of being a tool for managing it better.

Several more sophisticated methods for estimating FC exist and these are reviewed in the following sections.

### *Engine load factor based models*

In a review of the literature, Kecojevic and Komljenovic (2010) found that an hourly FC can be determined from:

$$FC = 0.3(LF.PW)$$

Where FC = fuel consumption (L/hr), LF is the engine load factor, 0.3 is a unit conversion factor (L/kW/hr) and PW is the truck power (kW).

Load factor is a function of payload, TR, and speed, so this equation is a simpler version of the total resistance and speed model.

### *Total resistance and speed based models*

Thompson and Visser (2003) modelled the relationship between TR and fuel consumption and found that two models are required to fully evaluate a particular haul.

The TR and speed-based model has potential benefits over the engine load model, namely the segregation of favourable and unfavourable TR segments and allowance for electric drive trucks. Still, the model requires total resistance to be input.

### *Rimpull based models*

The rimpull is the equivalent of all resistance forces which are in opposition to the truck movement. Rimpull is proportional to the truck weight and road conditions and is the force available at the tyre that is required to move the vehicle forward. Rimpull, therefore, is determined by total resistance and vehicle mass.

This force is limited by traction. The difference between the rimpull required overcoming total resistance and the available rimpull determines vehicle acceleration. Rimpull-speed-gradeability curves are provided by the truck OEMs (Figure 13).

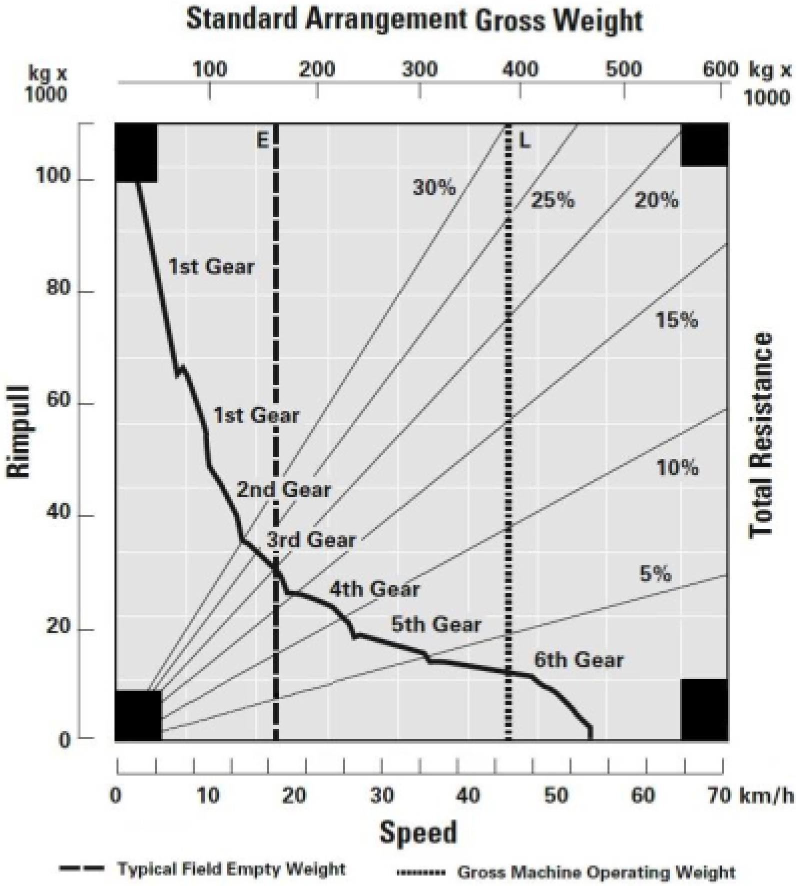


**FIG 13** – Rimpull-speed-gradeability curve for CAT 793D (from Soofastaei *et al*, 2016a).

Kostic *et al* (2019) give the method used by a popular haulage estimation software, Talpac (RPMGlobal). To calculate the fuel consumption, the assumption is that the fuel consumption is related to the percentage of 'rimpull' force used to the maximum (Kostic *et al*, 2019). The software calculates the fuel consumption for each segment of the route separately, for the movement of the full and empty truck, thus resulting in the final fuel consumption for the entire transport route.

Rimpull force can be calculated from OEM data, however, total resistance still needs to be known.

### *Artificial intelligence based models*

Soofastaei *et al* (2016c) developed a comprehensive model based on artificial intelligence methods for reducing fuel consumption of surface mine trucks. They found that truck payload, speed and total resistance are key parameters that affect fuel consumption.

An artificial Neural Network (ANN) model was used to determine the relationship between the key parameters and the haul truck fuel consumption. This relationship was found to be non-linear and complex, however, a function was derived and tested against real-world data. This function was utilised to generate a computerised learning algorithm based on a novel multi-objective genetic algorithm and estimate the optimum values of payload, speed, and TR to reduce the diesel fuel consumption by haul trucks in a surface mine.

The results showed that the best value of payload for the CAT 793D in this mine is between 250 and 270 t. The developed model recommends a truck speed between 13 and 15 km/hr for the CAT 793D in the analysed mine site. Driving in the recommended range of truck speeds can reduce the fuel consumption of haul trucks in this surface mine. Based on the data analysed by the GA model, the optimum range of TR is between 8.5 per cent and 9 per cent.

Whilst this work shows that fuel consumption optimisation is a complex problem, it also shows that it can be solved. There is no consideration given to productivity in the model and it is not possible therefore to comment on the production outcomes if the model recommendations were applied. It is expected however that the findings of Awuah-Offei, Osei and Askari-Nasab (2012) apply, and that fuel consumption reduction is at odds with optimal production outcomes. This is not to say that the FMS has no role to play in reducing emissions.

Again, this model requires total resistance as an input.

### *A better FMS fuel consumption model*

To be successful, a better FMS FC model would need to:

1. Be simple to implement and maintain.
2. Accurately reflect real-world road network elements such as stop signs.
3. Consider acceleration and deceleration.
4. Include truck speed, total resistance, and truck type.

Any of the models discussed previously could potentially be used by an FMS to model fuel consumption.

If the engine load factor is a product of total resistance and speed, then all the models considered require speed, TR, and truck type as inputs. The engine load factor approach modelling approach would require the engine load factor to be known for each road segment. Load factor could be estimated empirically (see Engine power and engine load factor) being a function of payload, total resistance, and speed (acceleration) however in practice this may not be accurate enough.

The other models which use total resistance would require an input of rolling resistance since gradient resistance is already known. Rolling resistance could be estimated for categories of roads (or individual roads) using guidelines or inferred using OEM sensors.

Where an OEM system is producing engine load factor data, the FMS could use this data directly to model load factors per truck type, for each road segment. These values could be extrapolated and used to build a table of load factors per gradient range, for each road type and truck type.

Acceleration is an important piece of information within the model. The FMS road network would need to include a method of designating the starting and ending velocity on each road type. For example, a road segment that terminates at a shovel would have an ending velocity of zero, and the same road segment for the outbound journey would have an initial velocity of zero but the truck would be under 100 per cent engine load. Similarly, any stops along the route could be modelled.

## THE ROLE OF FMS IN REDUCING $CO_2$ EMISSIONS

Having established that haul truck emissions are affected by driver behaviour, haul road and truck characteristics, what can the FMS do?

Even though FMSs are comprised of sophisticated hardware and complicated software, really what they do is enhance decision-making. In general, an FMS can enhance decision-making at 3 levels:

1. Real-time (minute by minute).
2. Procedures (define the best way to operate).
3. Policies (define how the mine runs overall).

### Better real-time decisions

As a real-time decision enabler (or maker in the case of automated truck assignments), the FMS acts as a tool in the Operational Plan (OP) period. Figure 14 shows that the objective of the Operational Plan (OP) is to comply with the Short-Term Plan (STP) whilst maximising equipment productivity. Maximising truck and shovel productivity are where the FMS optimisation algorithms have focused thus far.

To maximise truck and shovel productivity, the FMS optimisation algorithms evaluate each truck assignment (whether to a shovel or a dump) to minimise truck queueing and shovel idle time. For example, when a truck is requesting an assignment to a shovel, the FMS calculates the estimated time of arrival (ETA) of the truck at each of the possible shovels and builds a future queue of trucks at each shovel (since it has previously calculated the ETA of every other truck). These queues are used to calculate the relative production benefit of each option (in the form of truck queue time, empty travel time etc) and the truck is assigned to the most optimal destination subject to constraints. Also note that this is the case for loaded assignments, and that loaded trucks can also queue at dumping destinations.

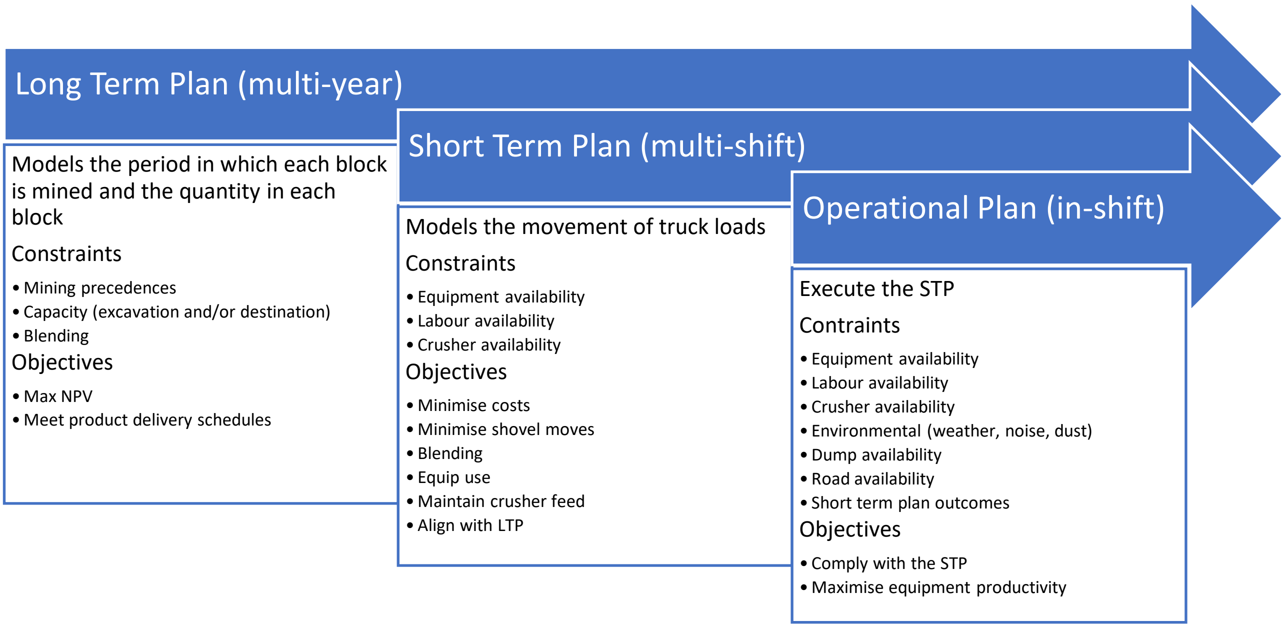


**FIG 14** – Generalised mine planning horizons.

The constraints are added to the FMS to force the optimisation algorithms to consider the STP objectives. Constraints include crusher feed rates, blending requirements, material priorities, shovel priorities etc. In this context, the FMS is 'micro-optimising' by making opportunistic truck assignments that improve productivity whilst delivering the STP outcomes.

### *Assigning trucks for lowest fuel cost*

To the author's knowledge, no FMS currently explicitly considers operating costs in its optimisation algorithms. In theory, the FMS could use activity-based operating costs to evaluate truck assignments and choose the destination that yields the lowest operating cost. For example, a cost model for truck hauling could be worked up which includes the major costs of fuel, tyres, labour etc and these costs applied to the haul assignment being evaluated. The optimisation engine would then choose the destination which gives the lowest operating cost per tonne. Similarly, the algorithms could focus purely on fuel cost and select assignments to minimise this.

There are at least two reasons why this is a bad idea. First, optimising specifically for fuel cost has already been shown to be a potentially conflicting objective with productivity (see Truck Allocation).

Also, optimising for operating costs will have the same outcome as the existing productivity optimisation algorithms. Assuming the unit costs are equal (cost per hour), and the output is increasing by 5 per cent per unit of time, we would expect a 5 per cent reduction in unit cost. So, optimising for minimum operating cost is achieved through the existing algorithms.

Second, it is not the job of the OP to select loads based on the lowest cost. The decision of which blocks to mine has been made by the STP. The STP has also dictated in which period the blocks are to be mined. A truck allocation algorithm that is based on operating cost alone would deviate too far from the STP or must be constrained so much to comply with the STP, that the operating cost objective would be lost.

This is not to say that no 'micro-optimisation' for fuel is possible within the OP. The following sections describe several.

### *Truck speed recommendations*

Remembering that the key drivers of fuel consumption are total resistance, truck characteristics and truck speed, what can the FMS do to control speed?

The FMS calculates the ETA for the truck at its destination to maximise production by assuming average travel speed on the best path. In many assignments, truck queueing at the destination is inevitable and predicted. In these cases, the truck is burning unnecessary fuel travelling at normal speed to the destination, only to sit in the queue since higher speed is related to higher fuel burn.

As previously discussed, the travel speed of the haul truck is a major driver of emissions, particularly on main haul roads where speeds are high.

The role of the FMS here is to recommend slower speeds without negatively impacting the optimal ETA, thereby saving fuel. The speed recommendation could be shown to the truck operator, or indeed the truck speed could be limited through an interface to the OEM system. Such vehicle intervention systems are currently available, and speed limitations are applied to avoid tailgating or excessive speeds down ramps.

How the travel speed is moderated over the haul warrants some further work. The benefit of slowing the truck on a particular segment of a road will be greater than on others, depending on potential speed, TR, and truck characteristics. Slowing the truck on higher speed sections (for example well maintained flat roads) will have a greater impact on fuel economy than slowing in-pit or on-dump sections.

Logically, all trucks *en route* to the destination will be slowed to maintain gapping and avoid a bunch of trucks arriving at the destination. The existing optimisation algorithms will still be in play, assessing alternative assignments for each truck to maximise productivity even while they are slowed to conserve fuel, and the new assignment may require the truck to speed up because the ETA does not indicate the idle time at the destination.

### *Best truck for fuel efficiency*

Using a model of fuel consumption for each haul route for each truck type (or even each truck if data is available), the optimisation algorithm may recommend specific truck types (or even specific trucks) for different hauls to minimise fuel. If excessive truck capacity is available, the FMS may recommend which truck types (or even specific trucks) to park up.

In general, this approach would favour the use of larger trucks over smaller ones. This approach may result in certain routes being preferred by specific truck types to conserve fuel – for example, electric drive trucks may be favoured on routes that are longer downhill, or truck sizes may segregate.

### *Idle time prediction*

In a similar way that the FMS forecasts truck arrival times, it also predicts idle time. If a shovel changes status to Delay and the Delay has a predicted duration, this delay time is built into the truck forecasts. So, for trucks that are in the queue at the shovel on Delay, the FMS could notify the truck operators how long the delay is expected to last and (subject to procedure) whether to turn off the engine.

## Designing and monitoring operational procedures

Operational procedures are important in defining how the mine operates. For example, 'turn the engine off if idling for more than 2 minutes is a procedure that would reduce fuel consumption. The FMS should be able to assist with the prediction of idle time (see Idle Time Prediction).

As discussed in Truck Operator Behaviour significant fuel savings can be made by using the FMS to identify and monitor improvement opportunities. The procedure given which aims to reduce fuel by reducing the number of truck movements in a queue can be implemented using the FMS. A simple message saying 'move forward in the queue now' could be given to all trucks at once.

Supplying the truck operators with meaningful fuel consumption metrics, and performance monitoring around them will give fuel savings. Going beyond a simple litre per hour metric, the FMS can contextualise the data for the operator:

- where in the cycle is the most fuel burned (map view)
- idling fuel burn
- league table showing most economical trucks (or operators).

Another important procedure relates to truck payload. Truck payload variability negatively impacts fuel consumption (see Payload). The FMS provides the shovel operator with timely information on the payload of the truck being loaded, allowing them to comply with any payload procedure and reduce variability.

## Defining policies

Policies define how the mine runs and are more holistic than procedures. FMSs are today used to enforce or implement many policies. For example:

- Ultra-class trucks are to be used in preference to smaller trucks. The FMS provides visibility to truck fleet utilisation by truck type, both historically and in real-time.
- Shovels are to be parked if hang time exceeds $x$ mins per load. The FMS can provide shovel hang time metrics to allow this policy to be enacted and a decision to be made quickly.
- Large trucks should only be loaded by large shovels. From productivity and a safety point of view this makes sense, but it will also reduce truck fuel consumption since there will be less loading time (and idle fuel usage).

The artificial intelligence model previously discussed was trained on FMS data, and the conclusions of the study could be taken as policies to reduce fuel usage. For example, haul roads should be designed for the total resistance of 9 per cent.

Similarly, a mine site may have a policy that truck speeds are not to exceed 20 kph in-pit (designed to reduce fuel consumption and maintenance costs). The FMS can be used to monitor real-time compliance to the policy, and alert operators and supervisors alike.

# CONCLUSIONS

The mining industry must reduce $CO_2$ emissions in the short-term. Haul trucks contribute over half of the mobile mining fleet emissions in an open pit mine, and since $CO_2$ is related to fuel consumption a short-term focus could be to reduce haul truck fuel usage.

The key drivers of haul truck fuel consumption are total resistance (gradient plus rolling), speed, and vehicle characteristics. These factors should be modelled within the FMS if the system is to be effective in reducing fuel consumption.

Whilst reducing fuel consumption may be incompatible with optimal truck productivity, there are ways in which the FMS can be applied to achieve this while minimising the productivity impact. The real-time truck assignment algorithms could be modified to intelligently recommend the reduced speed and better match trucks to haul routes to reduce fuel consumption. In addition, the FMS is a powerful tool that can be used to implement and monitor operational procedures and policies to reduce truck fuel consumption.

# ACKNOWLEDGEMENTS

The author acknowledges support from Hexagon to submit this paper for publication by the AusIMM, in particular the contribution of Hexagon colleagues Sean Perry and Jon Olson.

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# Impact of COVID-19 – case studies including innovative ways of working, what has changed and what needs to change

# How innovation trumps isolation, one mine at a time

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## ABSTRACT

Shortly after the World Health Organization declared COVID-19 a pandemic, the words 'social distancing' entered the global lexicon. The effects on the public of social distancing, lockdowns and travel bans are well documented. They also present unique challenges to the mining industry.

For Hexagon's Mining division, this posed a significant challenge. We have built our business – our reputation – on regional, customer-focused service and support. True to Hexagon's values of being professional, engaged and customer-focused, we innovated to overcome the distance dilemma.

Making automation a reality for our customers by implementing and supporting technology projects from a distance has helped deepen partnerships by making it easier to tap into our global experts' knowledge.

For example:

- Body cameras were at the heart of a successful safety installation at a mine in Ghana. Despite the pandemic lockdown preventing travel by our South Africa staff into Ghana, the project was completed with minimal delays.
- Traditionally, implementation and training for our Blast Movement Technology solution is done physically at the mine site. With COVID-19 travel restrictions in place, we have developed and successfully employed a remote deployment methodology.
- Using RealWear™ headsets in the field and utilising the knowledge management platform from HINDSITE, Hexagon workshop technicians have been able to support hardware repairs in local offices, which once would have been undertaken in our overseas offices.
- The same headset is being utilised to provide clients with remote mentoring sessions by our skilled team members who are based in our regional locations and unable to travel to their site without extensive quarantine commitments.
- In instances where our team members have had to travel to site, we have provided consumer technology such as VR headsets to help them cope with long periods of quarantine to and from site.

Post-pandemic, Hexagon will continue to refine and improve this method of working.

## INTRODUCTION

Shortly after the World Health Organization (WHO) declared COVID-19 (COVID) a pandemic, the words 'social distancing' entered the global lexicon. As governments, healthcare organisations and the public scrambled to respond to the crisis, one thing became abundantly clear: isolation was and, for most of us, still is the key to maintaining our health.

Hexagon, a leader in sensor, software and autonomous solutions, was well positioned to pivot and assist in the global response to the crisis. From simulations of the effects of social distancing to mapping the spread of COVID in real time, to drone technology delivering COVID tests, to providing communities with transparent geospatial information, Hexagon solutions continue to help ensure that innovation is readily available to those who need it most during uncertain times.

The effects on the public of social distancing, lockdowns and travel bans are well documented. They also present unique challenges to the mining industry.

## IMPACT OF COVID-19 ON HEXAGON'S SUPPORT AND SERVICES

Most mines are under immense pressure to remain safe, sustainable, and productive with many relying on staying ahead of the technology curve to achieve this. Human interaction is at the heart of implementing new solutions and training staff. But how do you connect with technical experts who are prevented by laws and lockdowns from entering your country, let alone your mine?

For Hexagon's Mining division, this posed a significant challenge. We have built our business – our reputation – on regional, customer-focused service and support. True to Hexagon's values of being professional, engaged and customer-focused, we innovated to overcome the distance dilemma.

### Problem statements

In this paper, we discuss five specific areas which had an impact on Hexagon's ways of working in the Services and Support area during the COVID-19 pandemic.

#### *Hardware installations on site equipment*

Part of the implementation for a safety technology solution at a mine in Ghana required the installation of hardware components, as well as training site personnel.

Typically, this would have meant that Hexagon personnel from South Africa would have travelled to site to install the hardware and conduct training. Travel restrictions meant that our staff were unable to undertake that travel.

The challenge was to find a way to leverage local partners in Ghana to carry out the works with minimal impact on the schedule.

#### *Implementation of new technology solution*

In the solution area of Blast Movement Monitoring, for all our new customers, the first step is to conduct site implementation and training for the customers' geology team.

Traditionally, this was conducted via a site visit from our consultants to the mine site to help the team understand the various steps from blast design to measurement, as well as interpreting the results using the software solution. With various restrictions in place due to the pandemic, site travel in some countries and mines became untenable.

The challenge was to continue with implementation and training with the constraint of consultants not being able to be on-site physically.

#### *Hardware repair*

Many of our solutions leverage our proprietary hardware. Hence, there is equipment from the field that occasionally has to be returned to our offices for repairs.

Under normal circumstances, quite often, equipment would be sent back to our head offices in Switzerland or United States, for these repairs to be undertaken.

The challenge was to find ways to leverage remote expertise in the company to support repairs in our local regions, closer to the customers.

#### *Remote mentoring*

Pre-COVID, our field technicians would make regular client site visits to troubleshoot and resolve issues while providing training to client staff.

With the long quarantine durations mandated for interstate and international travel, most sites trips are extended by weeks of non-productive downtime. It was difficult to justify a site visit unless the trip was for an extended period.

The challenge was to find alternative ways to continue the support without travel to site

#### *Helping people through extended quarantine*

Hexagon staff travelling to sites, both domestically and across borders is a regular feature for our support and services team. Traditionally, this was a routine activity, which had to be scheduled in consultation with the customer sites, meeting any site requirements for induction and Personal Protective Equipment (PPE).

Once site travel became an option, there were new mandates relating to quarantine measures, based on country and site requirements, and one of the more extreme instances experienced was Mongolia. One of our team members had to quarantine for two weeks upon arrival in Mongolia, followed by another seven days at the mine site before they could even commence a day of work on-site. On completion, our team member had to quarantine for 14 days upon return to Australia. While the nature of the work justified the need for the travel to the client's mine site, it should not be ignored that in total, a team member had to quarantine for a total of 35 days just to perform 19 days of work.

The challenge was to manage our personnel's mental health and morale during long solo isolation periods to meet quarantine requirements.

## INNOVATIVE SOLUTIONS

What innovative tools and process changes did we employ to solve these problems?

### Hardware installations on site equipment

For the technology implementation at the Ghanaian mine, installation works at the site were significant. There were over 220 pieces of hardware installations required for the collision avoidance solution, as well as installation of technology for operator alertness and personnel alerts.

Local partners had to be skilled-up in the installation and training of a variety of solution components, which they had not done before. Technology, such as remote body cameras (Figure 1), helped ensure that installation and training stayed on schedule.


**FIG 1** – Hexagon subcontractor using RealWear™ headset for remote installation assistance and Quality Assurance.

Willingness to embrace change was key to the successful installation, coupled with the mine site staff's willingness to adapt to the limitations imposed by COVID was hugely important.

In the words of a senior executive from the client operation, 'The resourcefulness displayed by everyone involved was impressive. It ensured that we completed the project with minimal delays – almost to the original project dates!'

The use of similar technology had been used during the planning and execution stages of projects involving hardware installation (Figure 2).


**FIG 2** – Testing the HINDSITE platform and RealWear™ headset in preparation for a client training session.

## Implementation of new technology solution

We have a well-defined and proven playbook for new site implementations, with the assumption that consultants would be on-site. This included training on the process and solution, familiarisation with the equipment and technology, demonstrating the usage in the field and then letting users becoming familiar under guidance.

We had to rewrite the playbook, eliminating the consultant's physical presence out of the equation. This meant adjusting the training duration to accommodate remote training, as the conversations were more structured and planned than when consultants are on-site with the user teams.

Where possible, local staff were skilled to provide training, which was found to be beneficial particularly to teach physical concepts and practical tips and tricks. However, this meant that additional training had to be planned for the local staff. Our consultants had to consciously change their teaching methods and techniques to adjust.

Different technologies were adopted for the remote interaction, depending on the customer and site preferences.

## Repair of hardware

In addition to using RealWear™ headsets in the field and utilising the knowledge management platform from HINDSITE, Hexagon workshop technicians are leveraging Smith Optics™ safety glasses with Google Glass™ Enterprise Edition (Figure 3) to support hardware repairs in local offices from Almaty to Brisbane and Perth to Balikpapan. These repairs might once have been undertaken in head offices in Switzerland or the United States.

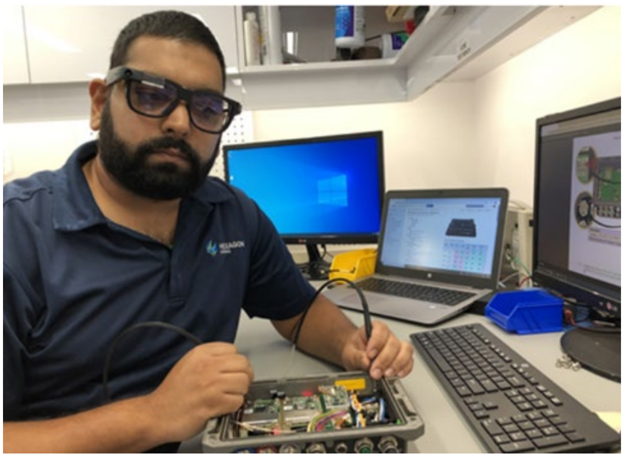

**FIG 3** – Workshop supervisor utilising Google Glasses.

Our experience with deploying some technologies to leverage remote expertise demonstrated that it was possible to use them to support repairs in our regions, closer to the customers.

In many cases, this has reduced hardware repair and return times, lowered costs, enabled the sharing and capturing of implicit knowledge in a corporate repository, and ultimately improved the customer experience.

### Remote mentoring

The alternative we implemented was to send RealWear™ headsets to the client mine sites and initiated remote mentoring sessions to guide them through the troubleshooting process and/or to provide any required training.

The entire session is recorded and transcribed for future reference by all parties, aiding sustained learning.

### Helping people through extended quarantine

In addition to supporting the teams who were traveling, with ensuring the appropriate documentation and helping them meet any pre-requisites, it was recognised that long periods of isolation during quarantine can be a challenge for most people. In addition, a number of different quarantine periods for a trip, as in the case illustrated earlier, meant that any additional support to the team members would be welcome.

One of the other innovations was to assign a VR Headset to all team members that had to do any quarantine time during their trip. This was meant to be a source of entertainment during those periods of self-isolation. While some people utilised this technology more than others, everyone appreciated the organisation going an extra step in ensuring their well-being.

'Having the Oculus VR™ headset in quarantine (Figure 4) not only provided me with some entertainment (where the local TV was in a different language) but also as a means to encourage me to keep up with physical exercise while being confined to a single room for the quarantine period.' Senior Project Engineer.

**FIG 4** – Hotel room in Mongolia during a 14-day quarantine stay (note VR headset on the table).

## LESSONS LEARNED

The changes in the ways of work and leveraging innovative tools and techniques described in this paper have evolved with time, as many of these had to be applied for the first time, and there was a lot of learning and feedback to make them work well. Some of the key lessons learned include

- Remote work and support means doing things quite differently, so significant contingencies and buffers for planning and coordination are required.
- Teaching physical concepts is much more difficult remotely.

- Willingness to embrace change was key to the successful installation, coupled with the mine site staff's willingness to adapt to the limitations imposed by COVID-19 was hugely important.
- Both Hexagon and Customer Site personnel needed to adapt to new ways of working. Having said that, we have been encouraged by the willingness of mine site personnel to make this happen.

## CONCLUSIONS

Overall, the COVID-19 pandemic has forced thinking of alternative ways of working, to suit the operating environment. This has resulted in the adoption of significantly different ways of working, which has also opened up new possibilities as these could continue to be leveraged as the constraints imposed by the pandemic are slowly removed. We have found areas where the new methods ultimately improved the customer experience, and these would obviously be retained and evolved further. Post-pandemic, Hexagon will continue to refine and improve this method of working.

## ACKNOWLEDGEMENTS

The authors acknowledge support from Hexagon to submit this paper for publication. We are also grateful to the following Hexagon colleagues that took advantage of the VR Headset and were generous to share their learnings and experiences:

- Peter Parnell – New Caledonia
- Mark Quinn – Mongolia
- Peter Kiem – New Caledonia
- Shaun Needham – Kazakhstan
- Greg Krause – New Caledonia

# Integration of open pit operations

# Transition from owner mining to contract mining – a structured tender process using a holistic approach

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## ABSTRACT

This paper discusses the operating model within Endeavour Mining Corporation regarding the options of owner mining against contract mining. It's based on the concept of 'are we carrying out the most efficient mining' regarding cost and performance. This question can only be answered through an extensive contract mining tender process and conducting a comprehensive comparison to evaluate the transition viability. Karma mine, located in Burkina Faso, is the subject of this paper.

The Tender model included sales of equipment, parts and tooling associated with the traditional Load and Haul Schedule of Rates (SOR).

The methodology focused on the structure of the tender process, which consists in the development of a SOR based on haulage profiles and mining schedule. This is a new Endeavour framework that allowed the bidders to compete on an even level. This was performed using a systemic analysis which encompassed all the elements involved in a transition such as the Human Resources (HR), taxes and customs implications, directs and non-direct costs (Finance and interest on assets, depreciation, insurance), contractor risk profiling. It was assessed against the normalised internal pricing tender-based. This paper gives an insight into how a West African mining operation addressed the sensitivities of the HR functions against the commercial aspects. This was one of the most successful transitions in West Africa and was carried out during the COVID crisis.

The outcome of this project has created value by detailing a lesson learned model for both commercial and HR aspects. This discusses the commercial comparisons along with the internal model with the underlying HR functions as a risk factor that could have jeopardised the continuation of the operation. This resulted in a smooth transition with the additional cash flow generated by the sale of the parts and equipment turned Karma into a viable operation.

## INTRODUCTION

The Karma project is located in north-central Burkina Faso, near the city of Ouahigouya, approximately 185 km north-west of Ouagadougou, the capital city of Burkina Faso. This project was purchased by Endeavour Mining in 2016 and operated as an owner mining operation. Through financial analysis of a reduced resource model, the project reflected a zero cash operation. This did not justify the business case to remain as an ongoing entity, and so alternatives were tabled by the senior executives. One of these alternatives was 'are we carrying out the most efficient mining with regards to cost and performance?'

Therefore, the directive was to confirm Endeavour operating model by analysis of owner mining to contract mining with a review initially based on costs and cash flow generation. As the process developed, all the aspects that might be impacted by this transition were highlighted and added to the assessment summary as an influential factor. As Snowden (2006) stated, there are several corresponding instances where the use of contractors in a mining operation is unlikely to be the best option.

> *[…] The development or introduction of a new operation into an existing owner mining culture. Hence the challenge of making this transition a success, Karma being an owner mining model. Therefore, the directive of cash flow could not only be considered in this instance.* Snowden (2006)

# PART I – TENDERING PROCESS

## Pre-evaluation

Karma mine is located in North-central Burkina Faso, near the city of Ouahigouya. The zone is considered a red zone due to the records of terrorist activities within the region. The mine is 90 per cent owned by EDV and 10 per cent by the government of Burkina Faso. The company acquired the deposit in the first quarter of 2016, commencing mining the oxide ore in the same year, with the first gold being poured in April 2016. The mining method utilised is a conventional open pit mining. Medium size mining equipment are used, including 120 t diggers with 90 t Haul truck to meet the production requirements. The first stage of the scoping study was to investigate on the location, suitable contractors with the ability to finance and support a mining operation transition without any disruption.

The key items in the process consisted of the following items and the appropriate tender return schedules were developed around them:

- sell the assets as part of the tender requirements
- sell all spare parts for mining works to the contractor
- transfer mining personnel to the contractor where required (exclude technical services and management)
- evaluate non-direct personnel and inform controlling departments of requirements/cost benefit on reallocating to other areas
- identify key business personnel and re-assign to another EDV entity
- evaluate the financial/commercial aspect of the tender returns against cash flow model/internal budget
- reduce contractual risk for EDV through amendments of a standard mining contract to reduce financial exposure to EDV.

The second stage of the scoping study was carried out to look at the key risks of the proposal. These are highlighted below:

- low LOM (Life-of-mine): two years – create high cost due to profit return of contractor over this short period
- assets – sale of assets can create high cost on contractor pricing through low utilisation on completion of the contract period
- labour – potential industrial disruption.

The outcome of the scoping study produced a high-risk exposure to EDV, which had to be taken into account to the alternative of maintaining the status quo. For the remaining LOM, however, the production schedule needed additional equipment to be met. This would require either new capital expenditure (CAPEX) or an hourly rental from a contractor over the period. The investment analysis in relation to the return was not deemed commercial effective. After the key risks and benefits were discussed, the senior EDV executive management confirmed that the tender process should occur as the next step of the assessment.

In order to ensure the orientation of the process, a high-level timeline with major milestones was set to ensure that. Table 1 shows the structured approach in the process so that key stakeholders were informed of the requirements and outcomes.

**TABLE 1**

Approach description.

| Phase | Approach© |
|---|---|
| Expression of interest and pre-tender qualification | This will be the first stage to indicate suitable contractors that will satisfy the tender process sufficiently. |
| Data collection | This will form two parts:<br>• Schedule information – this is the information required to populate the schedules.<br>• Tender Information – in conjunction with part 1 the internal tender information will use information gathered to assess the comparison tender. |
| Tender issue | Issue all tender/contract documents to ensure that a suitable return is received which can be sued to assess financial and operational risks and to allow a gap analysis of maintaining an owner operator to identify areas of improvement. |
| Internal tender | Carry out a comparable internal tender using RFT (Request for Tender) information with actual or indicated data applied. |
| Tender assessment | The assessment will identify:<br>• risks<br>• financials<br>• contractual<br>• human resources<br>• operational. |
| Report summary and recommendation | This detailed the financial, risks and viability of the tender process and recommendations to the EDV Exec team. |
| Implementation | Turns the process from theory into actual implementation. |

To lead this process to a successful outcome, a dedicated tender team was formed to deliver the tender submittals, tender returns, internal costings, evaluations, and recommendations. The structure included the following:

- Contract Owner
    - define the scope, timing, materials, equipment, standards, and all technical information applicable to the contract
    - make recommendations to Supply on tenderer selection, term, and structure of a contract
    - act as the EDV representative for the contract
    - accountable for ensuring compliance of the contractor
    - accountable for the contract schedule
    - ensure that the contractor complies with the RFT
    - ensure that the scope, time, or cost aspects of the contract are not varied without the appropriate approvals
    - ensure that the RFT is fairly and appropriately presented to Executive committee members.
- Contract Officer
    - ensure all activities are compliant with EDV tendering policies
    - ensure that submittals and receivals are issued and returned on time

    - prepare tender and contract documentation
    - assist contract owners with development of scopes of work
    - provide advice to the contract owner of procurement methods and strategy
    - provide advice to contract owners on any opportunities within the existing supply chain activities
    - arrange the logistics of tenders and award of contracts
    - assist in tender negotiations with potential contractors
    - identify period contract opportunities
    - maintain sound contract documentation and records
    - audit contract owners monitoring of legal compliance of contractors
    - process tender variations
    - ensure appropriate closeout of tender
    - maintain a register of and records of works.
- Contract Admin
    - arrange document control
    - maintain all correspondence and minutes of meetings
    - ensure records of all information is maintained on EDV filing system
    - act in absence of contract officer
    - ensure weekly reports are issued in time
    - assist in site visits and issuing of RFT information and subsequent coordination's
    - assist in financial assessment.
- Tender Engineer
    - work with all team to ensure technical submittals are issued on time
    - work on LOM and schedules to ensure in a suitable format
    - collect internal pricing data
    - carry out internal tender in line with contracts methodology detailing all assumptions and referenced material
    - coordinate site visits
    - carry out financial assessment and presentation of tender reviews
    - focal point of technical clarifications
    - collate tenders received questions on pricing
    - assist in tender negotiations with potential contractors
    - identify opportunities.

### *Expression of interest (EOI) and prequalification*

A provisional scope of work was agreed as a result of the review of the mining works that are going to be subcontracted. Subsequently, an internal preselection of contractor which are deemed competent to perform this work was done based on criteria such as: a track record of mining contractor experience, presence in the region, financial capability.

Eleven Contractors were identified for financial capacity and capability in the industry also with west African experience. The EOI intends to give an overview of the project by indicating to the bidders the background of the project, the general scope, details of the information to be submitted, communication interface for clarifications, proposed timeline of the tender process. In addition to the EOI. A pre-qualification questionnaire (PQQ) is included in the EOI package. PQQ is the first step of a tendering process; it intends to exclude from the process, companies unable to meet the criteria.

1. Do you have an existing Burkina registered entity?
2. Are you currently operating in Burkina Faso/West Africa Region?
3. Are you willing to operate within the Karma Riverstone region?
4. Do you have experience in a similar mining Contract of 1 million $m^3$/month operations?
5. Are you willing to purchase the existing equipment, spare parts and tooling associated with the Contract Mining works?
6. Can you raise up to US$30 million for purchase and working capital for the Contract Mining works?
7. Are you willing to tender on a short mine life, excluding any additional resources added to the LOM at this stage?
8. Are you willing to take over the existing workforce where required?
9. Are you willing to negotiate on an open book basis? should you progress through to Contractor selection after any submittals/correspondence?

The EOI was issued to eleven contractors. Out of these, three did not respond, and another three declined due to security reasons. The six remaining replied favourably to the PQQ. An invitation to tender (ITT) was therefore, viable to continue.

## *Data collection and analysis*

Data must be collated and analysed through this process to ensure that a reliable RFT is issued to the bidders and facilitate the comparable internal tender. As defined in the objectives, the transition aimed to transfer all the physical mining works to the contractor. Therefore, any aspects that might have an impact must be considered during the data collection. This includes assets, taxes, customs duties, legal procedures, outstanding contracts that might be impacted by the transition, and scheduling.

To carry out the pricing assessment, key information is required to ensure an RFT is submitted, accurately and there is no ambiguity in the Tender direction. Therefore, a formal request is sent to each relevant data provider. This ensured that accountability and timely responses were received and that the accuracy of the information provided sat with the department issuing it. The progress was tracked through a template developed for this purpose, as shown in Table 2.

### Asset Inspection and Valuation and Parts

The sale of the equipment was an integral part of the process and the inspection of the equipment was essential to review their current condition and valuation. A full inspection and identification of equipment, ancillary tooling and any temporary facilities that would/could be sold to a contractor and required to operate was carried out.

As a high-value item, an internal and independent external inspection and valuation was carried out. The external inspection being carried out by the Original Equipment Manufacturer (OEM) dealer BIA-KOMATSU. These inspections were included in the RFT package, to enable the bidder to make an offer based on this information. The internal inspection is subdivided into four axes per type of equipment:

- Track equipment:
  - component statistics
  - undercarriage statistics
  - KPI information (available hours, downtime, Mean Time To Repair (MTTR), Mean Time Between Stoppage (MTBS)
  - VHMS data (engine data, pump pressure, exhaust temp).

- tyre equipment: followed the same process as track equipment.

Table 3 is an extract from the internal inspection reports with all the details related to the components statistics such as the number of replacement and the hours, components hours adjustments, budget life, cost, current hours, hours to go, estimated due date, strategy of replacement, and the risk categories.

The inspection reports from the OEM were organised around two axes:

- General machine condition: pictures of main components and comments
- Machine health condition: engine speed (max and average)
    - blow by press
    - boost press
    - exhaust temperature max
    - engine oil press max and low idle
    - cool temp max
    - brake accumulator oil press max front and rear
    - economy mode
    - historical payload data.

On the valuation, three approaches have been identified. Those are:

- Writing Down Value: consist in an accounting function – five years depreciation as per EDV depreciation methodology (IFRS). Note this is a pure accounting function and not always reflective of the actual equipment value.
- BIA Valuation: consists of the life cycle costs based on asset valuation at purchase.
- Internal Valuation: Performed by EDV, based on inspection of assets and components life hours, Rebuilt etc and a fair market price value comparison by using.

Among these three approaches, the decision was to consider BIA Valuation and the EDV Internal valuation which were more representative of the equipment's current value. EDV asset's valuation was US$12 million against US$10 million from BIA. These were used to evaluate the subcontractor's offer against the valuations.

**Parts**

As indicated initially, the objective on the parts is to sell the stock and maximise on the purchase value.

The strategy behind the treatment of the parts is summarised below:

- Value is landed prices, so is inclusive of duties and clearing.
- Any purchase of Major components has been deferred. Critical parts have been zeroed for ordering and reviewed on purchases were completed by Mining and Maintenance Manager.

Furthermore, the list of parts was the current stock at time of RFT, although it was apparent the list at time of commencement would be different. The list, however, had adjustments on commencement date and the following was proposed and used to treat discrepancies:

- Total stock lower than indicated at time of RFT – the proposed volume will be adjusted to the correct value.
- Total stock higher than indicated at time of RFT – If required by the contractor then this will be sold at 100 per cent of cost value.

## TABLE 2

Data collection tracker – sample.

| No | Department | Company | From | To | Information Tile | Information details | Dependent Activity | Information format | Date requested | Date required | Date Received | Comments |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 1 | HR | Internal | Contract Owner | Regional HR Business Analyst | Labour Rates | Gang Rate or full current labour | Internal Pricing | Excel | XX | XX | XX | XX |
| 2 | Maintenance | Internal | Contract Owner | VP Mobile maintenanc e | Various Information | | Internal Pricing | Excel | XX | XX | XX | XX |
| 3 | Maintenance | Internal | Contract Owner | Roving Maintenan ce manager | Asset Value | Current Asset register based on +3000 hours | Valuation | Excel | XX | XX | XX | XX |
| 4 | Legal | Internal | Contract Owner | Corporate Legal Counsel | Sub- Contracts | Review of risks and liabilities of the current contracts associated with Contract Mining | RFT & Costing | Excel Table | XX | XX | XX | XX |
| 5 | All | Internal | Contract Owner | Stakeholde rs | Decision 1 | | RFT | Table | XX | XX | XX | XX |
| 6 | Bia - Assets | External | Contract Owner | BIA EDV consultant | Asset Inspection | An independent inspection and valuation of the Komatsu Mining Assets | Costing | PDF and Excel | XX | XX | XX | XX |
| 7 | Finance | Internal | Contract Owner | Finance manager | Asset register | Include all site for inclusion where applicable into SOW | RFT & Costing | Excel | XX | XX | XX | XX |
| 8 | Mining | Internal | Tender Engineer | Tech services manager | LOM Schedule | In Karma Format - Drilling / GC | RFT & Costing | | XX | XX | XX | XX |
| 9 | Mining | Internal | Contract Owner | Mining manager | Facilities | | | | XX | XX | XX | XX |
| 10 | Mining | Internal | Tender Engineer | HR Manager | | Staff List & Redundancy Simulation | Internal Pricing | Excel Table | XX | XX | XX | XX |
| 11 | Mining | Internal | Tender Engineer | Tech services manager | Mining Technical Data | | RFT & Internal Pricing | | XX | XX | XX | XX |
| 12 | Supply | Internal | Tender Engineer | Contract admin | Ity Contract Mining Assessment | | Internal Pricing | | XX | XX | XX | XX |

**TABLE 3**

Internal inspection extract – component statistics.

| Asset Number | Exact Model | Component | Change #1 | Change #2 | Change #3 | Actual Comp Change hours | Component Hour adjustment | Budget Life | Cost | Current Hours | Hours to go | Due Date | Strategy | Risk Categories |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| EX05 | PC1250SP-8R | ENGINE | | | | 1 | | 16000 | $ 132,686 | 10,996 | 5,005 | Aug-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | RADIATOR | | | | 1 | | 16000 | $ 6,277 | 10,996 | 5,005 | Aug-20 | Refurbish | High Risk |
| EX05 | PC1250SP-8R | FUEL INJECTOR #1 | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | SOC | Monitoring |
| EX05 | PC1250SP-8R | FUEL INJECTOR #2 | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | SOC | Monitoring |
| EX05 | PC1250SP-8R | FUEL INJECTOR #3 | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | SOC | Monitoring |
| EX05 | PC1250SP-8R | FUEL INJECTOR #4 | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | SOC | Monitoring |
| EX05 | PC1250SP-8R | FUEL INJECTOR #5 | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | SOC | Monitoring |
| EX05 | PC1250SP-8R | FUEL INJECTOR #6 | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | SOC | Monitoring |
| EX05 | PC1250SP-8R | TURBO | 8961 | | | 8961 | -500 | 8000 | $ 1 | 10,996 | 5,465 | Sep-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | WATER PUMP | 3105 | 6235 | | 6235 | | 8000 | $ 1 | 10,996 | 3,239 | Jun-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | ALTERNATOR | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | NEW | Monitoring |
| EX05 | PC1250SP-8R | FUEL PUMP | | | | 1 | 8500 | 16000 | $ 1 | 10,996 | 13,505 | Oct-21 | NEW | Monitoring |
| EX05 | PC1250SP-8R | STARTER MOTOR LH and RH | | | | 1 | 8500 | 8000 | $ 1 | 10,996 | 5,505 | Sep-20 | NEW | Monitoring |
| EX05 | PC1250SP-8R | AIR COMPRESSOR | 8961 | | | 8961 | | 8000 | $ 1 | 10,996 | 5,965 | Oct-20 | NEW | Monitoring |
| EX05 | PC1250SP-8R | AFTERCOOLER FAN MOTOR | | | | 1 | 8500 | 8000 | $ 5,746 | 10,996 | 5,505 | Sep-20 | NEW | High Risk |
| EX05 | PC1250SP-8R | PTO | | | | 1 | | 16000 | $ 48,986 | 10,996 | 5,005 | Aug-20 | REBUILD | High Risk |
| EX05 | PC1250SP-8R | PUMP MAIN P1 | | | | 1 | | 12000 | $ 19,958 | 10,996 | 1,005 | Feb-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | PUMP MAIN P2 | | | | 1 | | 12000 | $ 19,411 | 10,996 | 1,005 | Feb-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | PUMP PILOT & PTO LUBE | | | | 1 | | 13000 | $ 14,486 | 10,996 | 2,005 | Apr-20 | NEW | High Risk |
| EX05 | PC1250SP-8R | PUMP SWING | | | | 1 | | 13000 | $ 23,220 | 10,996 | 2,005 | Apr-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | ROTARY SWIVEL | | | | 1 | | 13000 | $ 15,582 | 10,996 | 2,005 | Apr-20 | NEW | High Risk |
| EX05 | PC1250SP-8R | SLEW BEARING | | | | 1 | | 24000 | $ 79,816 | 10,996 | 13,005 | Sep-21 | NEW | Monitoring |
| EX05 | PC1250SP-8R | SWING MOTOR FRONT | | | | 1 | | 16000 | $ 10,135 | 10,996 | 5,005 | Aug-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | SWING MOTOR REAR | | | | 1 | | 16000 | $ 10,135 | 10,996 | 5,005 | Aug-20 | REMAN | High Risk |
| EX05 | PC1250SP-8R | SWING MACHINERY FRONT | | | | 1 | | 16000 | $ 25,201 | 10,996 | 5,005 | Aug-20 | REBUILD | High Risk |
| EX05 | PC1250SP-8R | SWING MACHINERY REAR | | | | 1 | | 16000 | $ 25,201 | 10,996 | 5,005 | Aug-20 | REBUILD | High Risk |
| EX05 | PC1250SP-8R | CYL BOOM LH | | | | 1 | | 16000 | $ 3,567 | 10,996 | 5,005 | Aug-20 | RESEAL | Monitoring |
| EX05 | PC1250SP-8R | CYL BOOM RH | | | | 1 | | 16000 | $ 3,567 | 10,996 | 5,005 | Aug-20 | RESEAL | Monitoring |
| EX05 | PC1250SP-8R | CYL Arm | | | | 1 | | 16000 | $ 3,567 | 10,996 | 5,005 | Aug-20 | RESEAL | Monitoring |
| EX05 | PC1250SP-8R | CYL BUCKET LH | | | | 1 | | 11000 | $ 2,340 | 10,996 | 5 | Dec-19 | RESEAL | Monitoring |
| EX05 | PC1250SP-8R | CYL BUCKET RH | | | | 1 | | 11000 | $ 2,340 | 10,996 | 5 | Dec-19 | RESEAL | Monitoring |
| EX05 | PC1250SP-8R | MOTOR TRAVEL LH | | | | 1 | | 16000 | $ 16,659 | 10,996 | 5,005 | Aug-20 | REMAN | Monitoring |
| EX05 | PC1250SP-8R | MOTOR TRAVEL RH | | | | 1 | | 16000 | $ 16,659 | 10,996 | 5,005 | Aug-20 | REMAN | Monitoring |
| EX05 | PC1250SP-8R | FINAL DRIVE LH | | | | 1 | | 24000 | $ 75,141 | 10,996 | 13,005 | Sep-21 | REBUILD | Monitoring |
| EX05 | PC1250SP-8R | FINAL DRIVE RH | | | | 1 | | 24000 | $ 75,141 | 10,996 | 13,005 | Sep-21 | REBUILD | Monitoring |
| EX05 | PC1250SP-8R | BOOM | | | | 1 | | 24000 | $ - | 10,996 | 13,005 | Sep-21 | Refurbish | Monitoring |
| EX05 | PC1250SP-8R | Arm | | | | 1 | | 24000 | $ - | 10,996 | 13,005 | Sep-21 | Refurbish | Monitoring |
| EX05 | PC1250SP-8R | BUCKET | | | | 1 | | 6000 | $ - | 10,996 | -4,995 | May-19 | Refurbish | Monitoring |
| EX05 | PC1250SP-8R | SAFETY PARTS | | | | 1 | 8500 | 8000 | $ 24,636 | 10,996 | 5,505 | Sep-20 | REPLACE | High Risk |

If the contractor's proposal did not meet EDV's expectation, the intention was to transfer the parts to other sites. A full list of parts and ageing data with costs (landed) has been compiled and made available in excel format to ensure that the latest stock data would be issued at the RFT submittal date. The stock was inserted into the schedules, and the contractor was required to indicate what they want to purchase. A monthly update was done by the supply department. The initial HME (Heavy Mining Equipment) and mining stock value is shown the Table 4.

**TABLE 4**

Initial HME and mining stock.

| Items | Value (US$) |
|---|---|
| Stock on hand | 8 438 633 |
| Good in transit (update 22/02/20) | 1 986 948 |
| Open order (update 22/02/20) | 1 175 822 |

**Analysis Tax, Custom**

Tax authorities can apply unlimited and arbitrary fines on a company should they deem the rules on tax calculations have not been applied correctly. As an example, a case between Acacia Mining and Tanzania government resulted in a fine of 190$ billion for tax evasion offences. Thus, the obligation to analyse the updated tax regulations by a specialist in this matter.

Initial tax advice was sought from EDV group tax director for the proposal of contract mining. Discussions and analysis were carried out and applied. Below is the summary of discussion and confirmation:

- The sale is VAT (Value Added Tax) neutral.
- To sell market value against Written Down Value (WDV) value would attract on the profit (ie market value less WDV) 17.5 per cent in Burkina Faso.
- Customs Offices must approve all transfers (it could take up to 8 weeks).
- Duties under temp admission for transfer should be nil.
- Parts – as discussed, this would likely be a novated contract or an issue and subtract from invoices (Karma).
- Equipment's sale is sometimes considered in certain countries as a sale of business and that attracts up to 8 per cent registration tax on the value of the transfer (as opposed to the profit) – this is not the case in Burkina.

The sale process ended up in compliance with the tax system.

**Human Resources (HR)**

Human Resources was one of the critical parts of this process. Indeed, the issue at stake was social and community-based because it could have impacted relations with the surrounding community and affect the projects ability to operate. The HR team reviewed the various memoranda of understanding between the company and the workers and the collective bargaining agreement; this was to guide the process within a legal and juridical framework. But also to provide the framework for redundancy simulation.

This area is split into the different analysis, which includes the commercial value of the proposal and the practical changes (tangible) and community/legal issues (non-tangible).

The financial and numerical assessment were subjective to a percentage that the support functions provide to mining and had been consolidated in conjunction with the mining manager and the human resources manager. All direct mining employees for equivalent contracted works were deemed as 100 per cent.

There are two areas of HR that were required for this process:

1. Cost comparison of rates in the shadow bid – this is a comprehensive model that details departmental costs and was completed by Regional HR.
2. Redundancy costs was based on any legal requirement or any protocols signed by EDV – Karma and will be used to model cash flows in the potential transition if applied from owner to contractor.

With regards to employees, two scenarios were considered to steer the transition:

1. Transfer of employees to subcontractors maintaining the legal liability with EDV.
2. Redundancy of all staff.

Full cost simulation has been completed for both scenarios, with two options in the redundancy and thus three options in total.

A risk matrix assessment was developed to enable EDV to select the way of proceeding.

This was conducted by Karma HR HOD, and highlighted the potential impact on each scenario and the flexibility given by each one of them under a risk matrix format.

**TABLE 5**

Human resources – risk analysis.

| Case | Flexibility for Subcontractor | Risk analysis | |
|---|---|---|---|
| | | Employee | Karma |
| Transfer staff to subcontractor | • Free room given the subcontractor to organize the work according to it need<br>• Transferred staff to enjoy the same insurance benefit applied in the company | • People may be laid off time after working 2 or 3 months with the contractor. The fact is that they could have benefit more as social measurement karma .<br>• Contractor may do not comply with the national regulation( Karma to give guideline).<br>• Employee still want to compare his situation to the time he was working under Karma this can cause some strike on site | Karma can experience a deleterious social climate internally due to:<br>• Expansivity of Contract as the contractor has to keep all staff at the current level of payroll;<br>• Employees may require the same condition of transportation from site to Ouaga and Insurances;<br>• Employees may request the same benefits as it is at Karma for the mine closure that could maybe be difficult for the contractor.<br>• Employee will may request, despite keeping them at their current payroll, salary increase with seniority every year;<br>• frequent strike as the employees will continuously make comparison with EDV Staff |
| Redundanciy of all Staff | • Freely given way to the subcontractor to management his employment | • People may be recruited at an unacceptable level of payroll( Karma to give guideline).<br>• Certain workers may do not get employed as the subcontractor has the free room to recruit the ones it need(Karma to give guideline in the TDR). | • The negotiation with the DP's may take longer as Karma is releasing all staff.<br>• Karma may face social strike if there is not real guideline in the TDR to force the subcontractor to employ firstly the redundanced employees coming from the PAP. |

The cost from the simulation per option will be modelled into the cash flow analysis.

- Transfer staff to contractor – Option 1: **US$1.5 million**
- Staff redundancy on seniority (Direct and indirect) – Option 2: **US$1.9 million**
- Staff redundancy – 6 months (Direct and indirect) – Option 3: **US$2.6 million**

These costs were the redundancy values in terms of cash to payout. This is based on seniority varying from legal one to six month payout or full six months payout irrespective of the seniority.

On the rosters and salary grid subject, the following describes the approach:

- Preferred route– contractors to nominate their own salary grid/roster and panels to check the prices against our shadow bid and then when we see variations we can decide if we go through the pain of redundancy/strikes/costs.
- Non-preferred route – EDV nominate a roster/panel/salary grid and do not see the benefit of the contract mining.

Route 1 was considered for the tender document and comparison to existing salaries for each department and subsequent savings.

Following the decision on the preferred route, a cost simulation has been completed accordingly.

**Review of all the contract outstanding**

The first step was to identify all contracts in relation to direct mining activities. The second step was to assess risks and liabilities associated with those contracts. Thereafter elaborate a strategy with regards to the action to be conducted. A template has been elaborate for this purpose with the following items:

- contract details
- date
- counterparty
- term
- termination obligations and liabilities
- cost estimate on termination if applicable t
- recommendation.

These were all added to the decision register for recommendations.

These recommendations included three different options for treatment of the contracts. Those are: Termination of the contract, contract novation to the contractor or maintain the contract. Following the review, three contracts had been identified as actionable. Among those, two were recommended to be novate; those are tyre supply and maintenance contracts and one to be terminate (hydraulic hose supply).

**Risk**

Solarwinds MSP (nd) defines risk management as the process of identifying, monitoring, and managing potential risks in order to minimise the negative impact they may have on an organisation.

This section of the paper discusses first the risk assessment which was carried out all along from the scoping study up to the data collection.

Making such a transition involves several risks that need to be assessed before any start-up. This is to develop strategies to either transfer the risk or avoid it by finding a way to reduce its effects or accept the consequences that will eventually ensue.

As part of our approach, we have developed our risk management strategy by proceeding through various steps as listed in chronological order below:

- risk identification
- evaluation of risk significance
- evaluation of the likelihood of the occurrence of the risk
- risk prioritisation
- risk mitigation strategy
- evaluation of the post-mitigation strategy
- assessment of the post-mitigation probability.

Table 6 shows the risks identified in the process with mitigations actions. These can be regarded as subjective but nether less needed to be detailed in this process.

**Technical data collection**

This section was one of the critical phases of data collection since the technical and commercial evaluation accuracy depends on the quality of the information provided to the contractor to make their offer. Having an inaccurate set of information can lead to a misevaluation of the project's cost, thereby jeopardising the mine's future and creating a budget that's different from realised costs.

The data used to populate the schedule of rates (SOR) and included in the RFT package. Those are:

- volume to be mine during the life-of-mine (LOM) period
- design of all pits, dumps, and stockpiles
- dumping schedule to be used to design the haul profile
- site as-built face (Karma being a mine under operation since 2016)
- site layout
- updated technical report
- definitive feasibility study.

**Draft contract**

A standard AMPLA based contract amended to reduce automatic variations.

This, however, was based on a typical Schedule of Rates (SOR) and Monthly Management fees (MMF) contract for comparison to internal tender. This was based on LOM tendering and the conformity to the tender requirements were highlighted at the RFT stage. Non-conforming tenders make analyses difficult to assess and therefore, may create uncertainty to award a bid without understanding the underlying position. This also creates additional time onto the tender timeline which was already under a strict deadline

- Contractual instruments: these are the contract's formal instruments, such as the contract execution page and the schedules.
- General conditions of contract: these are EDV's standard clauses and requirements, such as insurance requirements, indemnities and responsibilities, security, retentions, force majeure etc.
- Special conditions of contract: these conditions define the general rules and regulations applicable to the individual contract.
- Scope of works: a comprehensive description of the work to be completed, including details of materials supply, completion dates, technical and commissioning related issues.
- Technical specification: describes standards and methods for the work.
- Tender schedules: information requested and supplied by tenderers that form part of the contract.
- Other Information relating to the contract.

## *Request for Tender (RFT)*

**Fuel Treatment**

A common industry practice for a mining company is to provide its own tax-exempt fuel to subcontractors (certified or not) due to logistic constraints. The fuel is bought by the mining company under its own name and exemption certificate, then stored in its fuel tank on-site to be issued to the subcontractors

On this subject, two options have been considered as appropriate for the transition with operational risk for each of them. The contractor must price the works without fuel; this will be a free issue with capped KPIs. The risk is an overestimate of fuel; then the contractor will not be penalised. Or the contractor price with fuel included; the mine owner will back charge the fuel supplied. The risk is the contractor adding a mark-up in their model for fuel, which is more likely to happen.

## TABLE 6

Risk register extract.

| | | | Risk Matrix | | | | | Risk Matrix (post Mitigation) | |
|---|---|---|---|---|---|---|---|---|---|
| No | Risk | Risk details | Value | Likelihood | Mitigation | Owner | Progress | Value | Likelihood |
| 1 | **HR Industrial action** | Allow the Contractor to nominate roster and salary grid and manage transition with EDV employees ? Will cause issues - DT operator in Plant compared to Contractor rates | High | High | GM Karma to discuss this with GP about this necessity on the survival of the project in current status and explain the protocol required for transition.<br>Country Manager will deal with national Government departments in Ouaga on advice from the GM | GM | 50% | Low | Low |
| 2 | **Main Mining Contract suitability - Risk Transfer** | Contract will not be suitable to transfer / mitigate risk in current AGO form | High | High | EDV Legal are rewriting current contract T&C | EDV Legal | 60% | Low | Low |
| 3 | **Fixed Asset Register** | Assets not accurate - complete and most High value assets have been picked up however some areas would be missed and known to be omitted from this exercise | High | High | Fixed asset register to be assessed by Maintenance and full estimation and updates to be collated before RFT issued | TB / SO | 50% | Low | Low |
| 4 | **Cost / value expectations reduction too high** | Expectations are high on savings | High | Medium | Ensure sufficient Tenderers are included. Ensure that all mining costs and associated costs are included and compared i.e Duties on parts, depreciation, | Project Owner | 0% | Low | Low |
| 5 | **Timeframe** | Timeframe tight - critical path not identified | High | High | Timeframe detailed actions completed and monitored for compliance | Project Owner | 90% | Low | Low |
| 6 | **Motivation / impact of support functions at risk** | People flight / disruption caused by uncertainty | Medium | High | LOM uncertain and GM to explain the next for this project to materialise to ensure Karma is maintained as a business | GM | 30% | Low | Low |
| 7 | **Suppliers and Subcontract** | Any cost / liability for transition | Medium | High | Review of contracts with terms & conditions / Costs and "terminate, Novate, maintain: would be identified | EDV Legal | 50% | Low | Low |
| 8 | **Assets Valuation** | Assets might not meet Value identified by current exercise (note SO used lower value which doesn't include mobilisation and commissioning costs) | Low | Medium | Valuation received in line with EDV Valuation | TB / SO | 0% | Low | Low |
| 9 | *Asset Purchase Tax Liability* | *Potential 20% tax liability to pay on FMV against WDV* | *High* | *High* | *Look at mitigating tax advantages in payments on assessment in NPV model* | *EDV Tax* | 0% | *Low* | *Low* |
| 10 | **Sites reluctant to facilitate Tender Process** | Poor / inaccurate information submitted. | Medium | Medium | All HODs to be briefed on situation. Mine Manager , GM other senior personnel supportive of this process | GM/SO | 0% | Low | Low |
| 11 | **LOM - not finalised in time** | Could cause a variation if not analysed and submitted with a high degree of certainty | Medium | High | LOM with high degree of confidence has been developed | Mine Manager | 100% | Low | Low |
| 12 | **Tax regime clarification** | Sale / Duties attraction | High | Medium | Further formal investigation of philosophy of what actual transaction would take place to ascertain likelihood and cost | *EDV Tax* | 0% | Low | Low |
| 13 | **Contractors have financial Capability to purchase / finance contracts** | Local companies have responded to pre-qualification although financial capacity uncertain. | High | Medium | Tier 1 or 2 Contractors selected. If Tier 3 is selected to continue then financial capacity needs to be confirmed prior to negotiations | Project Owner | 0% | Low | Low |
| 14 | **HR termination costs** | Cash flow pay-out of termination | Medium | High | This will be modelled in final calculations | EDV HR | 0% | Low | Low |
| 15 | **Savings - Fuel / Explosives** | Risk of free issue fuel / Explosives to reduce margin on these items can create costs if not properly managed (Engineers) | High | High | FOC with KPIs | Project Owner / Mine Manager | 0% | Low | Low |
| 16 | **Rise and Fall** | Contractors take advantage of weak knowledge and management and this results in cost creep (experienced both sides on all projects) | High | High | Look at strict R&F metrics - Contract Admin Knowledge and education | Project Owner / Mine Manager | 0% | Low | Low |
| 17 | **Operations feel that they were not involved / part of tender and stuck with potential outcome** | They become demotivated - power is out of their hands and they accept the higher varying costs of contractors | High | High | Mine Manger, GM, Maintenance Manager are all inclusive of process and informed | Project Owner | 0% | Low | Low |
| 18 | **COVID-19. Negotiation delay** | Faster negotiation face to face - potential delay on email / call and electronic correspondence | High | High | Review process and where available to face to face - bluejeans - 100 % dedication to resolve asap. Review travel restrictions | Project Owner | | Medium | High |
| 19 | **Asset Valuation** | Delay in process might cause a slight variation to asset value - purchase prices | Low | High | Allow 5% asset value reduction as risk item / value | Project Owner | | Low | low |

**TABLE 7**

Fuel option calculations.

| | |
|---|---|
| **Fuel issue Free of Charge (FOC) – option 1** | |
| Contract Price (FOC Fuel; load and haul) | $54 595 533 |
| Fuel required (litres) | 25 348 990 L |
| Fuel Price | $1.16 |
| Total Fuel Price | $29 404 827 |
| Total Contract Price | $83 000 360 |
| **Back charge – option 2** | |
| Contract Price (FOC Fuel; load and haul) | $54 595 533 |
| Fuel required (litres) | 25 348 990 L |
| Fuel Price | $1.16 |
| Total Fuel Price | $29 404 827 |
| Markup on fuel/Contract rates | 18% |
| New Fuel Price in contract | $34 697 697 |
| **Total Contract Price** | **$88 293 229** |

The total additional contract value would be US$5.2 million (approximately US$2.3 million per annum) which is a reallocation of profit to the contractor from EDVs cash flow. This is equivalent to a 4.5 per cent increase in the mining costs with a back charged fuel methodology applied. Therefore, a decision is imperative on the fuel treatment on any Contract and how it is applied, and all risks should be looked at prior to the decision made.

The preferred route was the option of free issuing the fuel, with the mitigating measures being the assessment of fuel requirement in EDVs internal model, which will be a basis of negotiation in case of overestimation from the contractor.

### Scope

As define by Fagan (2019), an RFT, or request for tender, is an open invitation to suppliers, asking them to send offers – usually as sealed bids – in a structured format. This section describes the work around the finalisation of the RFT and the invitation.

The preliminary scope issued during the EOI must be refined to define a working framework to be completed by the contractor. Hence a meeting with all the main stakeholders is held and a decision register is elaborate and send couple of days prior the meeting with decision points to be discussed during the meeting. Those are: Mine General Manager, Finance Manager, Mining Manager, Vice President Supply, Vice President Mining, and Chief Operating Officer.

The decision register was made up of the following information:

- decision points
- details/description
- risk/opportunity
- stakeholders comments
- final agreement per decision points.

The agreed final scope of works was a full mining, maintenance and ROM feed contract based on a reliable LOM schedule. Variations being a risk to the schedule, this will be minimised by the following measures:

- Ensuring that variations are activated on a 15 per cent volume change of production schedule.
- The variations will be based on a 6-month rolling schedule.
- EDV will maintain option of separable portions for short-term mining works on the project should the expense of mobilising equipment prove inefficient. These would be contracted to a local contractor or self-performed on hire equipment by EDV.

**Schedule Of Rates (SOR)**

The aim is to develop a framework whereby subcontractors can compete at the same level while minimising the disparity between offers. The absence of a haul profile gives subcontractors the latitude to create their haul profiles, which creates a subjective baseline for tenders' evaluation. From this standpoint, elaborating a SOR based on haulage profile per material type turned out to be suitable. It includes moisture contents, a source to destination profile, with a provision for overhaul and under haul. ROM (Run-of-mine) feed will be based on a set of distances and material types to give flexibility in feeding the crusher

It also gives the opportunity to assess the contractor ability to conduct the work by demanding a high level of detailed information, as presented in Table 8, detailing the various section of the SOR.

**RFT Package and Tender Period**

According to Stanley and Mikhaylova (2011), The RFP should specify the format for bid submissions, such as separation of technical and financial proposals, predesigned forms to be filled out, a checklist of supporting documents, and so on.

To provide clarity regarding the documentation provided for the tender, the following was highlighted in respect of the tender document:

- Request for Tender (RFT) – Part 1 – Tender information and conditions (inclusive of the Draft Master Service Agreement as an Attachment)
- Request for Tender (RFT) – Part 2 – Questionnaire about the contractor capability
- Annexure 1 Register of Supplier Concerns – Concerns/questions relating to the content of the RFT documents
- Annexure 3 Non-Disclosure Agreement
- Tender Acknowledgement
- Contract mining schedules of rates
- Link to Full Issued Tender Package on EDV Cloud Application
- A set of guidance notes were included in the RFT to help the contractor understand the philosophy behind the Tender and individual schedules to be completed. This was further explained during the site visits.
- A bidder may have some valid concerns and may want to raise it during the submission period. A RFI (Request for Information) had been developed for this purpose. The answer is sent and shared with all the contractor on an anonymous basis.
- After issuing the RFT package, a site visit is organised as per the timeline, during the submission period. The coordination was carried out in conjunction with the site team and the tender team. Furthermore, during the site visit, the tender team made a presentation on the SOR template to facilitate the submission, thus avoiding a considerable correspondence of email clarifications, and giving a clear submission from the bidders.

**TABLE 8**

Schedule of rates content.

| TABLE OF CONTENTS | | COMMENTS |
|---|---|---|
| Evaluation | EVALUATION | This is for reference and is the method used in scoring the Submissions |
| SCHEDULE A | RESPONSIBILITY MATRIX | This details the key items responsible for each party for pricing |
| SCHEDULE B | ESTIMATED CONTRACT PRICE | Linked for total Contract value |
| SCHEDULE C | QUALIFICATIONS | Please complete all areas with Contractors Qualifications |
| SCHEDULE D | TENDER TIMELINE | Please review and highlight any concerns regarding the tender timeline |
| SCHEDULE E | CONTRACT Terms & Conditions (T&C) | Please highlight clause and concerns regarding draft Document |
| SCHEDULE F | ASSET PURCHASE PRICE | Contractor is to insert their proposed purchase price of the assets |
| SCHEDULE G | PARTS LIST FOR PURCHASE | Contractor is to insert their proposed purchase available parts. Parts volume will change over the period of the tender and negotiations. ITY reserves the right to not sell any parts as part of this tender |
| SCHEDULE H | *TOOLING FOR PURCHASE* | |
| SCHEDULE I | *SAFETY STATISTICS* | Contractor is to complete the table provided |
| SCHEDULE 1 | *DESCRIPTION OF MINING SERVICES* | No submittal required - SOW describing the works |
| SCHEDULE 2 | *SITE DESCRIPTION AND INFORMATION* | No submittal required - SOW describing the works |
| SCHEDULE 3 | *PRECONTRACT INFORMATION* | No Submittal - Information that either party deems necessary to insert as part of the submittal or subsequent negotiations |
| SCHEDULE 4 | *STATUTORY DECLARATION* | No Submittal - Details submittal requirements by the Contractor for payment of Monthly fee |

| | | |
|---|---|---|
| SCHEDULE 5 | *REQUIRED AUTHORISATIONS* | No Submittal - Details of Authorisations |
| SCHEDULE 6 | FACILITIES PROVIDED BY PRINCIPAL | All Assets identified - all assets within the listed assets is deemed to be included |
| SCHEDULE 7 | CONTRACTOR'S MAJOR PLANT LIST | List of all existing and planned equipment. Contractor to list productivities as per table G2 |
| SCHEDULE 7.1 | PLANT AVAILABILITY & EQUIPMENT HOURS | Indicate the availability that is used to calculate equipment numbers and SOR |
| SCHEDULE 8 | *INSURANCES* | No Submittal - Details of the during the term of the Contract |
| SCHEDULE 9 | KEY PERSONNEL | List all key personnel that will be mobilised to the operation |
| SCHEDULE 9.1 | CONTRACT PERSONNEL | Full numbers of people mobilised to the operation in total - indicate rosters also for each category |
| SCHEDULE 9.2 | SUB-CONTRACTORS | A list of all Subcontractors and work type should be listed whether novated, existing, new , planed. |
| SCHEDULE 10 | DIESEL CONSUMPTION | This schedule details the planned diesel and Lubricant consumption forecasted per month |
| SCHEDULE 11 | CONTRACT PROGRAMME | A summary of the schedule per month in ore / waste |
| SCHEDULE 12 | HAULAGE PROFILES | Haul profiles from mined benches (sources) to dump bench (destination) |
| SCHEDULE 12.1 | HAULAGE PROFILE DETAILS | Haul profiles from mined benches (sources) to dump bench (destination) details |
| SCHEDULE 13 | *OPERATING REPORTING REQUIREMENTS* | No Submittal - Details of the Reports required by the Principal from the Contractor for the Operation |
| SCHEDULE 14 | MONTHLY MANAGEMENT FEE | This part is to list major individual personnel and equipment for the planned duration of the project |
| SCHEDULE 15 | MOBILISATION AND DEMOBILISATION COSTS | List cost to mobilise additional equipment if required over and above existing. Total equipment schedule for each month should be completed |
| SCHEDULE 15.1 | ESTABLISHMENT | list all purchases and additional establishment items |

| | | |
|---|---|---|
| SCHEDULE 15.2 | VARIABLE RATES - LOAD - Waste to waste dump | Schedule in Months |
| SCHEDULE 15.3 | VARIABLE RATES - LOAD - Ore ROMPAD | Schedule in Months |
| SCHEDULE 15.4 | VARIABLE RATES - CRUSHER FEED | List wet tonnage rates for rehandle from ROM / OFF Rom rehandle. Note Principal might opt for direct tip and so quantities / distance may vary |
| SCHEDULE 15.5 | HAULAGE FROM PIT EXIT - ADJUSTMENT RATES | This is for any additional overhaul from existing Haul Profiles |
| SCHEDULE 15.6 | DAYWORKS AND ADDITIONAL HIRE | Rate only and should include all items to carry out activity |
| SCHEDULE 15.7 | INVENTORY HOLDING | A list of planned inventories held by the Contractor to mitigate risk to the Project |
| SCHEDULE 16 | *DAMAGES FOR ORE LOSS AND DILUTION* | No Submittal - Details of the Calculation of loss or excess dilution of the Ore by the Contractor |
| SCHEDULE 17 | RATES AND PRICES - RISE AND FALL FACTORS | These should reflect workload in terms of % of loading, Hauling and other works (Drilling).They must include realistic fixed component |
| SCHEDULE 17.1 | RISE AND FALL LABOUR INDEX | The Contractor should fill in as much detail for allowances and salary grids to allow a proper assessment on R&F application in the Contract |
| SCHEDULE 18 | EARLY TERMINATION AMOUNT | Contractor to indicate the Termination amount for each month of the Contract Term |
| SCHEDULE 19 | *PARENT COMPANY GUARANTEE* | No Submittal - Details of the Submittal of PCG prior to commencement of the Contract |
| SCHEDULE 20 | *PRINCIPALS TAX EXEMPTIONS* | No Submittal - Details of the Tax exonerations that the Contract can apply for as part of the Contract |
| SCHEDULE 21 | *ISSUES TO BE DETERMINED BY EXPERT* | No Submittal - Details of determination by an Expert should the Contract require |
| SCHEDULE 22 | *MANAGEMENT PLANS* | No Submittal - Details of Management Plans to be submitted by the Contractor prior to Commencement of the Contract |
| SCHEDULE 23 | CSR COMMITMENTS | The Contractor should fill in as much detail for allowances and should complete the financial commitments (in USD) to community projects |

## Comprehensive evaluation

The EOI has been issued to eleven contractors. From the six who replied to the PQQ, two contractors pulled-out from the tender process following the site visit. Among the four remaining contenders, two formed a Joint Venture (JV) to submit their bid. Three bids were therefore received for evaluation.

Our evaluation process had three stages with the objectives to select a suitable contractor that understands the requirements of the project with the most attractive economic offer.

The first stage was to analyse the schedule submittals from the first tender returns to allow: Initial assessment and notification. This included a one-hour call to each contractor. The second stage is the evaluation of the second tender returns. After the clarifications of the stage 1, an extra one week is granted to the contractor to refine their submission pursuant to our feedback. The third stage is the negotiation phase.

The tender team conducted the tender evaluation in collaboration with support members that provide guidance and inputs for the final review. Those are general manager, mining manager, HR manager, Legal counsel, VP maintenance. From this evaluation, a recommendation is made with supportive data to the tender committee. – The tender committee is the organ formed by different levels of stakeholders in charge to designate the contractor based on the tender team's information.

The evaluation is carrying out against several factors. Those are regrouped into the following sections:

- Health, safety, environment, quality and sustainable development
- Technical and supplier capability
- Commercial
- Risk.

A bid evaluation matrix was constructed to evaluate with scores on a scale of 1 to 3, with 3 being the worst mark. Each item had a weight; the scores were then multiplied by the weight to have the final score, which will guide the tender committee's recommendation.

This matrix aims to have an objective assessment with a set of criteria deems essential for the successful execution of the project. These evaluation criteria have been weighted in accordance with the expectation of EDV, in terms of the value the contractor will provide to this specific project.

The review considered

1. Current mining costs + proposed indirect costs on current operations
2. Difference between operational activities and proposed Contract mining
3. Total LOM costs – owner/Contractor
4. Risks/opportunities associated with Contract mining
5. Considered weighted averages of activities and major costs
6. Be independently reviewed
7. Recommendation.

The assessment will include any Non-direct staff and G&A expenses that would result from a change in operating model within the group and these will be isolated and identified.

During the approval process financial benefit indicators were used to vote the project. These indicators will be used to ensure the project and business objectives are met.

**TABLE 9**

Benefit indicator.

| Indicator | Project value |
|---|---|
| Total Costs | Total cost of LOM and 5-year contract length |
| Cost per tonne | Isolated cost per tonne including MMF |
| NPV | Based on cash flow model |
| Asset Purchase Valuation | Based on tender submittal |

### *Internal pricing and evaluation*

An internal pricing tender-based was carried out along during the tender period, to serve as a basis of assessment from the different bid offers. This was the threshold that enabled the contractor's financial proposals to be benchmarked and highlight the cost-effectiveness of the transition. It also assessed the operational risk – the contractor's ability to perform by looking at elements such as the type and the number of equipment, fuel burn rate, availability, productivity and staff numbers.

**Financial assessment – Choice of Methodology: Depreciation and Finance**

One of the risks which has been identified and should be separated, is the understanding of contractors cost against an owner's cost. This understanding is imperative to show that without it, decisions would be based on information that cannot be objectively compared.

- **Depreciation** – Although this will not be a cash cost to the site as an owner, this is still an accounting cost (below mining costs) on the Profit and Loss (P&L). This cost applies to a contractors rates, whereas it will not appear in a Mining cost report. If this is not compared, then this will lead to contract mining never being considered as owner mining will always look more commercially advantageous.
    - **Depreciation comparison** – The second understanding of this is the way depreciation is applied. If we use IFRS depreciation of a straight line 5-year cost, this will create an artificial comparison. A contractor cost for equipment is based on an hourly unit for depreciation purposes and applied in their rates. Therefore, to get a comparison, then the same methodology should apply. Failure to do this would inadvertently make a contractor look cheaper for the first five years and again create a wrong decision if applied using this method
- **Finance Costs** – Similar to straight-line depreciation cost, these financial costs would make Contract Mining more attractive in the equipment's finance period. It's a cash cost, but an accelerated one compared to the equipment's life causing the finance period to look more expensive for owner mining and then reverting to a cheaper owner cost, post finance term.

We used a comparable unit depreciation method to look at unit rates or the wrong decision could be made on the timing as explained above. This is how a Definitive Feasibility Studies (DFS) comparison is calculated or any owner/contracting comparison

Furthermore, EDV accounting allows for certain functions/costs to be allocated in other departments/regions. With a transition to Contract Mining these costs would reduce or zero in the relevant areas. This would be a function of contract mining; however the direct costs of mining would not benefit. These include and not limited to, the following:

1. Personnel reduction in other departments serving Mining
2. Finance and interest on spare parts
3. Insurance of equipment (site and Corporate)
4. Transport and Camp Costs
5. Duties and clearing on parts (these are now allocated to mining parts)
6. Depreciation (as discussed above).

However, these costs would be allocated directly to a contractor cost in their schedule of rates would not be seen instantly as a difference to current mining costs.

The following approach has been considered for our evaluation process:

- Maintain these costs as an overall comparison from Owner to Contract Mining and highlight them out as non-direct.
- Use hourly depreciation as the non-direct comparison.

### Evaluation

A risk profile of the three contenders was carried out along with the financial and capability assessment. As a result, the high-level summary emerged:

- **Contractor 1**: Australian African-based contractor with drilling operations in the West Africa region. Trying to break into the comprehensive mining services market with a low-risk profile.
- **Contractor 2**: a JV between a Portuguese contractor operating in the region and a Malian civil contractor. The risk being the JV formed during the tender process can be fractious.
- **Contractor 3**: west African based contractor with operations in EDV sites; carried out SOR blasthole drilling at Karma.

The SOR made it possible to carry out the commercial evaluation through the various rates provided, the technical evaluation through the contractor's ability to carry out the work, and the risk evaluation based on the direct operational activities.

For example, in schedule 17.1, the contractor is instructed to provide the rise and fall labour index. The index was used to analyse the notional gang rate. It is defined as the hourly cost per working hours of the mining junior staff (included in the equipment rate).

As meant in the HR section, the risk of having an offer from a subcontractor with a wage scale substantially lower than the current one was classified as high due to potential industrial action; For instance, the case of Goldfields in Tarkwa, Ghana 2018, whose transition led to a strike. Since Karma is a site with an annual production of approximately 100 000 ounces of gold, one day of stoppage would correspond to a loss of 274 ounces. Considering the realised gold price of $1600, a one-week shutdown would have resulted in a loss of 1918 ounces, representing $3 million.

First principles budgeting using industry metrics, and actual data from the site, were used to carry out the internal tender. The followings assumptions were used for the shadow bid:

- Proposed Collective Bargaining Agreement rates used for the internal tender
- Site Budget was used for non-scope activities costing and updated where required based on LOM tender
- Scope activities costing are tender-based
- Major components will follow the OEM recommended change-outs for the internal tender
- Depreciation for the comparative tender method will follow standard life hour methodology.

The resources needed to carry out the work in terms of quantities (Equipment numbers, manning, operating hours) and consumables (Fuel, components, parts) were determined from the internal pricing. Subsequently, those costs were added to the non-scope mining costs to generate the overall mining costs. The contracted overall mining cost was calculated as below:

*Pricing from the bidders*

+

*Fuel (Fuel Capping included)*

+

*Assets, parts and tooling sales*

+

*non-scope mining costs*

-

*no more needed non-direct cost*
*from owner mining model*

=

Contracted overall mining cost

After that, those costs (Owner and contractor) were modelled and compared to each other.

The cash flow forecast considered all factors, including:

- Asset sale and loan repayment
- Personnel redundancy
- Contract termination payments
- Contract tender pricing
- Additional staffing costs (ie Contracts Manager, Admin)
- Parts sale
- Reduction in non-directs ie Software, insurance which are paid centrally
- Working capital finance costs.

The table below summarises the cash flow model from the various options.

Important to note that the discount rate used is the weighted average cost of capital of the company.

**Negotiations**

A notice was sent to the contractor three to formally notify him that his proposal is considered competitive enough to enter into contract negotiations; this was not a notice of award, same dependent upon the parties reaching an amicable agreement on terms and conditions by no later than a preferential date selected by EDV. The objective is to:

- To understand the proposal and discuss schedules
- To go through safety and expectations
- To go through Legal T&C
- Contractor to present their case for selection
- To agree on commercial terms (negotiate an additional 5–10 per cent discount)
- Issue a Letter of Intent.

# TABLE 10

Cash flow comparison summary.

| Run 2 - April 14th 2020 | | | $/tonne | | | Implied cost | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| MINING COST | Initial inflow/Outflow | Initial inflow/Outflow | Y1 | Y2 | Y3 | Y1 | Y2 | Y3 | NPV * | TOTAL |
| | | | | | | | | | 7% | |
| EDV MINING COST | $ - | | $ 2.95 | $ 2.37 | $ 2.56 | 50,371 | 45,930 | 18,213 | 106,759 | $ 114,514 |
| EDV MINING COST (With Non direct) | $ - | | $ 3.13 | $ 2.56 | $ 2.81 | 53,539 | 49,627 | 20,013 | 114,776 | $ 123,180 |
| | Upfront Payment | EDV Staff Redundancy | | | | | | | | |
| Mining Cost - Contractor 1 | 11,955 | -2,942 | $ 2.97 | $ 3.00 | $ 3.75 | 50,701 | 58,148 | 26,740 | 116,581 | $ 138,530 |
| Mining Cost - Contractor 2 | 0 | -2,942 | $ 2.50 | $ 2.38 | $ 2.92 | 42,700 | 46,224 | 20,834 | 104,750 | $ 112,701 |
| Mining Cost - Contractor 3 | 0 | -2,942 | $ 2.07 | $ 2.58 | $ 3.17 | 35,354 | 50,106 | 22,578 | 102,555 | $ 110,981 |

NB: Following the financial assessment, contractor two and three offers were deemed economically viable. In the view of the risk profile, contractor three was identified as the preferred contractor to engage in a discussion, with contractor two being held in abeyance and contractor one rejected by notice.

The initial plan was for the negotiations to be held in London, at the Head Office for the following reasons:

1. Professional setting with no distractions
2. EDV Legal are present to assist
3. Seniors EDV management executive are available to meet if required.

However, due to the global pandemic, this option was abrogated. Negotiations were held as indicated in the risk management matrix by video call with the different stakeholders.

The contractor and EDV reached an agreement with a 5 per cent discount on the overall contract price.

The contractor's assets purchase proposal was US$10.1 million with a monthly instalment over three months. With regards to parts, a US$4.83 million offer, which represents 69 per cent of the value (US$7 million) at the time of the transition, over an 18-month period.

## PART II – POST-EVALUATION

### Implementation

An implementation aims to execute a precise action plan addressing several points. Following the adjudication, a phased deployment schedule specifying the main decision points throughout the deployment process was defined and indicates deployment activities. Integration took place without creating abrupt changes that could interfere with the smooth running of activities. It was done in such a way that everyone adheres to the project. Hence the importance of identifying, beforehand, the tangible (human resources, assets, document) and non-tangible (procedures, relationship with community) elements directly or indirectly related to mining, for better planning and optimum deployment of activities.

A series of meetings is organised with each HOD with a pre-developed action tracker to run the meeting. The different departments are:

- Mining
- Mobile maintenance
- HR
- Supply
- CSR (Community Social Relationship)
- Health, Safety and Environment
- Security.

The action tracker list the items, update from the principal and contractor, meeting objective, agreement made on the action made during this meeting, action, responsible of the action, action date closure, status (open, in progress, closed, overdue) and comments.

### Lesson learned

This transition with the contribution of all the parties was successful with a smooth integration and values delivered above our expectation. This was the first from our company and led the path to future transitions from owner to contract mining works. Yet during the implementation phase some unidentified issues arose throughout the process but were resolved with appropriate actions.

In the interest of continuous improvement, an internal and external survey on lessons learned on how the process could be improved was conducted. The overall feedback was positive, and some valid points raised by contractors were recorded and will be used for future tenders.

The table below is summarising the lessons learned from Karma, with mitigation measures for the upcoming transition project.

The benefit of the transition proved to be financial, in the sense that the mining cost resulting was 4 per cent lower than forecasted (US$2.32/t versus US$2.42/t), which is translated into a savings of US$724 000 for a yearly production of 7.2 million tonnes, and avoided the liabilities accruing the major components change-outs normally associate with owner mining.

The cost profile is expected to maintain against budget with some key points of the transition below:

- No further capitalisation of major components
- No accruals of labour costs (mining operators)
- Cash flow improved through no HME parts holding.

At the community and social relationships level, the transition was a success since no strikes were recorded. This was mainly achieved through an upstream risk assessment and strategy and effective coordination of the HR and CSR team, which maintained clear communication between the different parties throughout the process. All this during a period of a global pandemic. The transition came into effect on 2 June 2020.

## CONCLUSIONS

In a context where Karma mine's operating performance combined with the value of the deposit did not meet EDV group's criteria. Challenging the operating model was the obvious step in the continuous improvement process. Therefore, the objective was to assess our current operating model against those proposed by the contractor and then subsequently appraise the transition viability through a comprehensive analysis of all components affecting by this activity. This was done through a scoping study to determine the framework within which the transition could be effective, followed by a data collection and analysis required to build a structured process.

The evaluation ended up being economically viable with a cash release of US$10.1 million from the asset sale on the first three months starting on the contract commencement date with 30 per cent as upfront payment which enables the financing of the staff redundancy. From the risk standpoint, contractor three was identified as low risk.

Following the award, the transition occurred with only a 24-hour disruption to the operations on the contract's agreed commencement date. The pre works made this a seamless transition with operations maintaining production to the pre changeover levels.

The transition from the owner mining model to the contract mining model seems to be a complicated process considering the various impacted cores and non-cores components. Nonetheless, this is possible by accurately identifying those factors, the risks associated with this process, and a suitable strategic implementation plan.

## ACKNOWLEDGEMENTS

I would like to thank my superior Spencer Osborne, the Vice-President Mining, who gave me the opportunity to be part of this journey.

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## TABLE 11

Lessons learned from tender.

| No | Item | Issue details | Karma Mitigation | Ity Mitigation | Comments |
|---|---|---|---|---|---|
| 1 | Asset List | Additional Assets added during tender - negotiations | Last minute rush to get agreement - 3 x Sino trucks outstanding | Refresh Asset register monthly | Ensure other items like simulators , filter washers, other are included |
| 2 | LV , Ancillary & Processing Equipment | Servicing and repair of the equipment staying with Karma | This was not in original SOW but was added during negotiations on a cost per item to service base or cost plus to fix | Ensure all items are picked up - use same format of pricing | Check site on any other obligations |
| 3 | People | Planned to keep some people in Mining who were eventually transferred to Contractor but in hindsight required | Reallocating duties | Ensure sufficient senior field guys are retained | ROM pad supervision and keeping control of Contractor s |
| 4 | Negotiations | These were held on Skype and restricted to internet access | This resulted in minor confusion on some small costs. Resolved prior to signing | To review on isolation restrictions | Look for suitable location where limited restrictions to close out quicker |
| 5 | Doc Control | Emails still being used - lost some doc control on key documents | had Egnyte set up | Current setup is Teams (working documents) , Egnyte (final - complete documents) | Ensure people use and reject emailed documents. Nadege is supervising doc control |
| 6 | Contracts | Termination in time | Agreed to terminate and transfer costs to Contractor | Get these on timeline | Difficult as terminations are 90 days so hard to terminate unless sure contract is formed - engage key suppliers earlier |
| 7 | Day works | Day work rates for other items were not requested at time of tender | To close out - WA 500 rate was negotiated in time for Contract signing | Ensure Tender team are aware of any planned equipment movements | LP, Pumps, |
| 8 | Stores | Obsolete stores - allocations | The list is outdated, and adjustments will be made | Re issue list on LOI | |
| 9 | Tools sales | Initial tender team never created list for tender documents although aware were an asset | This was negotiated satisfactorily in time | Get full list and issue as part of tender | |
| 10 | Customs Inspections | These took 6 weeks to complete - late to engage | Engaged consultant to expedite inspections - will not affect progress of change over | Get engaged as soon as LOI issued | |
| 11 | SOW | Rehandle road not included | Using dayworks | Either include in SOW or get SOR to maintain | |
| 12 | IT | Allocation of software and computers on changeover | Some will be use by Contractor temporarily | Get list of IT available - might not need it with restrictions | This is a function of Covid restrictions also |
| 13 | Importation Documents | There is no central repository of import documents of EDV assets | Finance or Supply Chain teams might have them. | Obtain from HMS/Arion archives all importation documents ASAP | Create a central folder to archive all importation/contract documentation. Legal have some, Corporate have other… |

| | | | | | |
|---|---|---|---|---|---|
| 14 | Projects | Inclusion of ancillary works in project costing and mining model comparison | Dayworks | Ensure included - Laisse with Projects | |
| 15 | Ancillary | Fuelling of ancillary unit like genset with no fuel bowsers. | Dayworks | Get list of scope for all non-sell items | |
| 16 | Fuel | Fuel tag management (task carried out by HME under owner mining model) | Reallocate duties | Ensure allocations are swapped prior to changeover | |
| 17 | Fuelling Generators | There are some small generators that need fuelling daily which were done by mining - as we sell equipment, we need these to continue to be carried out | We will agree a set number of hours a week | List and put in schedule how many "other" equipment require servicing by Contractor | |
| | | | | | |
| **A Letter was sent to 3 x Contractors who submitted, and the following is feedback which may be incorporated into the ITY tender** | | | | | |
| | Contractor 1 | Upfront payment required - this made contract hard to price up and accept with low lOM | This was the criteria | None - will be listed in ITY EOI for $50 million as prequalification | This reflected in CD price and qualifications |
| | Contractor 1 | As a general comment, the site operating life remaining, significant cash up-front spend to take over plant, equipment and inventory, the site location and the global Covid-19 pandemic all combined were factors that may not have brought out the best result for all parties. | This is unfortunate but we managed to form a contract at a good rate | None | |
| | Contractor 3 | Some vehicles were added/removed from th | Agree there were some changes | Ensure SMI understand that a sale agreement means transfer of equipment and to be sure that the list should maintain | |

# A tailored solution to determine plant capacity using strategic analysis

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## ABSTRACT

There is no single factor that determines the appropriate capacity for a mining project. It is driven by the size of the resource, the capacity of the market, the availability of capital, the availability of power, the availability of water, and a myriad of other considerations. There is also no single solution to the question of how big the project should be. However, to develop a project, the study team needs to select the configuration options that it regards as the best for the project. To be able to make this selection, the study team should consider the range of available options and do so at a level of detail that gives it confidence in the conclusion but does not lead to multiple detailed design exercises. Having considered the range of options, and with a clear definition of the project owner's objectives and constraints, the work to be carried into feasibility studies can be focused appropriately.

This paper discusses the process of completing strategic mine planning to identify project configuration options to assist in the selection of the processing plant throughput rate (Capacity Study) for a large greenfield open pit mine. The processes described in this paper allow for the basis of design to be set for subsequent studies that meets the objectives at a risk profile acceptable to the owner.

The relationship between the deposit, processing plant, mine, market and other constraints can be assessed holistically to rank options based on a range of metrics, rather than considering the processing plant as a standalone entity, thus enabling the process flow sheet and plant capacity decisions to be tailored to the project and corporate objectives.

As part of the Capacity Study, multiple strategic life-of-mine schedules were developed that allowed the effect of varying multiple parameters to be quantified and incorporated into the plant capacity decision. The key parameters assessed included: mining bench height and selective mining unit (SMU) size, associated mining dilution and ore loss, variable cut-off grades, mining equipment, pit staging and final pit stage selection, stockpiling strategy, vertical rates of advance in the pit, preproduction mining duration, and mining and processing rates and ramp-ups. Other choices that are potentially important to the project can be assessed. The Capacity Study also considers both the inclusion and exclusion of Inferred Mineral Resources to understand the risk associated with resource uncertainty and its impact on the plant capacity decision. Other uncertainties can also be considered to understand the impact on the choices, such as metal prices, foreign exchange rates, and metallurgical recovery.

Identifying and selecting the plant capacity is a key decision to be made for all projects. Not only does it have a material effect on the project's upfront capital cost, but also a long-felt effect on future cash flows. It can change the way in which a mine is operated, and ultimately the overall project value. Understanding the physical and financial risk associated with plant capacity is essential for the informed stakeholder.

Adopting the described approach to determine plant capacity ensures key decisions were given adequate consideration and were underpinned by a systematic approach with defendable outputs, rather than it being a decision based on inadequate analysis.

## INTRODUCTION

Conducting strategic mine planning early in a project's study phase provides significant opportunity to improve a project's outcomes and allows key basis of design (BOD) decisions to be set for subsequent studies whilst reducing the risk that some options are prematurely rejected or overlooked entirely. Clear study objectives, combined with specialist software skills and experience, are essential in achieving the desired project outcomes.

This paper summarises the approach and outcomes of a study (the Capacity Study) undertaken using strategic mine planning to analyse project configuration options to assist in the selection of the processing plant throughput rate for a large greenfield open pit mine.

For confidentiality reasons, the project is not named in this paper, with most metrics and outcomes shown in relative terms accordingly.

The Capacity Study was commissioned by a mining company and undertaken by a mining consultancy as part of a prefeasibility study (PFS) being undertaken by the lead engineering consultancy. Independent reviews of all studies were undertaken by a third party. The recommended plant throughput rate identified in the Capacity Study was accepted by the mining company, allowing the BOD to be set for the processing plant, and for the PFS to proceed at the identified project scale.

The relationship between the deposit, processing plant and mining equipment can be investigated by linking their properties to particular plant throughput rates. Using pit optimisation tools and scheduling of the resulting pit inventories allows comparison and ranking of mining and processing schedules and the likely economic outcomes for a range of plant throughput rates, allowing the full potential of the deposit to be unlocked.

The deposit is extensive, with a strike length of approximately 3 km, and dips at approximately 30°. Exploration drilling has not yet closed off the resource at depth. The mineralised zone is typically several hundred metres wide on a typical mining bench.

The proposed processing method involves crushing, flotation and metal recovery. The Capacity Study approach is equally applicable to operations with one or more products.

The initial capital costs for the various project configurations estimated for the Capacity Study were in the billions of dollars.

The general approach adopted for the Capacity Study was:

- Identify the objectives of the Capacity Study, whilst also considering the required accuracy, inputs and time frames.
- Develop a mine planning approach to achieve the study objectives.
- Develop and collate inputs, with agreement from stakeholders.
- Undertake strategic mine planning.
- Analyse results.
- Peer review.
- Reporting and communication of results to stakeholders.
- Authorisation to proceed from the client. Formal decision recorded on a decision register.

# OBJECTIVES

## Primary objective

The primary objective of the Capacity Study was to identify the preferred plant throughput rate for the PFS, at a minimum scoping study level of accuracy, based on the inputs available at the time.

A wide range of plant throughput rate scenarios were considered, supported by multiple life-of-mine strategic schedules, that allowed the effect of different decisions relating to key parameters to be quantified and incorporated into the plant capacity decision.

The key metrics used to rank the scenarios, to identify the preferred scenario, were:

- Net present value (NPV)
- Internal rate of return (IRR)
- Total initial capital expenditure
- Ratio of NPV to initial capital expenditure (a capital efficiency ratio)

- Maximum cumulative negative undiscounted cash flow
- Payback period.

### Secondary objectives

Secondary objectives of the Capacity Study were to understand risks associated with the decision on plant capacity, in the context of various uncertainties at the outset of the PFS. For example:

- Geological uncertainty. If mineralised material classified as Inferred Mineral Resources under the JORC Code (2012) does not upgrade to Measured or Indicated Mineral Resources, would that affect the plant capacity decision?
- Pit stage selection. How would the corporate decision of whether to maximise project NPV or maximise ore reserves affect the decision on the plant capacity? And how would NPV reduce to achieve a higher ore reserve?
- Mining bench height. Would the preliminary PFS assumptions used in this Capacity Study relating to mining bench height and mining fleet affect the decision on plant capacity, thus precluding changes to mining fleet assumptions later in the PFS, once the findings of PFS mining equipment trade-off studies became available?
- Metallurgical recoveries. How does applying a higher threshold cut-off grade in studies (compared to what will likely be adopted in operations), to manage risk associated with limited metallurgical test work for lower grade material, affect the decision of plant capacity?

## MINE PLANNING APPROACH AND SOFTWARE

The general approach adopted in mine planning was:

- Develop mining block models to include allowances for mining dilution and ore loss, with a range of bench heights and selective mining unit (SMU) sizes considered.
- Match SMU size to the loading units.
- Match the truck size to the loading unit size.
- Match the loading and trucking unit rate to the plant throughput and mining rates.
- Develop a mining inventory for each processing throughput rate option.
- Pit stage selection.
- Strategic scheduling.
- Analyse the results.

The various specialist mine planning software identified as the most appropriate tool, and subsequently used in the Capacity Study, to undertake aspects of mine planning were:

- Dilution and ore loss modelling undertaken using Studio OP™, version 2.7.33.0 (by Datamine) mine planning software.
- Development of interim and final pit limits based on pit shells undertaken using Whittle™, version 4.7.1 (by GEOVIA, Dassault Systems) pit optimisation software. High level analysis was also completed using the Milawa scheduling module of Whittle.
- Life-of-mine strategic scheduling undertaken using Scheduler™, version 7.0 (Minemax). Control over scheduling constraints and stockpiling strategies was achievable using Minemax, with an optimised result, in terms of NPV, produced for the given inputs and constraints.

## INPUTS

The input parameters applied in the Capacity Study were a combination of preliminary inputs developed for the PFS, from across multiple disciplines, nominally at scoping study level of accuracy, to allow various trade-off studies to be completed, and other non-cost related inputs which were sourced from a prior scoping study, such as overall pit slope angles. Inputs were developed and supplied by several stakeholders including the mining company, lead engineering consultancy and the mining consultancy. The types of inputs collected for the Capacity Study are summarised in Table 1. The majority of these input parameters were ultimately updated as part of the PFS.

**TABLE 1**

Capacity study inputs.

<table>
<tr><th>Input</th><th>Comments</th></tr>
<tr><td>Geological block model</td><td><ul><li>3 dimensional</li><li>Classified in accordance with the JORC Code (2012)</li><li>Undiluted</li></ul></td></tr>
<tr><td>Financial</td><td><ul><li>Metal price</li><li>Discount rate</li><li>Government royalty</li><li>Foreign exchange rate</li></ul></td></tr>
<tr><td>Market constraints</td><td><ul><li>No constraints applied. The approved PFS BOD assumed the market would accept all product.</li></ul></td></tr>
<tr><td>Metallurgical</td><td><ul><li>Head grade dependent metallurgical recoveries</li><li>Different recovery formulae for two different metallurgical domains</li></ul></td></tr>
<tr><td>Pit slope parameters</td><td><ul><li>Varying angles based on geotechnical domains</li></ul></td></tr>
<tr><td>Ore related operating costs</td><td>Fixed and variable components provided by lead engineering consultancy to allow tailoring for different plant throughput rates:<ul><li>Processing</li><li>General and administration (G&A)</li><li>Sustaining capital expenditure</li></ul></td></tr>
<tr><td>Capital expenditure</td><td><ul><li>Process plant, for a range of capacities</li><li>Tailings facilities, off-site infrastructure, supporting site infrastructure including mining facilities, for a range of capacities</li><li>Mining equipment ownership costs were incorporated into mining operating costs by way of an equipment leasing component</li></ul></td></tr>
<tr><td>Mining operating costs</td><td><ul><li>Mining costs for all ore and waste activities were developed for three different classes of mining fleet, for scenario analysis</li><li>Haulage costs based on haulage simulation and analysis</li></ul></td></tr>
<tr><td>Cut-off grades</td><td><ul><li>Threshold cut-off grade. The life-of-mine minimum grade delivered to either stockpiles or the processing plant. The threshold cut-off grade selected for the Capacity Study was greater than the economic cut-off grade due to unavailability of metallurgical test work at low-grades</li><li>Mill feed cut-over grade. Variable over time depending on system capacity and availability of ore</li></ul></td></tr>
<tr><td>Stockpile grade bins</td><td><ul><li>Three grade ranges were applied for material greater than the threshold cut-off grade (noting that ultra-low-grade material referred to herein is below the threshold cut-off grade)</li></ul></td></tr>
<tr><td>Ramp-ups</td><td><ul><li>Plant ramp-up curve was specified by the lead engineering consultancy which involved an 18-month ramp-up from commissioning to full capacity</li></ul></td></tr>
</table>

| Input | Comments |
|---|---|
| | • Mining ramp-up included a nominal pre-strip-mining rate in the year prior to plant commissioning to supply waste for run-of-mine (ROM) pad and other construction projects and to supply some initial ore for commissioning. Mining rates were at full capacity by the second production year |

# MINE PLANNING

The key steps undertaken in the mine planning process are explained in the following subsections.

## Develop mining block models

Mining block models were developed to approximate and incorporate the anticipated amounts of mining dilution and ore loss for the various scenarios considered. All mining projects incur mining dilution and ore loss as part of the overall mining process, resulting from several of factors. The common causes of mining dilution and ore loss are:

- internal waste zones that cannot be excluded during mining at the chosen SMU
- blasting that causes ore mixing with waste
- the accuracy of mining to ore/waste boundaries
- ore mixing on ore/waste boundaries between pit stages
- other operational causes including pit floor cleaning/non-level floors, truck dispatch errors, and ROM stockpile practices.

The mining block models were developed by regularising the resource block model to a range of uniform grid sizes representing SMUs, with an SMU approximating of the smallest practical block size that a geologist would mark out and a loading unit attempt to mine. Regularisation is only one method of developing a mining block model, and was the most appropriate method for this study.

The regularisation process combined all material, whether ore or waste, within the grid block and the grades were averaged over all the material. Cut-off grade analysis was then applied to the averaged SMU grade and any blocks that had average grades below the threshold cut-off were rejected as waste.

Waste that was incorporated into regularised ore blocks during this process was recognised as 'mining dilution'. Ore that was recorded as waste in regularised blocks that no longer satisfied the threshold cut-off grade was recognised as 'ore loss'.

A range of models were developed to review the impact of different grid patterns, and corresponding SMU sizes. Evaluation of the resulting models demonstrated the trend of increasing mining dilution and ore loss with increased SMU size, as shown in Figure 1, for ore tonnes and contained metal.

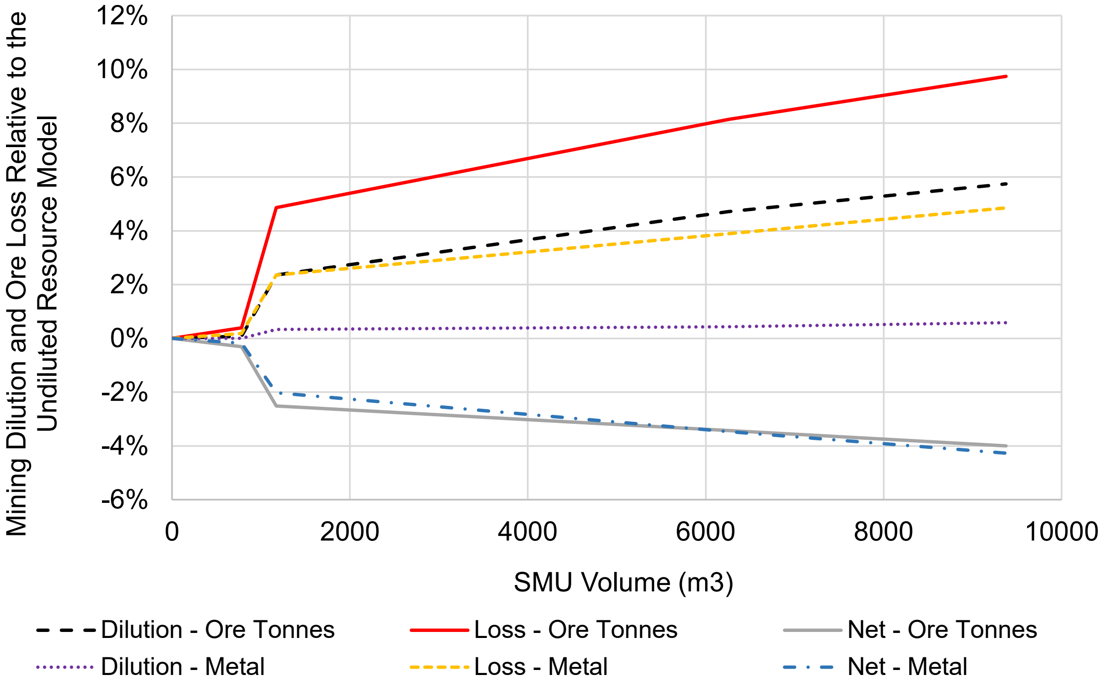


**FIG 1** – Mining dilution and ore loss, by SMU volume.

## Match the SMU size to the loading unit

Simple guidelines based on operating experience with similar machinery and material types were used as the basis for matching potential loading units to SMU sizes, to ensure that the selected machines were not too large for a particular SMU size.

In practice, the guidelines are affected by the ability and training of the operator, safe operating procedures for the mine, and the environmental conditions. Therefore, these guidelines were devised for average conditions over the life of the mine and may be conservative for some cases.

The adopted guidelines were:

- Loading unit bucket width to be less than 75 per cent of the block dimension in the strike direction.
- No less than 10 loading buckets of ore per SMU.
- Face shovels used where blocks are greater than 5 m high (approximately 6.5 m post blast at 30 per cent swell).
- SMUs of equal dimension in the X and Y dimensions on the basis that the deposit is wide, whereas a narrower deposit may warrant narrower SMUs across the strike direction.
- SMU X and Y dimensions should be equal to or larger than the bench height.

The equipment options considered for the Capacity Study were chosen to provide upper and lower limits, with the SMUs selected for analysis shown in Table 2.

**TABLE 2**

SMU and loading unit matrix.

| **Bench height (mZ)** | **Area (mX × mY)** | **360 t face shovel** | **530 t face shovel** | **830 t face shovel** |
|---|---|---|---|---|
| 5 | 12.5 × 12.5 | ✓ | | |
| 7.5 | 12.5 × 12.5 | ✓ | ✓ | |
| 10 | 25 × 25 | | ✓ | ✓ |
| 15 | 25 × 25 | | | ✓ |

A 15 m bench height could swell to approximately 20 m after blast heave, which is generally outside manufacturers' recommendations for 830 t hydraulic face shovels and some rope shovels. Operators do mine these bench heights with such equipment and this option was included to provide an upper limit for the Capacity Study, although shovel productivities were reduced to allow for inefficiencies associated with face hang-ups and associated bulldozing requirement.

## Match the truck size to the loading unit

Truck sizes were matched to loading units by considering the number of passes required to fill the truck. The trucks would be weight limited due to the heavy bulk density of the material. Ideal matches were selected based on three to four bucket loads per truck, with the resultant selected truck/loading unit size combinations defined as:

- Fleet A: 360 t face shovel loading 170 t truck – four passes (4.2 unrounded, fifth pass not taken, truck slightly underloaded).
- Fleet B: 530 t face shovel loading 220 t truck – four passes (3.6 unrounded, bucket only partially filled on fourth pass).
- Fleet C: 830 t face shovel loading 300 t truck – four passes (4.1 unrounded, fifth pass not taken, truck slightly underloaded).

## Match the loading to the plant capacity and mining rate

The loading unit size was matched to the plant capacity and associated mining rate by constraining the maximum and minimum number of units, so that only fleet sizes that would maintain operational flexibility but avoid equipment congestion were considered. Constraints were applied in scheduling to reduce loading unit numbers to incorporate practical considerations such as ramp traffic and loading unit working space.

The specific hierarchy of rules applied to ensure the loading and truck unit sizes were matched to the target plant throughput and the associated total pit movement scenarios were:

- Rule 1. Only the scenarios that required between approximately three and eight loading units to achieve the required mining schedule were considered further, for strategic scheduling in Minemax.
- Rule 2. Additionally, a mining rate constraint was applied for each stage that related to the estimated maximum number of mining fleets that could be accommodated on the same working bench in each stage.
- Rule 3. Additionally, a global mining rate constraint was applied for each annual period which related to the anticipated number of ramps available for haulage in the period, to limit haulage volumes and avoid truck congestion and interruptions on the ramps. It was assumed a maximum of 50 trucks per hour could travel each ramp, a function of an assumed safe following distance between trucks and truck travel speed, which would necessitate approximately three ramps – generally, one ramp for ore and two ramps for waste.

The assumption of a minimum of three or more loading units was specified to ensure that mining can continue in multiple faces without excessive movement of the loading units. Preliminary mining schedules indicated that in periods of steady state mining, generally three stages were being worked to maintain steady ore production and waste stripping.

Table 3 shows the matches resulting from the guidelines on loading unit size. Each case also shows the SMU size related to each mining fleet combination. Mining costs were developed to reflect the average cost of mining ore and waste using the respective loading fleets, giving consideration also to economies of scale associated with different mining rates.

**TABLE 3**
Plant capacity, mining fleet and SMU combinations.

| Plant capacity (Mtpa) | | | | | SMU size (mX × mY × mZ) | Mining fleet | | |
|---|---|---|---|---|---|---|---|---|
| | | | | | | A (360 t FS, 170 t truck) | B (530 t FS, 220 t truck) | C (830 t FS, 300 t truck) |
| 25 | 30 | 35 | 40 | 45 | 12.5 × 12.5 × 5.0 | ✓ | | |
| 25 | 30 | 35 | 40 | 45 | 12.5 × 12.5 × 7.5 | ✓ | ✓ | |
| 25 | 30 | 35 | 40 | 45 | 25 × 25 × 10 | | ✓ | ✓ |
| 25 | 30 | 35 | 40 | 45 | 25 × 25 × 15 | | | ✓ |

The approach in this analysis was used to allow rapid generation of options which had consistent guidelines applied. Detailed scheduling of the mine production would still be required at the appropriate time to ensure practical mining requirements are met.

## Develop an inventory

Interim pit stages and the revenue factor 1 pit shells were generated through pit optimisation using the Whittle implementation of the Lerchs-Grossmann algorithm.

Whittle develops a series of concentric, or nested, pit optimisation shells, each generating the maximum undiscounted operating cash surplus for the set of economic parameters used to develop that optimisation shell. The shells are developed by varying the product price above or below the base case price chosen for the Capacity Study – the revenue factor 1 product price. But once defined, all pit shells are evaluated at the revenue factor 1 product price, and resultant cut-off grade. The shell which uses the revenue factor 1 price will always have the maximum undiscounted cash flow for a particular series. Pit shells developed at revenue factors greater than 1 return negative operating cash flows, on an incremental basis for that part of the pit larger than the revenue factor 1 pit shell.

A matrix of scenarios was effectively developed whereby each of the scenarios had a unique combination of inputs, based on:

- Processing and G&A operating costs which varied dependent on processing rate.
- Mining costs which varied depending on bench height and mining rate; and
- Mining dilution and ore loss which varied based on bench height.

Eleven representative cases, drawn from the 30 cases derived from scenarios described in Table 3, that spanned the range of the plant throughput rates, SMU sizes and fleet combinations, were considered for pit optimisation analysis. The resultant pit inventories from pit optimisation are shown in Figure 2, in relative terms to each other. The sizes of the resultant revenue factor 1 pit shells were reasonably similar, with the total tonnes of the revenue factor 1 pit shells within approximately 15 per cent of each other. The size of the pit stages ultimately used in strategic scheduling are also shown in Figure 2.

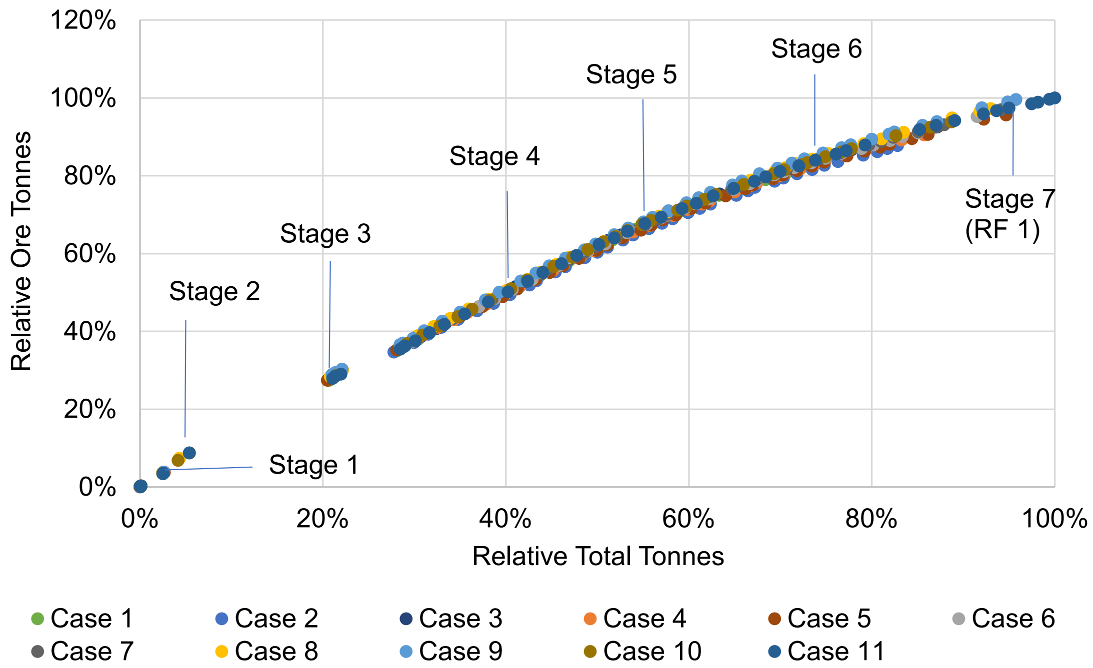


**FIG 2** – Relative ore tonnes versus relative total tonnes, by pit optimisation case.

Given the similarity of the pit limits from the pit optimisation analysis, one case was then selected as representative of the group for use in determining pit stage spatial limits.

The indicative pit stage limits were selected through evaluation of different pushback combinations using the Milawa scheduling module in Whittle. The largest pit limit considered at this point of the study was based on the revenue factor 1 shell. The combination of pit shells, being the interim pit stages and the ultimate pit shell, that returned the maximum discounted value when scheduled was selected. Then the minimum mining width module within Whittle was applied to modify the selected shells so that minimum mining widths of at least 75 m were incorporated to ensure safe, practical mining would be achievable.

Subsequent strategic scheduling using Minemax analysed the effect on discounted operating cash flows of excluding the larger, deeper pit stages from the schedule, as discussed in the 'Sensitivity Analysis' section.

Conceptual views of the pit stage spatial limits developed for the Capacity Study are shown in Figure 3. This one set of pit stages, or combinations thereof, were applied in strategic scheduling to each of the relevant mining block models.

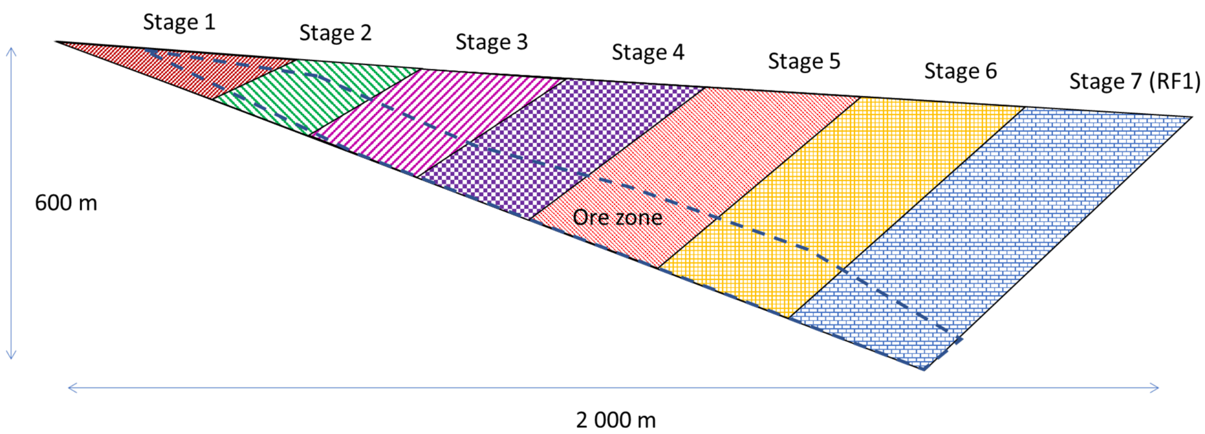


**FIG 3** – Conceptual pit staging – cross-sectional view.

## Strategic scheduling

Strategic schedules were developed using Minemax for each of the numerous scenarios considered, of which 38, as shown in Table 4, are summarised herein.

**TABLE 4**

Strategic scheduling scenarios.

| Plant capacity | Bench height | Stage | | | | Ultra-low-grade material included | Inferred mineral resources excluded |
|---|---|---|---|---|---|---|---|
| Mtpa | m | 4 | 5 | 6 | 7 | | |
| 25 | 7.5 | | ✓ | | | | |
| | 10 | ✓ | ✓ | ✓ | ✓ | ✓ | ✓ |
| | 15 | | | | | | |
| 30 | 7.5 | | ✓ | | | | |
| | 10 | ✓ | ✓ | ✓ | ✓ | ✓ | ✓ |
| | 15 | | ✓ | | | | |
| 35 | 7.5 | | ✓ | | | | |
| | 10 | ✓ | ✓ | ✓ | ✓ | ✓ | ✓ |
| | 15 | | ✓ | | | | |
| 40 | 7.5 | | ✓ | | | | |
| | 10 | ✓ | ✓ | ✓ | ✓ | ✓ | ✓ |
| | 15 | | ✓ | | | | |
| 45 | 7.5 | | | | | | |
| | 10 | ✓ | ✓ | ✓ | ✓ | ✓ | ✓ |
| | 15 | | ✓ | | | | |

The 5 m high bench height scenarios are not shown in this strategic scheduling analysis because this bench height did not allow functional schedules to be developed when the practical mining constraints and target plant feed rates were applied.

The scenarios shown in Table 4 include sensitivity analysis scenarios completed on the identified cases for stage and final pit limit selection, mining bench height and associated mining costs, exclusion of Inferred Mineral Resources from plant feed, and inclusion of ultra-low-grade material in plant feed (below the threshold cut-off grade). Each strategic schedule yielded detailed mining and processing physical and financial outputs to allow full analysis and comparison of each scenario.

Pit stage 5 was selected as the ultimate pit stage for the base case scenarios for all bench heights on the basis that inclusion of stage 6, or stages 6 and 7, only slightly improved NPV (<2 per cent), whilst approximately doubling the amount of waste material. This was tested by analysing the 10 m bench height scenario schedules for five different plant throughput rates ranging from 25 Mtpa to 45 Mtpa. Adopting stage 5 as the final pit limit did not change the decision of plant capacity, as discussed further in the 'Sensitivity Analysis' section.

The inventory for each case was scheduled to satisfy the required plant capacity for that particular case. Minemax was used to create the schedules to satisfy the following guidelines:

- Each schedule is based on the aforementioned pit staging, refer Figure 3.
- Maximum mining rates were applied depending on the plant capacity and mining equipment constraints.

- Maximum mining rate for each stage that related to the estimated maximum number of mining fleets that could be accommodated on the same working bench, based on the average area of each stage.
- Maximum mining rate related to the anticipated number of pit ramps available for haulage, to incorporate consideration of truck traffic. It was assumed a maximum of 50 trucks per hour could travel each ramp (based on an adopted minimum safe distance between trucks, and truck travel speed), which would necessitate approximately three ramps in most cases (one ramp for ore haulage, two ramps for waste haulage, with some crossover).
- Stockpiling and reclaim strategy using grade bins. Base case scenarios were completed using mineralisation above the threshold cut-off grade, whilst sensitivity analysis was undertaken including ultra-low-grade material as plant feed. The inventory above the threshold cut-off grade was split into three portions, based on different grade ranges, to represent low-grade, medium-grade, and high-grade ore, that provided granularity and allowed the variable mill feed cut-over grades to be determined over time. Mineralisation marginally below the threshold cut-off grade was identified for separate stockpiling and referred to as ultra-low-grade.
- Vertical rate of advance (VRA) in the pit was limited to eight benches per annum after analysis of mining cycles and bench turnover rates. This limit was applied to ensure that the production schedule would not exceed a practical, long-term sustainable mining development rate.
- Plant ramp-up to full production by the start of the third production year.
- Mining ramp-up to full production by the start of the second production year.
- Stockpile size limits applied based on-site layout constraints.

## Analyse results

For the 38 strategic schedule scenarios analysed, operating costs and revenue schedules were developed from the mining and treatment schedules, and capital costs were added to the operating cash flows to provide an overall cash flow model for each case. Note that mining equipment ownership costs were included in the mining costs as an operating cost, modelled as a leasing cost. A range of financial and physical measures were reported for the cases to enable ranking of the plant capacity options, as summarised further in the 'Summary and Recommendations' section.

Figure 4 shows the NPV for each plant capacity/mining equipment configuration scenario and indicated the highest NPV was achieved for a plant throughput rate of 40 Mtpa when combined with mining 7.5 m and 10 m high benches, and 45 Mtpa when combined with mining 15 m high benches.

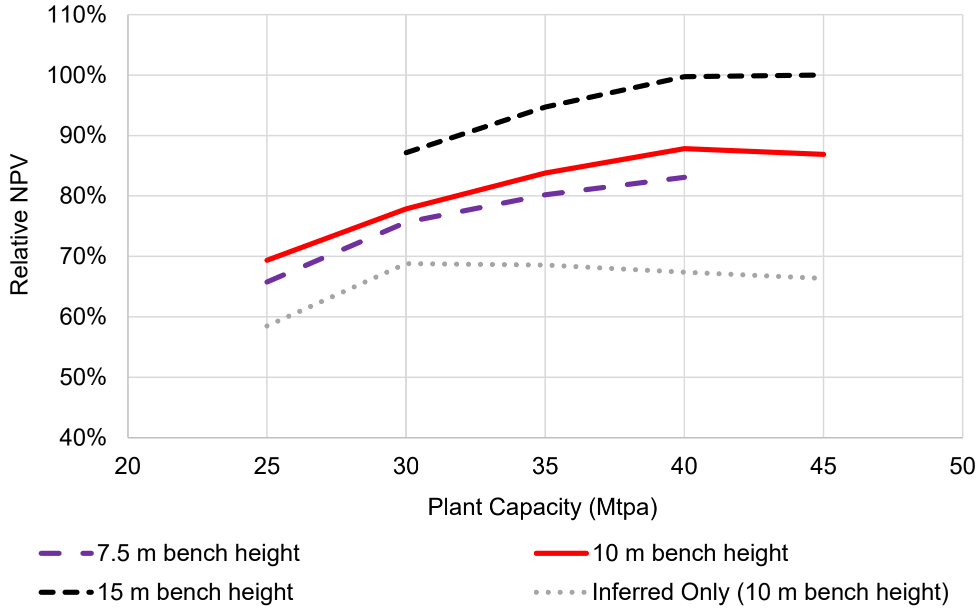


**FIG 4** – NPV versus plant capacity, by bench height.

Any gains in selectivity from mining the smaller bench heights, at any of the plant throughput rates tested, were more than offset by the higher unit mining costs associated with a smaller bench height. This was attributed to the reasonably low (<5 per cent) net effect on the ore tonnes resulting from the anticipated mining dilution and ore loss.

A plant throughput rate of 45 Mtpa was not recommended, despite this rate ranking the best in terms of NPV (but not other metrics). With increased plant throughput rate, mining rate increased and physical mining constraints relating to VRA and equipment numbers had greater effects, resulting sometimes in undersupply of ore to the plant and significantly reduced stockpile balances, as shown in Figure 5. Operationally this carries more risk as there was a reduced buffer against unexpected interruptions to ore supply. The rankings did not capture the risks, particularly for the 45 Mtpa case NPV and IRR metrics, which showed only modest improvement compared to the other plant capacity cases.

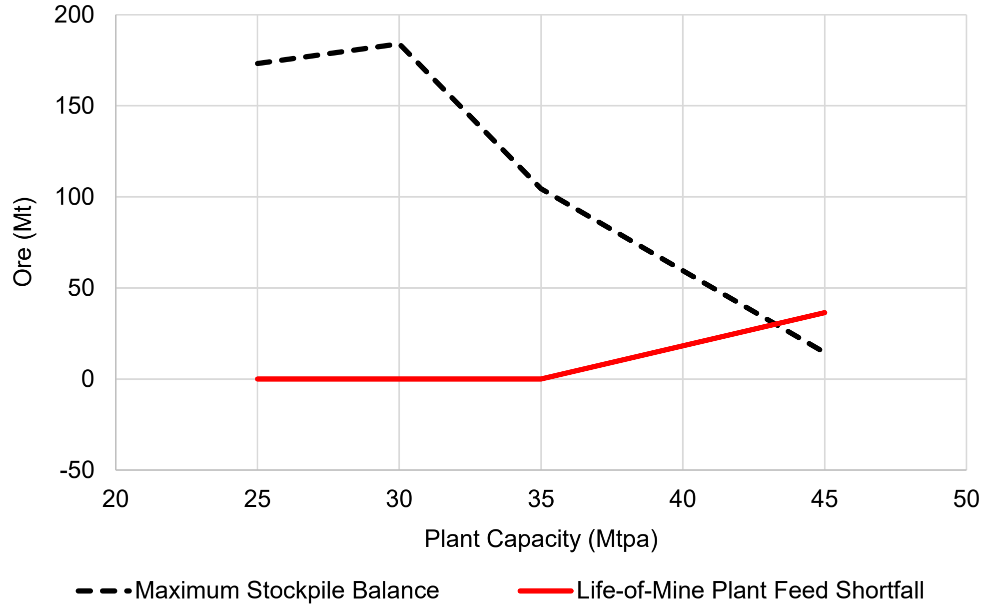


**FIG 5** – Stockpile balance and plant feed undersupply versus plant capacity.

Figure 6 shows the ratio of NPV to initial capital expenditure, and in all cases, irrespective of the selected mining equipment and bench height, the highest ratios were achieved for plant throughput rates between 30 Mtpa and 35 Mtpa.

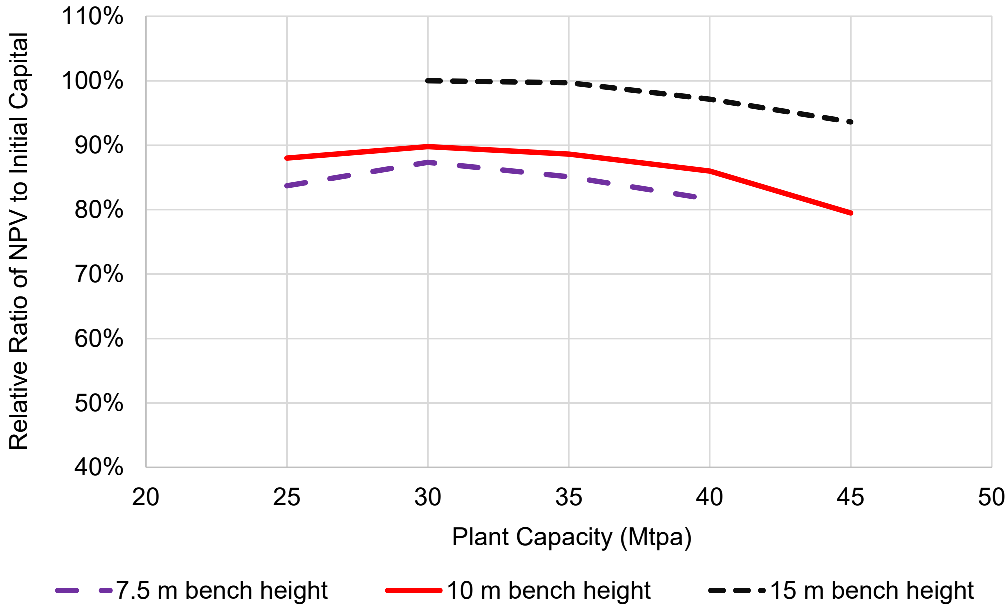


**FIG 6** – Relative ratio of NPV to initial capital, by plant capacity.

The peak IRR for the 15 m bench scenarios was achieved at a plant capacity of 45 Mtpa, although differences in IRR were negligible (<3 per cent) for the plant capacities ranging from 35 Mtpa to 45 Mtpa.

For the 7.5 m and 10 m bench scenarios, irrespective of the selected mining equipment and bench height, the highest IRRs were achieved for plant capacity between 30 Mtpa and 35 Mtpa, although in several cases the differences were negligible. For example, differences between the 30 Mtpa, 35 Mtpa and 40 Mtpa 7.5 m bench height scenarios, were negligible (<1 per cent).

## Sensitivity analysis

Analyses were completed on the sensitivity of the plant capacity decision to changes in the final pit limits (revenue factor 1 shell versus various smaller pit shells), exclusion of Inferred Mineral Resources from plant feed, and inclusion of ultra-low-grade material as plant feed. For these analyses, the 10 m bench cases only were considered as the outcomes were relative to other bench height options for the same plant capacities.

The decision of the plant capacity was independent of the final pit limits selected, within the ranges tested.

Figure 7 shows the NPV for different plant capacities using different pit stages as the final pit limits. The same NPV was materially achieved when selecting stage 5 as the pit limit compared to scenarios that also include larger stages. Choosing the smaller stage 4 reduced the NPV.

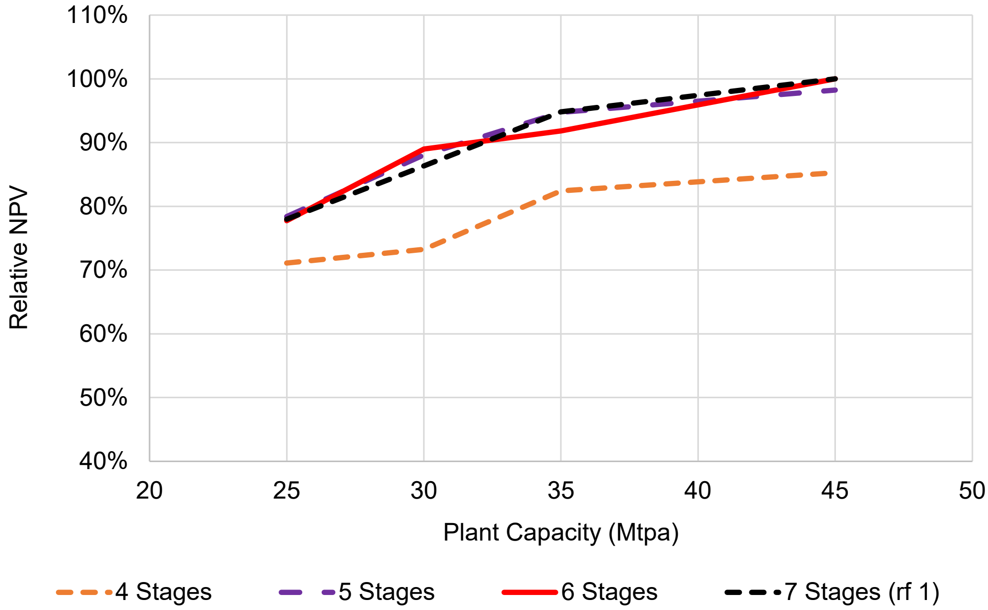


**FIG 7** – NPV versus plant capacity, by final pit limits.

The inclusion of ultra-low-grade material in the plant feed had negligible effect on the NPV and did not affect the decision on the plant capacity. This is because it would be processed at the end of the mine life so it would not displace higher grade plant feed. Its associated revenues were therefore discounted more than material processed early in the mine life.

Exclusion of Inferred Mineral Resources as a plant feed source did reduce the NPV, as shown previously in Figure 4, and would affect the decision on plant capacity. For this analysis, when considering just NPV, the 30 Mtpa case had a higher NPV, but other factors to consider with the exclusion of Inferred Mineral Resources were:

- Stages 1 to 4 only remained economic as they contained the majority of the Indicated Mineral Resources. Pit stage 5 was uneconomic and was therefore excluded from the mining schedule.
- A reduction in total plant feed tonnes of 50 per cent resulted, relative to the pit stage 5 case which included Inferred Mineral Resources.
- Mine life generally halved for all the plant throughput rate scenarios.

- Payback period generally doubled.
- Mining rate constraints, coupled with a reduced inventory, resulted in an undersupply of ore to the plant in some years.
- It would be likely that the majority of Inferred Mineral Resources would convert to Indicated, based on previous drilling and resource modelling, but there remained tonnage and grade uncertainty.

## SUMMARY AND RECOMMENDATIONS

The results of the strategic scheduling scenarios referred to in Table 4 were consolidated and ranked approximately in Table 5, from best (1) to worst (5).

General observations:

- Increasing the plant throughput rate increased the NPV and IRR but required greater initial capital expenditure.
- All scenarios ranked equally for payback period.
- A plant throughput rate of between 30 Mtpa to 35 Mtpa ranked best when measured by the ratio of NPV to initial capital expenditure.
- Considering all the metrics presented in the analyses, the optimum plant capacity appeared to be 30 Mtpa to 35 Mtpa with diminishing returns for increasing investment above 35 Mtpa.

A plant throughput rate of 45 Mtpa was not recommended, despite this rate ranking the best in terms of NPV (but not other metrics). With increased plant throughput rate, mining rate increased and physical mining constraints relating to VRA and equipment numbers came into greater effect, resulting sometimes in an undersupply of ore to the plant and significantly reduced stockpile balances. Operationally this carried more risk as there was a reduced buffer against unexpected interruptions to ore supply. The rankings, particularly for the 45 Mtpa case NPV and IRR metrics, did not capture these risks.

**TABLE 5**

Ranking of scenarios by plant capacity.

| Plant capacity | NPV | IRR | Max cum negative undiscounted cash flow | Payback period | Total initial capital expenditure | NPV – initial capital expenditure ratio |
|---|---|---|---|---|---|---|
| (Mtpa) | Rank | Rank | Rank | Rank | Rank | Rank |
| 25 | 5 | 5 | 1 | 1 | 1 | 3 |
| 30 | 4 | 3 | 2 | 1 | 2 | 1 |
| 35 | 3 | 1 | 3 | 1 | 3 | 2 |
| 40 | 2 | 2 | 4 | 1 | 4 | 4 |
| 45 | 1 | 4 | 5 | 1 | 5 | 5 |

Other general observations were that the decision of the plant throughput rate was:

- Independent of the final pit limits selected, within the ranges tested. Selecting pit shells smaller than the revenue factor 1 pit shell resulted in generally the same project value but required less waste movement, resulting in a smaller environmental footprint and reduced exposure due to a shorter mine life (operational, cost and price risks). Additional cutbacks could be added at a future time when knowledge of the deposit and processing performance would be better understood.

- Independent of the inclusion of ultra-low-grade material as a plant feed source. Including this material had negligible effect on the NPV.

Excluding Inferred Mineral Resources as a plant feed source did reduce the NPV and affected the decision on plant capacity. Pit staging was affected, with stage 5 becoming loss making. Plant feed reduced by approximately 50 per cent with a corresponding shortening of mine life.

Subsequent detailed mine planning would incorporate decisions based of the Capacity Study findings and other PFS trade-off studies. The next immediate decisions required, that stemmed from the Capacity Study, were confirmation of the plant capacity by:

- The mining company. Review and acceptance of the Capacity Study, culminating in a decision on the plant capacity and authorisation to proceed with the PFS using the agreed plant throughput rate as the PFS BOD.
- The lead engineering firm. Providing a target range enabled more detailed work by the lead engineering firm to refine the processing rate, within the recommended range of 30 Mtpa to 35 Mtpa, to reflect the combination of specific plant components ultimately selected.

## CONCLUSION

The tailored approach presented in this paper was applied in an actual consulting scenario, with much input and feedback provided by various stakeholders throughout the duration of the Capacity Study – primarily the mining company and its independent reviewer, the lead engineering firm, and the mining consultancy team.

Applying this tailored approach enabled a sufficient range of scenarios to be considered, supported by robust mine planning specific to the project, which highlighted the benefits and risks of each project configuration. This approach allowed the study team to articulate and demonstrate the reasons for various decisions, thus validating the approach as auditable and defendable. The availability of detailed cost models as both inputs and outputs to the Capacity Study, and detailed strategic schedule outputs, allowed queries from the reviewer and other stakeholders to be answered in an efficient and comprehensive manner.

The Capacity Study ultimately allowed the mining company to make an informed decision about the processing plant capacity, cognisant of the physical and financial risks, thus forming the cornerstone of the PFS and allowing it to progress.

The approach, although delivering a tailored solution for this specific project, could be applied to any greenfield project where the plant capacity is yet to be finalised.

## REFERENCES

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# Re-defining Compliance Planning through data integration and ultra-short-term scheduling

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## ABSTRACT

Compliance Planning is an active approach to ensure mining operations align more closely to their longer-term plans. Through Compliance Planning, operations can not only measure their performance on intervals as short as intra-shift, but they can then apply corrective measures to ensure they remain as close as possible to the plans. Compliance Planning targets short-term and execution level teams, as these are the shortest duration planning divisions commonly found in mining operations. Compliance Planning combines live information with intelligent planning capabilities to provide a framework that enables mining operations to realise a significant improvement in reconciliation of plan to actual movements.

## INTRODUCTION

Mining operations are as varied as the grains of sands on a beach. Every operation is unique in some fashion. The same commodity with identical characteristics can still demand completely different approaches and operations based on geography, leadership, and experience. Despite these differences, there are several commonalities that are found in nearly all mining operations. Every operation needs to prepare a plan for what they will produce and how, and every operation must then also follow this plan up with a reconciliation of how well they performed against this plan. This exercise is not one to be taken lightly as it drives many of the decisions an operation will make in the short and medium term. Despite the critical nature of the reconciliation exercise, the approach and technology applied to formulation of reconciliation is often antiquated and manual. Technology advancements over the last several years, however, has now made it possible for operations to adopt a far more active role in managing the gap between plan and actuals.

This active engagement is referred to as Compliance Planning. Through Compliance Planning, operations can not only measure their performance on intervals as short as intra-shift, but they can then apply corrective measures to ensure they remain as close as possible to the plans. Compliance Planning targets short-term and execution level teams, as these are the shortest duration planning divisions commonly found in mining operations. Compliance Planning combines live information with intelligent planning capabilities to provide a framework and tools to ensure mining operations remain closely aligned to plans.

## TERMINOLOGY

It is important to define some basic terminology first. Different operations around the world may use different terminology or apply these terms in a different context than what is defined here. For the purpose of this paper, the following terms will apply:

- Reconciliation
    - Reconciliation is a broad term used to articulate both the exercise undertaken to determine the gap between planned material movements and the material that was actually moved. In many cases, the actual material moved is initially determined based on quantities tracked throughout the period, the final determination will often use spatial measurements between surfaces. This is often conducted at the end of a month and serves as a marker to measure against progress throughout the year.
    - Reconciliation is a reporting exercise that usually looks at the accumulation of movements and the variations between two static pit positions (end of previous month, end of current month).
    - Often when measuring material movements from shapes that are not completely mined before the end of a reporting period, surfaces are modified to represent *in situ* progress through a block. This then considers quantities and qualities compared to the modelled

values; though it doesn't allow a planner to accurately consider the variation that has been imposed through blasting.

- Conformance
  - Conformance is defined here as the movement of materials from the pit as defined in the plan (Figure 1). This needs to be considered for quantity, shape, location, and in sequence. Consideration of start and end times and dates are not required, as the end point of reconciliation will account for the variations in the reconciliation exercise.
  - Quantity is inclusive of all volumetrics, tonnages, and qualities associated with the shape. This is important, as variations in density, grade, and material needs to be considered when aligning plan to actual.
- Non-conformance
  - Non-conformance is alternate movement of material that effectively meets nearly all the same criteria as conformance (ie it consists of material defined by shape, quantity and location), except it has occurred out of sequence as defined in plan.
- Non-compliance
  - For the purposes of this paper, non-compliance represents the movement of material that has occurred during the nominated period that is neither from the same location, nor within sequence.
  - While compliance has not been defined here, compliance can be viewed as alike to conformance (Figure 2).
- Planning and scheduling
  - Though the two terms are often used interchangeably, for the purposes of this paper, the terms can be separated in order to define to a greater level of detail regarding the tasks and assumptions that need to be established.
  - Planning consists of the determination of the areas in which a commodity is to be recovered, usually based on a mineable grade and quantity. Planning also consists of establishing the set of parameters (eg quantity, capacity or rate) that will be used to define the ability of the mine to produce a product.
  - Scheduling operates in conjunction with planning to determine the sequence in which activities must occur in order to meet a set of targets or objectives. A schedule utilises the parameters, constraints, and assumptions developed in planning to establish the sequence of activities over time.

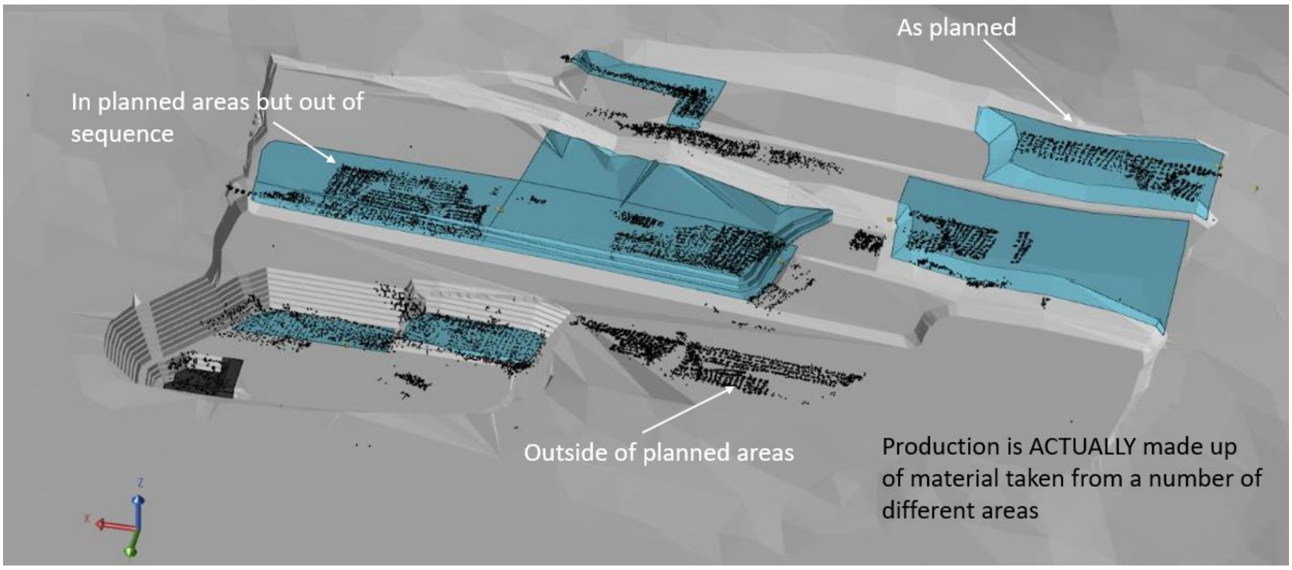


**FIG 1** – Bucket position GPS locations against monthly plans.

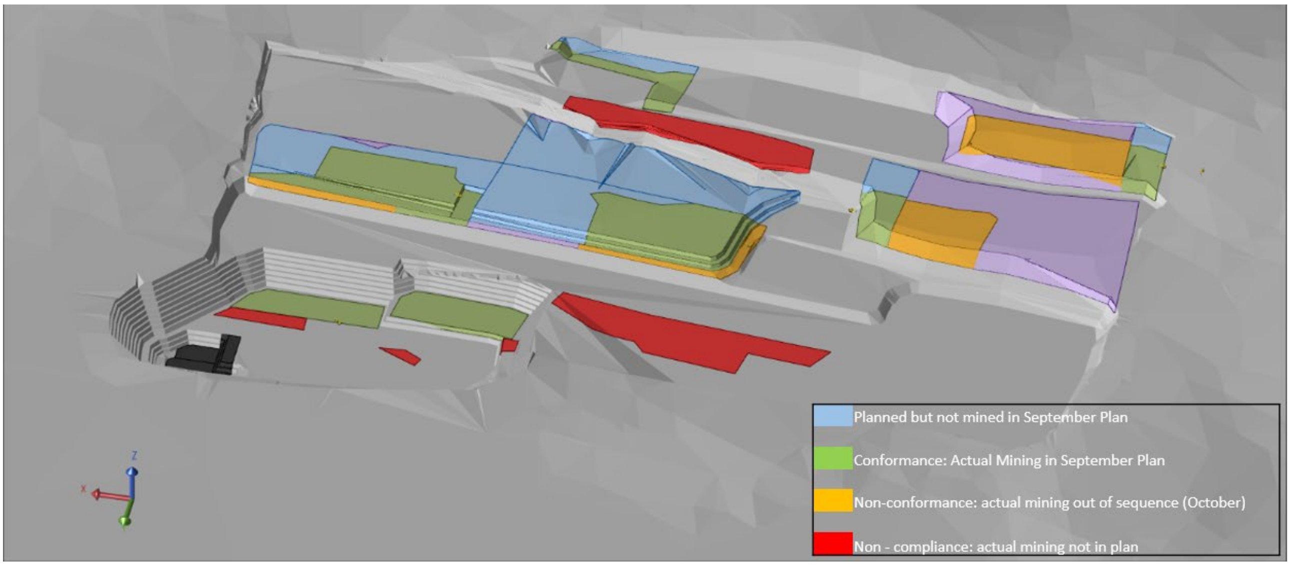


**FIG 2** – Breakdown of plan versus actual areas.

## CURRENT STATE – RECONCILIATION

Reconciliation is an exercise and outcome commonly used in most mining operations today. The act of reconciliation has two core associated functions:

1. Reconciliation is used to determine the performance of operations as measured against planned activities over the same period of time.
2. Establish an audited, consistent progress point for the plans against which subsequent progress will be measured. In plain terms, it is establishing a fixed status of the mine operations that measures performance since that last status point and the starting point for the next exercise.

While the methods used to establish this status point are not the core topics of this paper, it is acknowledged that there is no gross, fundamental issue with the approaches. Rather, this paper explores alternative methods of operational planning and scheduling so that the outcomes of reconciliation can realise a greater degree of alignment to plan.

### Myth 1 of short-term planning

One of the key and prevailing myths of short-term planning is that it is simply a long-term plan with greater resolution. This is a key contributor to the approach typically undertaken by operations to establish short-term plans – that the long-term plan is to be executed by taking a schedule and simply dissecting it into a number of smaller parcels of the larger parcels scheduled in the long-term plan.

This approach is fundamentally flawed as it assumes the same principles of making assumptions applies to both the long-term plan as the short-term, or operational plans. This places extraordinary pressure on the long-term plans to be developed to a level of detail that is not realistic so that the shorter-term plan can be derived to an appropriate resolution. This approach would be akin to playing a round of golf with only a driver. While the club may be effective in placing the ball within range of the hole, it is simply the wrong tool to be used for the approach shots – never mind putting on the green.

Alternatively, short-term plans tend to be regularly decoupled from the longer-term plan, effectively rendering longer-term plans obsolete almost from the moment they are published. This approach, while common, can be even more fallible, where there is a constant disconnect between planning and operations. Ultimately, this leaves mine management personnel in the position of navigating virtually in the dark.

This black and white approach creates an unhealthy discord in operations. Whereas a compromise between the planning horizons can bring much greater stability to the planning process, with short-term planning taking tactical responsibility for plans while the longer (ie medium and longer-term plans) provide the strategic direction.

## Myth 2 – There is a right or a wrong duration for short-term planning

Every mine planner will have a perspective on what the correct duration for a short-term plan should be. In truth, there is no one correct answer. Each operation needs to determine the appropriate duration for a short-term plan depending on tactical requirements. Operations with greater variability in production rates, material classification management requirements or greater uncertainty in geology will require a shorter-term plan.

Ultimately, the driver for the determination of a short-term plan resides in establishing what that tactical requirement is. Perhaps the most robust component of this tactical requirement is to remove the reliance on 'assumptions' in the short-term plan. In other words, at what time horizon can you confidently reduce or eliminate planning assumptions? As an example, while a planner may use an assumed availability and utilisation to define the production rate of an excavation unit, they may find that for the next two weeks they can comfortably define and confidently plan for that digger to work X number of hours each day.

For the purposes of this paper, planning durations are considered as simply short and long-term plans. There are, of course, a number of other categorisations that are also used, such as medium-term planning, budget planning, life-of-mine planning etc. These are all valid, well recognised categorisations, and have their own objectives. However, they can be treated as variations of the long-term plan referenced here.

# WHY ARE WE TALKING ABOUT PLANNING PHILOSOPHIES?

This background overview is required as the concept of Compliance Planning requires a fundamental shift in understanding how planning should be conducted. It requires the structure of defining what the existing definitions of planning are intended to accomplish – whether it's the longer-term strategic direction of a mining operation or the tactical execution of a shorter-term plan. By establishing the responsibility of each of these planning methodologies, the opportunities to maximise the outcomes in each methodology without the burden of preconceived definitions can be identified (Figure 3).

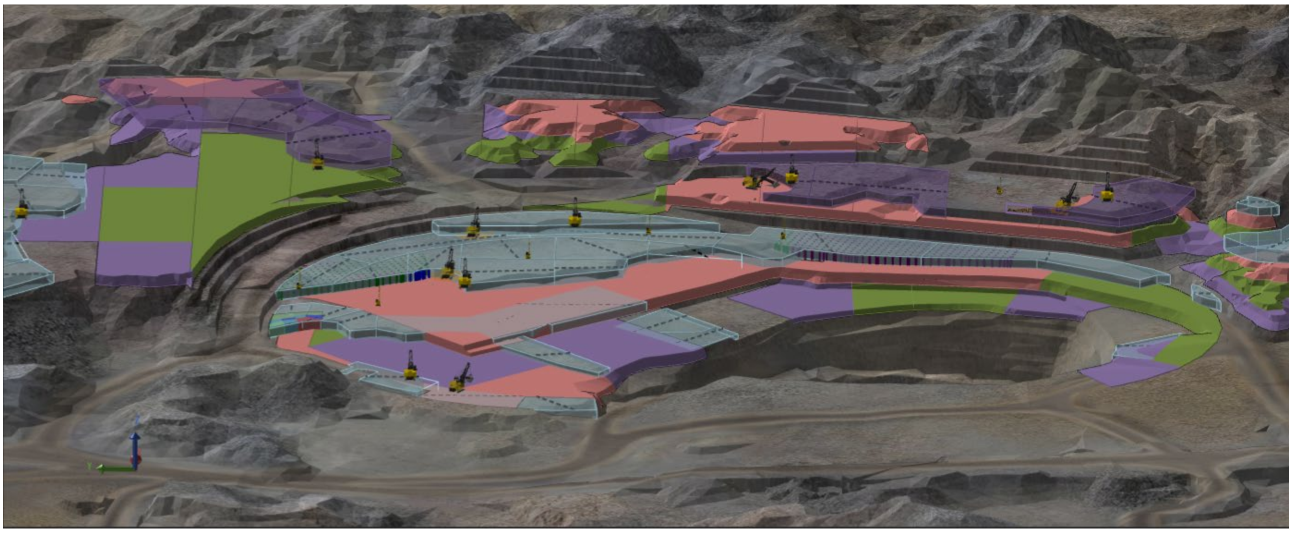

**FIG 3** – Aligning operational schedules and long-term planning horizons.

To establish Compliance Planning in the context of these other, well recognised definitions of planning, it becomes necessary to define not just what short and longer-term planning are, but how they interact with each other.

Long-term plans provide the overall direction of the mining operations and consider spatial orientation and quantity over time. The long-term plan provides both the overall scope of mining activities over a specified time frame, but also manages the expectations of production for the larger audience. The long-term plan essentially answers these questions:

- What is expected to be produced over the period of time?
- Where do we expect to mine to by a particular point in time?

- What equipment performance do we need to meet the targeted production and location?

Conversely, short-term plans are a detailed definition of the actions that need to occur in order to meet the expectations set by the long-term plan. Essentially, the short-term plan answers the following questions:

- How will we meet the expectations of the long-term plan?
- What operations and resources are required in order to achieve the long-term plan?

Taking this perspective, the interactions between the two horizons is more obvious. There doesn't need to be a clear separation between what is a long-term plan versus what a short-term plan is. Rather, we can consider it more from where the objectives need to be set. In most operations, the monthly horizon is often considered a sensible cut point between what needs to be done, and how it will be accomplished.

## WHAT IS COMPLIANCE PLANNING?

While the long-term plan provides the overall objectives, it usually remains more static, as it should. The long-term plan manages the expectations of the mining operations over a substantial length of time, and thus changes to it should be driven by enough variation in the base parameters used to drive the long-term plan to warrant the time and effort to do so.

The greater variations should occur in the short-term planning horizon, as this is both where the greatest number of variations should be expected, as well as where there is the greatest concentration of variables that will affect production. The drivers in short-term planning are focused on ensuring that progress continues towards the objectives set in the long-term plan. When one investigates deeper into the parameters that are in play, and there are a number of variables that can be adjustment, modified, and compensated for, the uncompromising nature of time means that those parameters that are tied most tightly to time (eg equipment production rates) are often the most important consideration in decision-making. Oftentimes, operations focus on keeping primary equipment 'being productive' for this reason. This is also why it is so critical that, to the best of our abilities, we keep the primary equipment working in the right place at the right time. This is the primary focus and objective of Compliance Planning.

Compliance Planning is the proactive counterpart to reconciliation and focuses on how to make planning decisions with the sole intent of remaining compliant to the long-term plan. This can take several forms within the short-term planning window. However, it is most effective in what can be described as the 'ultra-short-term' or execution space. This is usually anywhere from the next 12 hours to 72 hours and focuses on ensuring that within this short time window, a clear set of actions can be taken to resolve and address variations to plan information with the sole objective to remain as close to plan as possible. This can then extrapolate out further, so that by remaining compliant or as close to compliance as possible in the short-term window, operations inevitably remain more closely compliant to the longer-term plans. This also forms some of the triggers that maybe used to initiate re-scheduling and re-forecasting, where such tasks are only undertaken when either the base assumptions in a longer-term plan are adjusted, or a mine progress marker has been fundamentally missed.

## WHAT IS REQUIRED FOR COMPLIANCE PLANNING?

Compliance Planning is driven by the presence of enough information about current operations to allow users, observers, and decision-makers to be able to make decisions with confidence and in a timely matter. Compliance Planning, like many modern-day analytics applications, harnesses the massive amounts of data that is collected in a mining ecosystem (including FMS data, maintenance information, inventory management systems etc) and displays it in a sensible, intuitive fashion so that triggers for action can be easily defined.

Compliance Planning also requires the ability to overlay this information against planned information, as this provides the context against which performance can be measured. This also manages the trigger points at which a notification can be provided to the user and provide a call to action. The calls to action are unique to every operation, and thus the decision of 'what' action to undertake is likely a 'live' decision, as opposed to any effort to automate the response. That said, as technology and wisdom continue to grow, there is no reason not to expect that more and more of these responses could be automated and delivered without human intervention in the future.

Lastly, Compliance Planning requires a flexible platform that supports the ultimate need to modify plans should other methods of reparation prove inadequate to correct production variations. This platform requires the following capabilities:

- The ability to rapidly display or represent live information, or as close to live as possible. This can take the form of survey data, Fleet Management System (FMS) data, GPS locations, or any other relevant information that allows a compliance planner to rapidly ascertain what the current, updated status of the operation is. The more 'live' and non-interpreted the data is (ie requires no conversion of the data from one from to another, including importing and data entry), the easier it is for a planner to process the data as required (Figure 4).
- The ability to intersect the live information with planning information. With little data manipulation or interpretation, the compliance planner needs to be able to derive from information presented that 'the plan expects us to be at location X, and instead we are at location Y'. This allows the compliance planner to rapidly comprehend the triggers that had been raised and proceed towards resolution to assess the steps that can be actioned without the need to reschedule.
- The ability to modify the plan and assess the options for modifying the schedule while still reaching the expected long-term plan milestone (progress at the end of a period). This encapsulates the ability of the platform to run through scenarios from the current live state through the end of the next milestone. In addition to the operational limitations of the plan, the platform must have the ability to assess all downstream movement to understand the impact of the production variation as well as the ability to still hit the original long-term plan milestone.

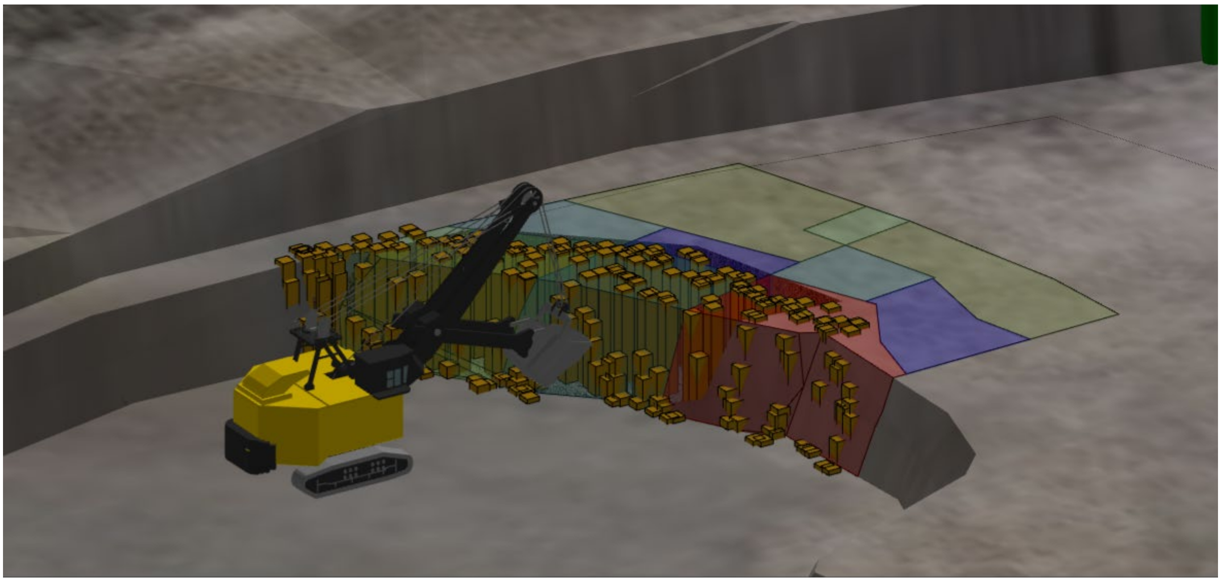

**FIG 4** – Live bucket positions against planned dig blocks.

## HOW DOES COMPLIANCE PLANNING WORK?

Compliance Planning is active planning and decision-making in the short-term space. Compliance Planning effectively operates as a decision support mechanism to identify, notify, and facilitate alignment to plan targets. By utilising a combination of live production information in conjunction with overlaying plan and historical behaviour, and a dynamic planning environment, the tools can be placed to ensure live tracking against plan and clear visibility of the mitigating measures that need to be taken to ensure plans remain compliant.

### Step 1 – Live planning versus actual

The first step with Compliance Planning is the establishment of some form of a dashboard or other live display of current production progress against plan. This requires an FMS that provides location, production progress, and sufficient data to be able to isolate each piece of equipment. Dashboards are the tool of choice in the case, as a dashboard will allow operations to configure the alignment of plan versus actual in the context that is most relevant to each operation.

In Figure 5, a sample dashboard is shown. Colour coding has been used to quickly draw the attention of users to areas of concern, with red being a concern, while green is in line with plan targets at that specific time in the shift. Rather than only focusing on the typical ongoing metrics such as availability, utilisation and tonnage moved, the dashboard also focuses on forward projections identifying if current progress will remain compliant to plan or fall short. The two forward projections shown below identify the projected total material for the shift based on current progress, as well as the rate that will need to be maintained to achieve that production.

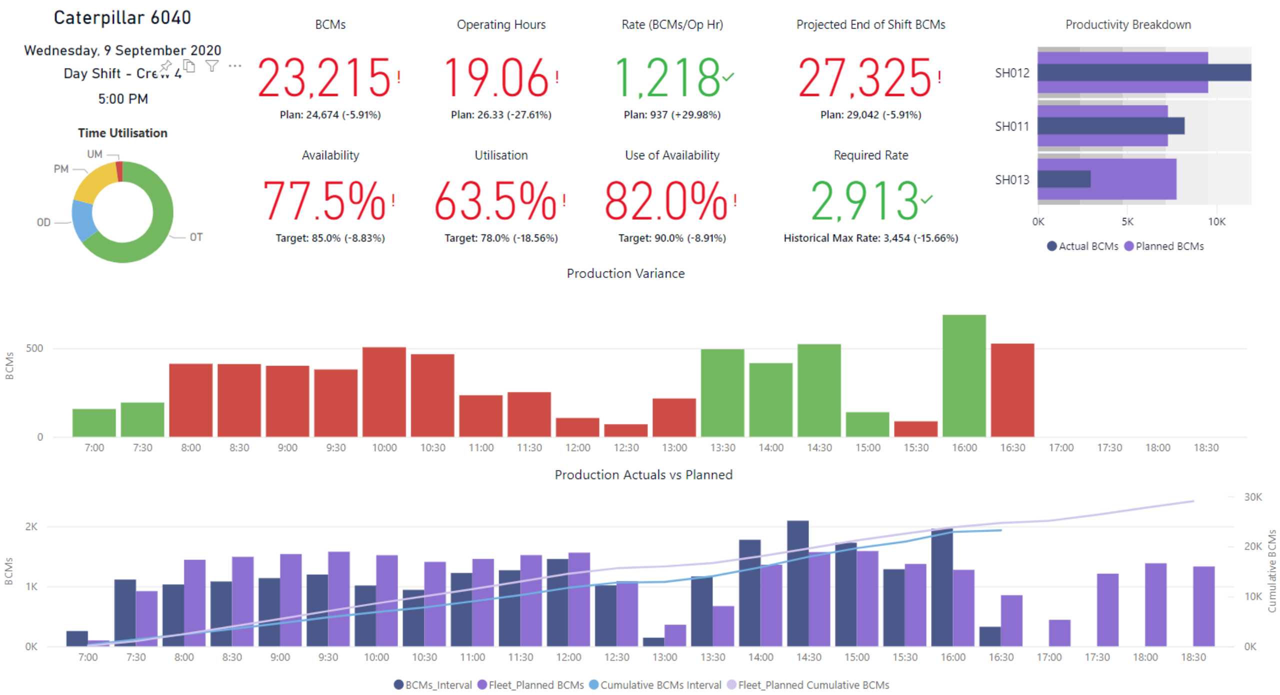


**FIG 5** – Typical Compliance Planning dashboard.

It is important in the use of these types of projections that consideration is made for the behaviour of equipment throughout the shift and the variability to follows. It would be erroneous to assume that productivity profiles follow a linear pattern throughout the shift. In this example, the historical performance of the particular piece of equipment with that particular operation is used to provide a behaviour template against which production is displayed against.

This is only a simple example of how the overlay of live information against plan information can be made. There are a multitude of other options, and each operation would have to determine what works best for their operation.

## Step 2 – Establish a rapid response list of productivity sprints

With the presence of a Compliance Planning dashboard, operations will effectively be able to quickly ascertain that to meet compliance for the current shift, action must be taken. With consistent, regular monitoring, it will be possible to identify that action is required. It is imperative that operations are prepared to take action as soon as triggers are activated, as there may be only a small window of opportunity to correct the course.

Such actions require the approval and support of management to ensure that there is minimal delay to act. Such a list would include options such as hot seating at changeover, re-allocation of trucking, limiting haul road traffic, preparation of contingency access, or delaying a schedule maintenance activity if safe to do so.

The intent here is to ensure that, while larger, system wide changes require considerable effort, time and buy-in to implement, we have the ability to make small incremental adjustments for a short duration to improve outcomes on a shift-by-shift basis. This aligns with a greater overall philosophy that, by making little changes through the course of a shift to make each shift more compliant, we can inevitably get closer to compliance over the course of a week, and thus over a month.

### Step 3 – Preparing for change

Inevitably, circumstances will emerge that defy any of the conventional productivity sprint responses to bring plans back into compliance. When this happens, the focus must shift from recovery in the current shift to preparation for the next. Rather than trying to correct the course, planners need to prepare to discover an alternative route to meeting the long-term planning milestones.

This requires a more dynamic planning environment than what is currently used in the longer-term space. There is rarely enough time for a short-term planner to carry through the entire planning process for a short-term correction. Short-term planners often fall-back onto spreadsheet calculations and assumptions to try to manage until the next short-term planning cycle. This effectively leaves operations continuing production without any real understanding of how well they are tracking.

This is akin to using a GPS to navigate to a destination. However, once a wrong turn is taken, rather than recalculating right away, the driver continues to drive in what they believe is the right direction until some time has passed and the driver pulls over to re-set the GPS.

This platform should have the ability to accept live data, easily update mine and demand chain status with inventory and progress and allow schedules to be generated quickly. It is necessary to have the ability to re-reserve and to adopt updated geologic considerations such as intrusions, grade control blocks, and other undetected anomalies. Finally, the platform needs to have the ability to realign (effectively recalculate on GPS) the current progress to the nearest long-term plan milestone.

### Compliance Planning in action

In practice, Compliance Planning would be incorporated into daily practice rather than a planned activity to be conducted within a longer period of time (ie a week). On a daily basis, data feeds are automated, so there would be no requirement to spend the first couple of hours each morning updating the status of operations and compile the results of the previous shift – as is commonly done today. Throughout the course of the shift, monitoring of the compliance dashboard is maintained, so that warnings and alerts can be assessed as they occur.

Planning would be updated with only brief adjustments and checks to ensure there is still compliance towards the long-term milestone. Where substantial change has occurred and more substantial changes are required, the dynamic planning platform described in step 3 would be engaged to ensure a practical plan is established that meets the long-term milestone. If that is no longer achievable, then the final trigger has been actioned, and the dynamic planning platform needs to generate an updated milestone (ie stage plan) for longer-term planning to be updated with.

## COMPLIANCE PLANNING RISKS

Compliance Planning introduces a shift in how mining companies conduct their operational planning. In the past, short-term planning existed as an extension of long-term plans, while operational or execution planning was conducted in a more tactical, manual manner. By creating a framework under which operations can assess and transition into a more structured approach for that tactical, execution planning space. Within this framework, operations can define the use cases for collected data from FMS and other digital systems that transforms this data into proactive instead of reactive actions. While the benefits of such a shift have been defined in this paper, it is also worth noting that there are some associated risks.

### Risk 1 – Change management

Implementing change in mining operations is a well tread path, and thus the industry is keenly aware of the challenges and the methods to execute change management safely. Where pivoting in the longer-term planning environment has a buffer that allows for change to be implemented over time, change management in the execution space has little room for error. As such, it is likely necessary that processes be run in parallel with previous execution processes before a full change-over can be realised.

### Risk 2 – Integration and enterprise

One of the core requirements of Compliance Planning is the need for timely and easily sourced data from FMS and other data capture systems. Without the ability to assess deviations from the plan in a timely manner, the benefits of compliance planner are significantly reduced.

## CONCLUSIONS

The mining industry has undergone significant technological change over the last 25 years. The adoption of technology has always been strong, though often with a more measured, steady adoption approach than what is used in other industries. With the massive improvements in network connectivity, computing capability and speed and data storage, coupled with exponential growth in data analytics and increasing intelligence in how people and technology can connect, the time for change is now. Best practice Compliance Planning is an approach that utilises these latest developments in technology to provide operations the framework to proactively improve their compliance to longer-term plans in a practical, executable manner with minimal disruption to current operational practice.

# The use of collaboration and management to turn-around the performance of a failing copper mine

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## ABSTRACT

This abstract discusses a heap leach copper mine that was failing to perform and as a result was not meeting targeted production. The mine had numerous technical challenges including: high water inflows, low levels of geotechnical monitoring, performance issues with mining, resource model and grade control reconciliation issues, and unexplained low levels of performance from the heap leaching operations.

A team from various organisations with specialist skill sets including: a forensic accountant, the owner's representative and a mining engineer, was arranged to visit the site. The task was to investigate the situation and determine what could be done to turn the performance of the mine around. After an initial assessment in late 2018, efforts were directed to better understanding the operations and improving the performance of the site through preventative management. A key part included the creation of production forecasts that could be achieved that would help restore confidence in the operations.

Uncertainty in the mine achieving production forecasts and the performance of the heap leaching operations were contributing to the decline of the operations. Investigations uncovered the fact that while the resource model could reliably predict total copper, a lack of sequential leach data meant the accurate prediction of soluble and insoluble copper species, both important for heap leaching, was difficult to do with any degree of certainty. The crushing and mining contractors also appeared to have suffered somewhat from a lack of management by the owner's team and performance had affected the production of copper.

Collaboration and management were used by the team to understand and stabilise the operations. Ongoing collaborative efforts by the management team, site personnel and contractors resulted in a positive outcome for the owners and their mine – and could at yours.

## INTRODUCTION

The Tschudi Copper mine is located in the Oshikoto Region of Northern Namibia and is about 20 km west of Tsumeb. Exploration has occurred in the Tschudi locality since the early 1900s and more recently diamond drilling by Tsumeb Corportion Limited (TCL) in the 1990s, and later Weatherly Mining Namibia Limited (Weatherly) undertook underground trial mining and diamond drilling between 2007–2008. In 2008 underground trial mining operations ceased due to low copper prices.

The author understands that significant local exploration in the Tschudi area was an infill drilling campaign was last completed in 2016, and more recently infill drilling in 2018–2021.

The timeline for the development of the Tschudi mine from exploration to current operations is (Anonymous, 2016):

- Construction of heap leach pads and the SX-EW plant commenced in January 2014.
- Pre-strip mining started in August 2014.
- First ore agglomerated and stacked in January 2015.
- First cathode was produced by February 2015 and nameplate production was reached in December 2015.
- Major water inflow into Pits in May 2018 and recovery operations.
- Team assigned to improve the performance of the mine in October-November 2018.
- Completion of mining in December 2019.

- Copper production due for completion in June 2021.

From December 2017 to April 2018 the mine achieved or exceeded forecast production (stripped) copper tonnes. However, after a major water inflow in May 2018, copper stripping was negatively affected with stripped copper 15 per cent lower than forecast from May to October that year.

From October 2018 to December 2019, the team assigned to improve the performance of the mine were heavily involved in the mine production and planning, financial management and copper leaching operations and what follows is a summary of the project background, some of the challenges that were faced by the team including site personnel and contractors, and how they were managed.

# GEOLOGY

## Regional geology

McKinney *et al* (2009) report that the Tschudi Project area is located within the Otavi Mountainland of northern Namibia, which forms part of the Northern Carbonate Platform of the Pan African Damaran orogen. Tschudi mine is situated on the southern limb of the Tschudi-Uris syncline, which is part of a folded sequence of dolomites in the Hüttenberg Formation (Otavi Group) overlain by sandstone units (Mulden Group). The footwall of the southern limb of the syncline consists of chert and dolomite of the Hüttenberg Formation. Van Wyk (2017) describes the syncline as about 2 km wide and 800 m deep.

## Local structure

The Tschudi-Uris Syncline dips between 25° and 35° to the NNW. The northern limb has a steeper dip of about 50° and the syncline plunges at between 5° to 9° to the west. The axis of the syncline trends between 070° to 085° (McKinney *et al,* 2009) and is illustrated in Figure 1.

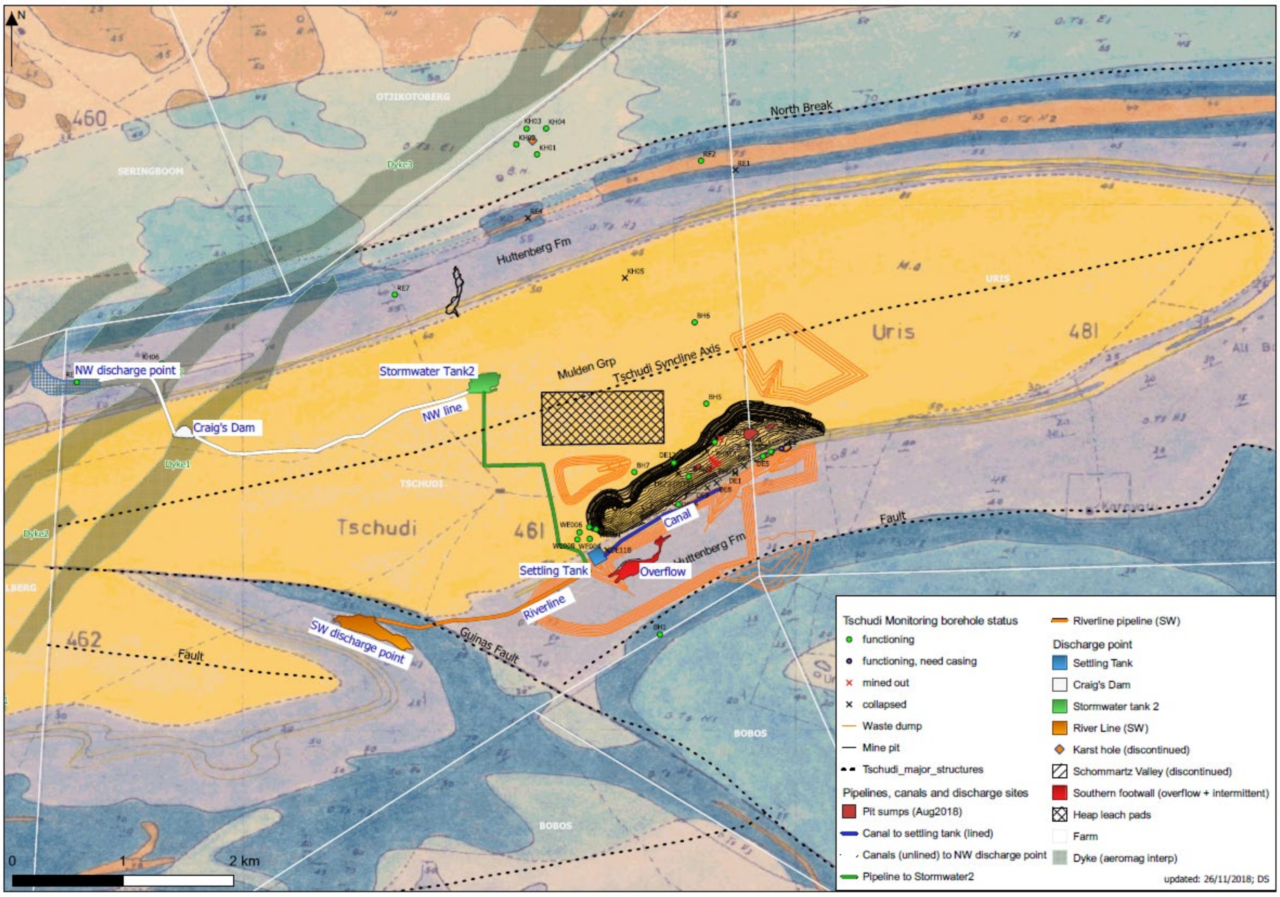


**FIG 1** – Tschudi geology and dewatering infrastructure.

## Mineralisation

The Tschudi orebody is hosted within the basal arenites and conglomerates of the Mulden Group. The copper mineralisation is preferentially developed in the base of the arenite sequence on the southern limb of the syncline, as a disseminated, continuously distributed roughly planar sheet, varying in thickness from 2 m – 40 m. The deposit has a strike length of about 2.5 km which is defined by drill holes.

Oxide mineralisation outcrops at surface and extends down to about 55 m, followed by a transition zone to about 75 m and then a predominately sulfide zone. Copper is the economic metal being targeted, for while silver roughly tracks the higher copper grades it is not recoverable using the Heap Leach/SX-EW process selected (Cameron, 2012).

The main copper minerals at Tschudi from dominant to less dominant are:

- Oxide zone (OX):
  - Malachite then cuprite, azurite, and minor amounts of chalcocite, digenite and covellite.
  - In the basal conglomerate and lower oxidised arenites the dominant minerals are malachite and chalcocite, and less amounts of covellite and cuprite.
- Transition: a mixture of oxides and sulfides (CC).
- Sulfides:
  - Bornite (BN) and chalcopyrite (CPY) are dominant, with some minor chalcocite and digenite.

## Geometallurgical modelling

In 2018 it was found that historic drill holes were assayed for total copper, acid soluble copper, cyanide soluble and insoluble (residual) copper only after 2008. Before 2008, drill hole assays were total copper only (1978–1997) and total copper and acid soluble (2007–2008). In 2007–2008 the soluble copper assay was by hot acid technique, which was not comparable with other acid soluble results.

Cyanide soluble copper was not assayed for because heap leaching was not considered at that time.

By late 2018 there was very little sequential assay data remaining and a decision was made to implement a grade control program using blasthole rigs. For these programs 18 m drill holes were designed on a 50 m by 15 m pattern and when drilled were logged and samples were collected at 1 m intervals. Samples were first analysed by XRF, and to ensure a quick turnaround of the sequential leach assays from the on-site lab the samples were to be composited into 3 m samples. The resulting assays were used to update the grade-control model for use in short-term mine planning.

### *Weathering zone surfaces*

Prior to 2009, oxide, supergene and sulfide surfaces were modelled but proved highly unreliable in the underground trial mining. In the 2009 Mineral Resource Estimate, McKinney *et al* (2009) model cells were classified as being oxide or sulfide based on a surface representing 50 per cent acid soluble copper, where greater than 50 per cent sulfuric acid soluble copper was classed as oxide, or base of weathering if deeper. This process produced a surface about 55–65 m below ground level. A mixed zone to about 75 m below ground level followed, then a dominant sulfide zone.

While McKinney considered it a reliable guide on other South African deposits, a more metallurgically effective surface was recommended to be derived in the future with metallurgical and mineralogical test work.

By 2015 the resource model was updated using previous and updated sectional interpretations and mineralisation envelopes (or grade shells) from copper assay values, with a cut-off of 0.15 per cent Cu. Lithological units were only generated for calcrete and dolomite and wireframe solids were separated into lower and upper, with domains defined by oxidised/leached (XOX) weathering zone, mixed zone (OXMIX), chalcocite zone (CHAL) and calcrete (CALC) (Herman, 2015). These ore type zones were used for mine scheduling until early 2019.

### *Mineral species*

In late 2018, work was started by Mining One to further refine the definition of weathering and associated metallurgical recoveries, with the current resource block model being re-coded for the type of mineral species using sequential leach data. A heat map of the sample points with their distribution based on acid soluble copper ratio (ASCu/TCu) and residual copper ratio (Cu_R/(Cu_R+CNCu)) ratios are in Figure 2.

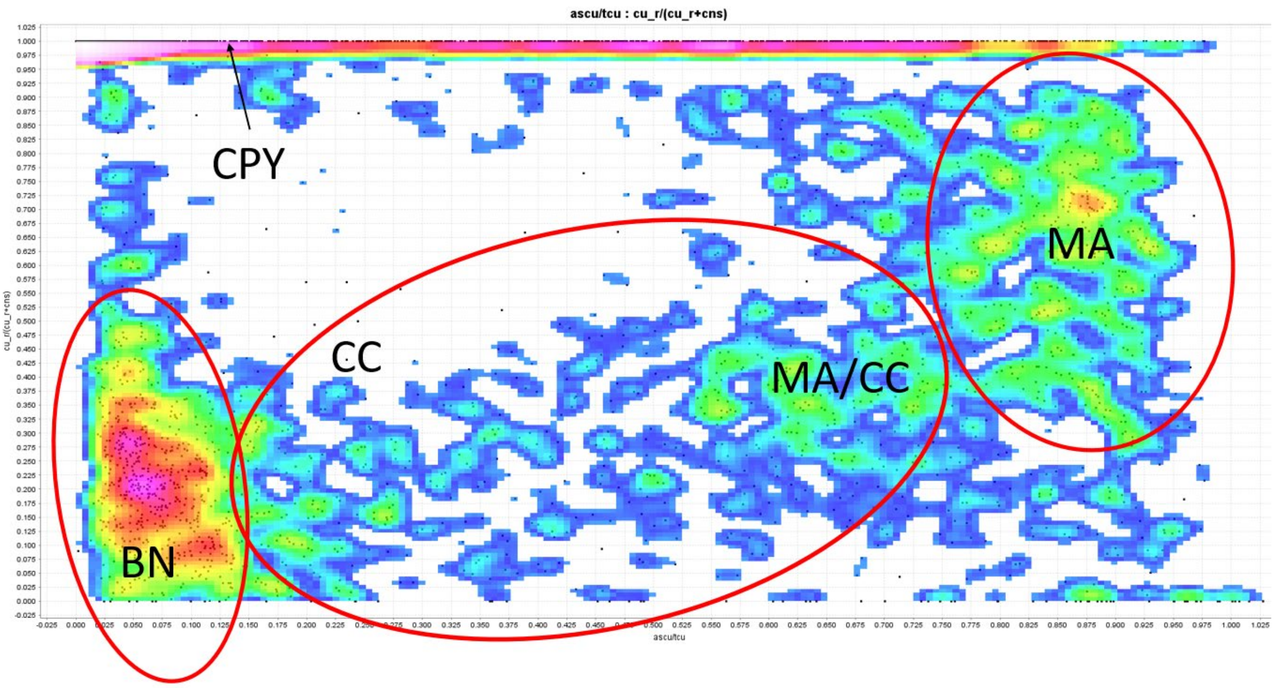


**FIG 2** – ASCu/TCu versus Cu_R/(Cu_R+CNCu).

The mineral species definitions used in the resource block model are as follows.

Chalcopyrite (CPY) where Residual Copper/(Residual Copper + Cyanide Soluble Copper Ratio) >0.65 and Acid Soluble Copper Ratio <0.2.

If not CPY then:

- Bornite (BN) where Acid Soluble Copper Ratio <0.2
- Chalcocite/Covellite (CC) where Acid Soluble Copper Ratio >0.2 and <0.5
- Malachite/Azurite (MA) coded to (OX) where Acid Soluble Copper Ratio >0.5.

A representation of the changes in the resource block model is shown in Figure 3.

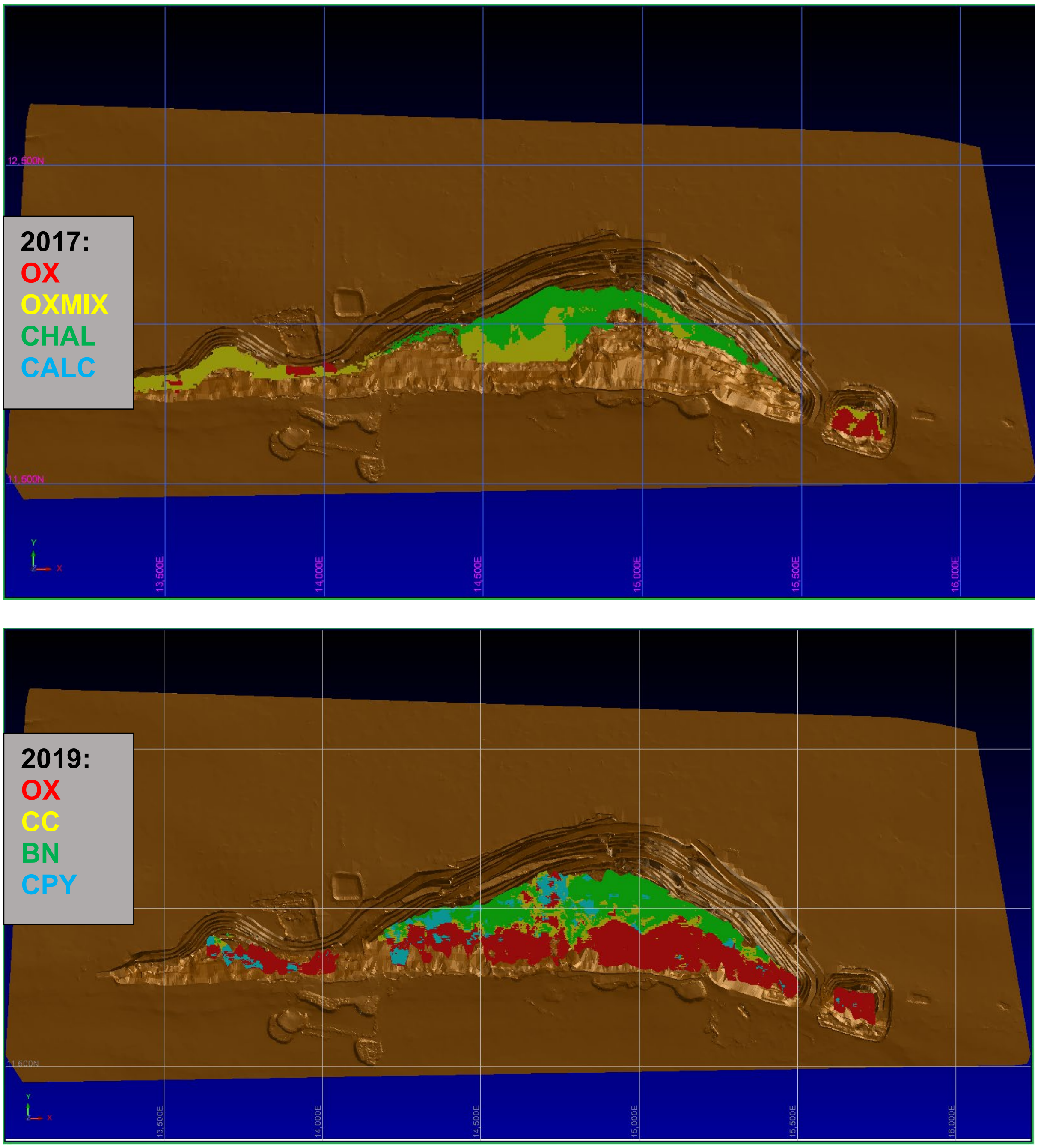


**FIG 3** – Pre-2019 and post-2019 mineral zones in block models (Cu >0.3 per cent).

Once the block model was updated for mineral species, processing recoveries were updated to reflect the expected heap leaching performance for each, and these are shown in Table 1.

**TABLE 1**

Ore type and expected leach recovery of Total Copper.

| Ore type | Recovery (%) |
|---|---|
| OX | 90 |
| CC | 70 |
| BN | 55 |
| CPY | 3 |

The expected heap leach performance was based on the Tschudi plant average for 2018 as reported by Jo and Loyola (2018b). Recovery for oxide matched the expected ideal for the Tschudi OX species but was lower than the expected (regular) recoveries of 79 per cent for CC, 65 per cent for BN and 4 per cent for CPY and was lower again compared to ideal recoveries. Heap leach performance and recoveries will be in more detail a little later.

## MINING AND CRUSHING

At Tschudi, mining was undertaken by a mining contractor using a drill and blast, load and haul production cycle. Excavators and truck fleets were equipped to move up to about 900 000 $m^3$/month and this was done by mining 12 m benches in four flitches of about 3.5 m in height (after allowing for heave).

As shown in Figure 4, generally the mine ex-pit movement tracked at or slightly below forecast. Issues with dewatering, geotechnical instabilities, equipment availability and occasional lack of prioritising of areas to be developed made this an ongoing challenge.

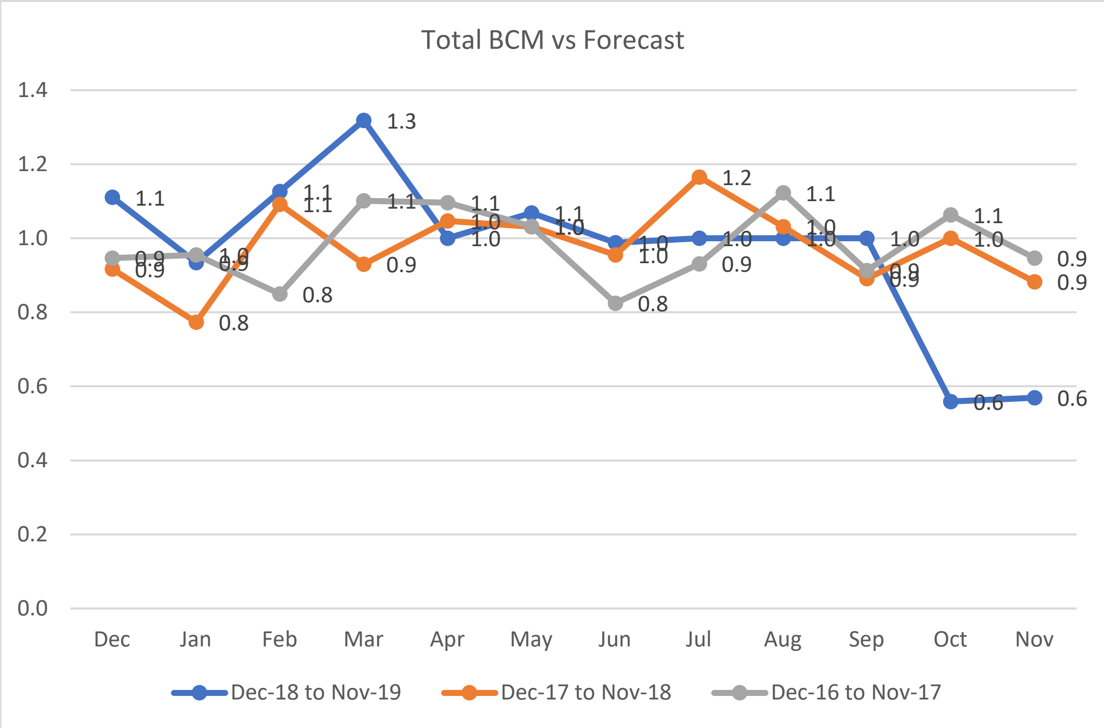


**FIG 4** – Total mine movement versus forecast.

The mining contractor also completed heap leach pad construction, while a separate contractor was engaged for crushing, agglomerating and heap leach stacking. Stacking could be carried out to a maximum rate of about 2.8 Mt/a.

### Mine planning

Mine Planning was completed by an on-site Mining Engineer using Minesched™ (by GEOVIA, Dassault Systemes). In late 2018, it was noted that the mine production forecasts had assumed a nominal 5 per cent grade dilution and 100 per cent ore recovery. However, after studying block model reconciliations to ore mined, ore tonnes over the preceding nine-month period were on average 17 per cent higher and grade 8 per cent lower.

To account for the differences, or at least partially due to the potential for reversal from monthly variations, grade dilution was increased to 10 per cent while ore recovery was kept at 100 per cent. Efforts were then directed to understand and improve grade control and mining practices including preferred dig direction and improving orebody knowledge (ore and waste identification) training for mining supervisors and key operators.

In October 2018, it was also recommended to bring forward oxide and chalcocite ores in mine planning to assist with earlier production of copper due to the faster leach kinetics and high ferric ($Fe^{3+}$) levels due to dissolution of $Fe^{3+}$ oxide minerals; solubility of secondary sulfides like chalcocite in sulfuric acid/$Fe^{3+}$ solution is higher than in acid only. To do this, upper-level oxide ores needed to be targeted and waste stripping to expose them was advanced in Pit 2.

## Hydrogeology

Opencast mining of overburden had exposed the top of the dolomite unit (footwall) and since 2016 mining had continued to advance below the groundwater table and in late 2017 was some 40 m below the water table. In 2016 in-pit pumping dewatering rates were about 1400 $m^3/h$ from two sumps (Pit 3 and Pit 4) in Figure 5.


**FIG 5** – Key dewatering distribution infrastructure at Tschudi (from Sarma, 2018).

In May 2018, after a routine blast in Pit 3 and Pit 4 that was next to the dolomite footwall a major fracture was affected (Figure 6) and an inflow of water resulted at an estimated peak inflow rate of 2700 $m^3/h$. During the event, groundwater levels dropped in the dolomite aquifer by 3 m and in Pit 4 the water level rose by 24 m over 19 days. Using a combination of diesel and electric pumps, pumping from pits was increased to an estimated 2500–3000 $m^3/h$ and to the end of May 2018, a total of 500 000 $m^3$ was pumped from the pits.

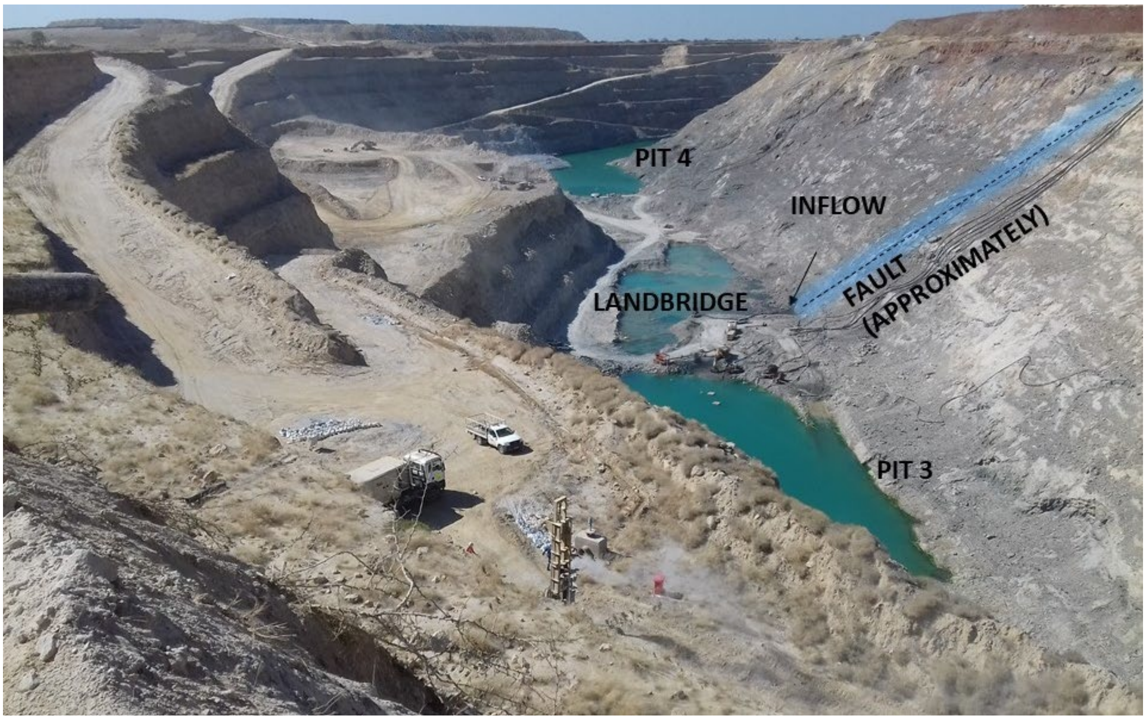


**FIG 6** – Pit after May 2018 flood (from van Wyk, 2018).

By 30 May 2018 water levels had stabilised and then started to decline. With the reduction of groundwater levels in the aquifer inflow rates gradually reduced. By November 2018 mine pit water levels were at preflood levels, however during this period an estimated 25 per cent of the pumped water was recirculating through the southern footwall back into the pit (Sarma, 2018).

After considerable effort, some 200 days after the flood event a suitable discharge point to the NW that was separated from the Huttenberg Formation dolomite by dykes was used, significantly reducing recirculation.

Ongoing production was not trouble free and between November 2018 and late 2019 the site experienced:

- Delays with construction of pontoon for pumps in late 2018.
- Pump failures (not an unusual occurrence) and issues with disposal of excess of water.
- The sinking of a pontoon and pump loss in late April 2019.
- Another flood event in June 2019.

Throughout this time, ongoing follow-up by mine management was required to ensure the re-establishment of sumps and efficient operation of pumps to not adversely affect mining operations.

## GEOTECHNICAL

After an instability was observed in the pit ramp in the northern hanging wall in November 2018 (Figure 7) a geotechnical review by Mining One (Stuklis, 2018) identified issues that included but were not limited to:

- On the north wall of the pit, numerous batter and multibatter structurally controlled failures had removed the catch berms above the ramp resulting in limited to no catch capacity, presenting potential exposure of personnel to rockfall hazards.
- The proposed cutback design had inter-ramp angles (IRA) between 57–62 degrees were too steep compared to the 55 degrees in fresh sandstones based on the current level of confidence in the geotechnical model, the current understanding of the structural model and the performance of existing slopes.

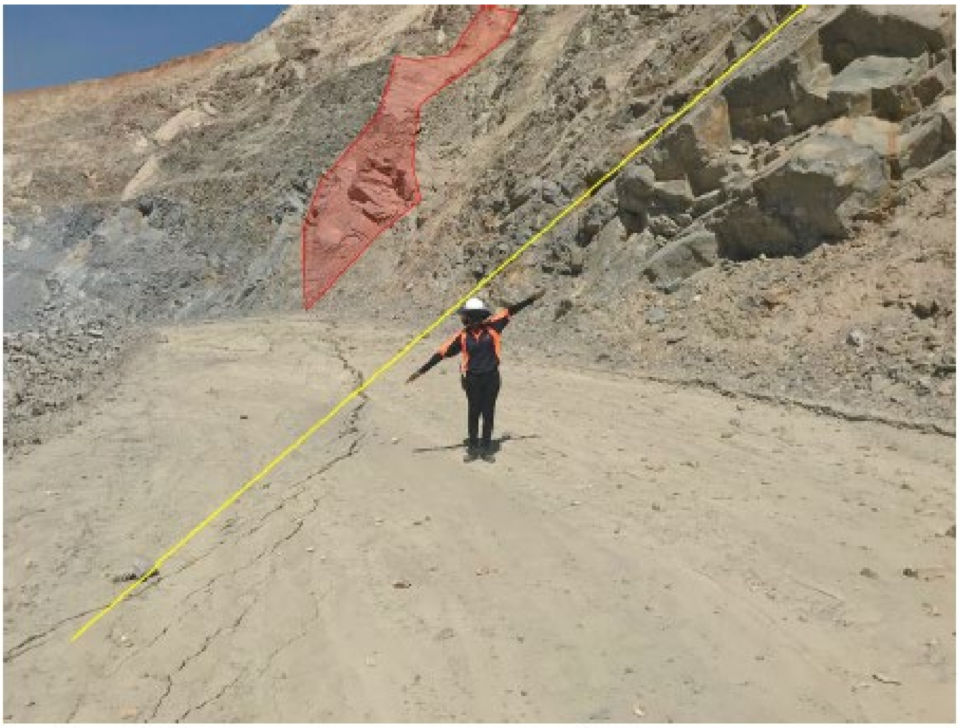

Figure 3-6 Pit 5A Access Ramp, location of distressed section. Orientation of J2 (yellow) demonstrated by Weatherly Geologist. J1 shown in red. Note tension cracks on access ramp prior to failure.

Figure 3-7 Cross-section line through distressed section of Pit5A access ramp. Projection of J2 length and dip (shown in green) below access ramp inferred.

**FIG 7** – Pit 5a Access Ramp instability (Stuklis, 2018).

Positive observations from the same review noted that:

- Recent (since October 2018) survey prism monitoring has been initiated for the north wall at the location of the unstable ramp section.
- Current lower pit slopes of the 5a north wall, below the ramp, show that more catch berms have been maintained and have not failed, reflective of less steep IRA and bench face angles than higher up.

Key recommended actions included:

- In absence of an update to the geotechnical and structural models. IRAs on the northern pit wall be limited to between 41 and 45 degrees.

- Conduct a more detailed review of all geotechnical data and provision of regular on-site geotechnical engineering support with additional support off-site.
- Establish or update the geotechnical and structural models.
- Installation of a broader prism monitoring network and establishment of routine monitoring (Figure 8).
- Structural domaining of rock fabric and major structures should be carried out. This will provide a clearer understanding of where any opportunities to steepen pit walls may exist and conversely identify wall areas where slope angles should be reduced.
- In order to achieve the slope design blasting practices should be reviewed and updated to best suit the operation and minimise rockfall and other types of wall instability.

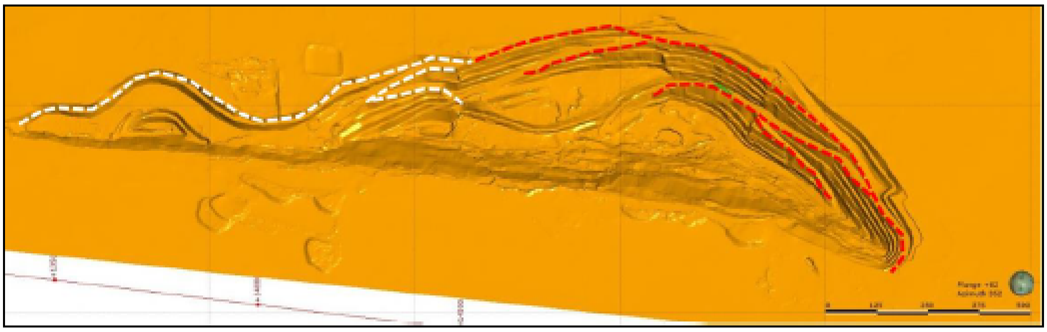

**Figure 3-11 Recommended locations for establishment of a primary prism network (PPN) shown with dashed lines – highest priority locations shown in red; lower priority in white. November 2018 as-mined pit shown.**

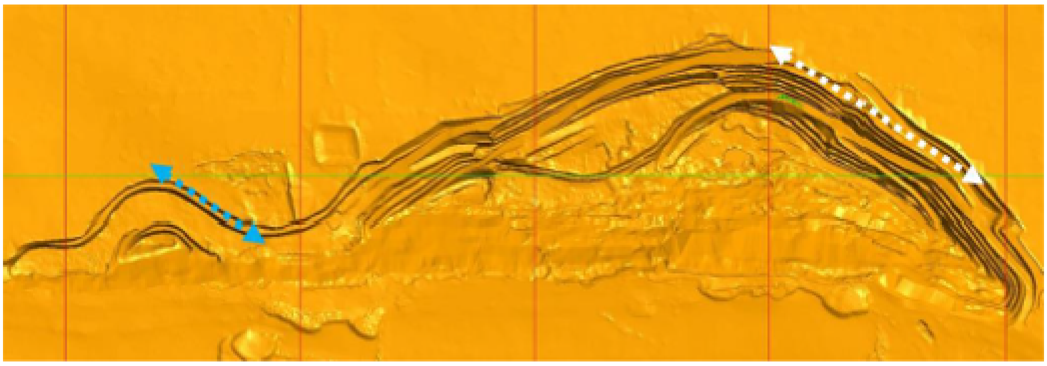

**Figure 3-12 Pit slope wall orientation(s) most adversely affected by rock fabric (structures) forming kinematically unstable blocks, based on current slope performance (white dash line: current wall sections subject to wedge and planar failure modes; blue dash line: section of pit wall with potential for kinematic instability in the future)**

**FIG 8** – Recommended primary prism network and adverse pit slope orientations (Stuklis, 2018).

Plans were then put into place to implement the recommendations and reduce risk to personnel and the operation.

## HEAP LEACHING

Recoveries for the Tschudi copper species were based on The Normalisation Method which allows estimation of copper mineralogical species in ore using different chemical analyses (Jo and Loyola, 2018a) by partial or sequential copper analysis. For cost and value, partial extraction was recommended, however as existing assays at Tschudi were based on sequential leach these were used as the basis for all future chemical analysis.

As mentioned earlier, other than for oxides, in 2018 the actual heap leach performance at Tschudi for each mineral species was lower than the expected (regular) and lower again compared to ideal recoveries. These are shown in Table 2.

**TABLE 2**

Ore type and expected leach recovery of Total Copper.

| Ore type | 2018 Average recovery (%) | Regular recovery (%) | Ideal recovery (%) |
|---|---|---|---|
| OX | 90 | 90 | 90 |
| CC | 70 | 79 | 89 |
| BN | 55 | 65 | 73 |
| CPY | 3 | 4 | 5 |

The underperformance of the heaps was suspected of being mineralogical, chemical, or due to the dynamics of the heap bed due to physical characteristics of the ore, aeration levels or saturation.

The typical number of days to achieve the maximum recovery for each mineral species are:

- OX: 350 days
- CC: 700 days
- BN and CPY: 700 days.

## Bacterial leaching

### *Column leach tests*

Before the site visit in late 2018, efforts were already underway to improve heap leach performance. Column leach tests to evaluate the bacterial oxidation capacity to convert ferrous ($Fe^{2+}$) to ferric ($Fe^{3+}$) in transition ores were already being commissioned. The aim to determine the best irrigation solution, environmental conditions for high bacterial growth and to evaluate the effects of different ore characteristics on oxidation capacity (Appolus *et al,* 2018).

### *Percolation tests*

Concerns over the physical characteristics of the heaped ores affecting aeration rates and production lead to the processing team to evaluate the effects of curing time on fines migration. To do this, after some research it was found that the evaluation of agglomerates using the Kappes percolation test (Pyper, Kappes and Albert, 2015) may be suitable. The test was applied to transition ore using 2 minutes of agglomeration at a discharge moisture of 8.5 per cent and an acid addition set point of 10 kg $H_2SO_4$/tonne ore to test curing times of 0 hrs, 24 hrs (in use by operations) and 48 hrs.

Results indicated that the 48-hr curing time had preferred characteristics of: the highest draining flow at close to 37 000 L/hr/$m^2$; lowest total suspended solids at 1.8 g/L and lowest total slump (the critical criterion as noted by Pyper, Kappes and Albert (2015) of 7.7 per cent versus a maximum slump target of 10 per cent. The results prompted a change in curing time from 24 hrs to 48 hrs (Appolus *et al,* 2019).

While this was a positive step in improving the performance of heap leach operations, it is the understanding of the author that the Kappes percolation test was applied only to transition ores and not oxide only, primary only or expected ore blends with oxide.

### *Heap aeration*

A trial to promote bacterial leaching by increasing heap oxygen levels of at least 10 per cent for periods of 150–200 days in 2017 was considered successful, and aeration fans capable of 1–1.5 kPa were purchased with aeration rings. Later, the installed fans and aeration rings were found to be producing oxygen levels less than the minimum 10 per cent in a majority of the heap panels under aeration and larger fans capable of producing 15 kPa were purchased. It is not known if longer-term oxygen levels in heap panels improved because of the increased fan size.

Additionally, at the time of the percolation tests, it is the understanding of the author that there was anecdotal evidence that high levels of saturation in the heap, may have been contributing to a lack of aeration and lower oxygen levels.

## COPPER PRODUCTION

Copper production of the mine versus forecast is shown in Figure 9. Copper production can be seen to be trending down after the flood event in mid-2018 which was reversed and then sustained.

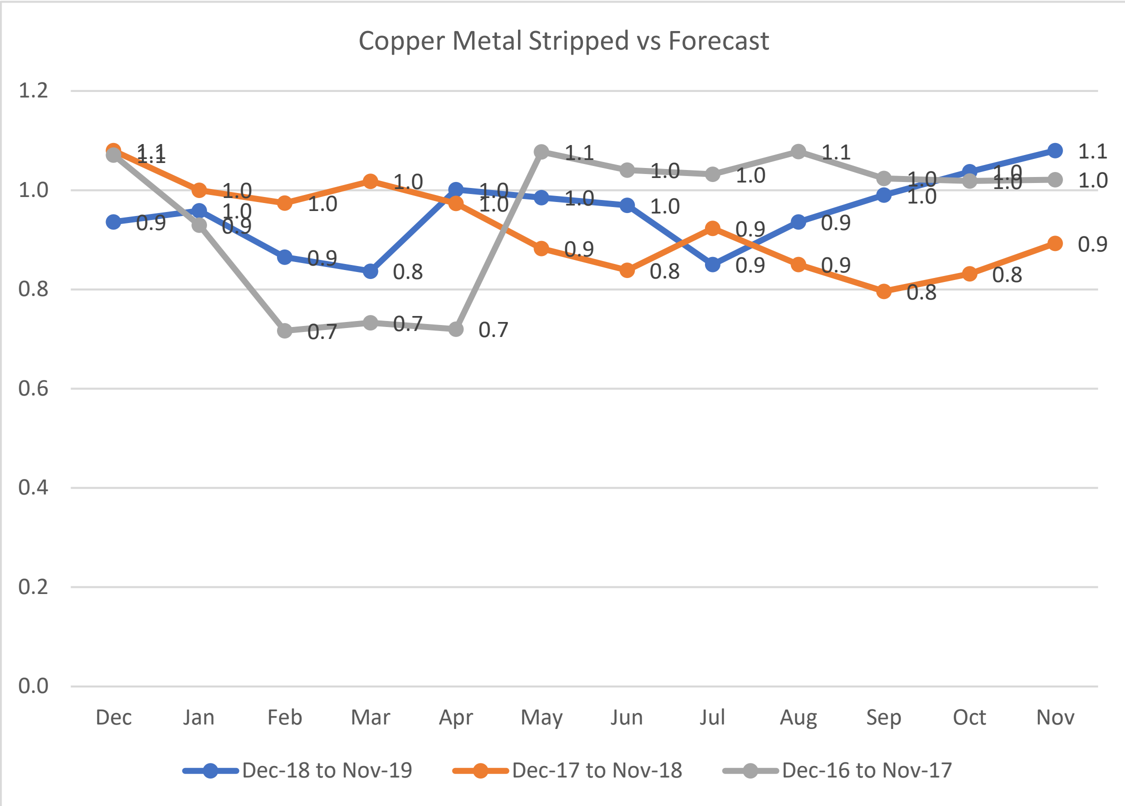


**FIG 9** – Copper metal stripped versus forecast.

## CONCLUSIONS

It is not easy to clearly demonstrate an improved performance, however it is the authors understanding that even in a falling copper price environment until early 2020, the mine went from an operationally loss-making venture to consistently profit-making from May 2019 to date.

In challenging conditions, the achievement of forecast mining material movements and stripped copper tonnes from Dec-18 to Nov-19 were almost identical to two years before (Dec-16 to Nov-17) when 98 per cent and 96 per cent of forecast material movement and stripped copper tonnes were achieved respectively.

To help minimise risks with the Tschudi project, further work is recommended including:

- Geotechnical data gathering including material characteristics, and defect strengths.
- Heap leaching investigations including:
  - agglomeration tests to optimise binder levels (moisture content, acid addition).
  - percolation tests on oxide only, primary only, and expected ore blends.
  - optimisation of the levels of saturation and aeration to enhance bacterial leaching.


## ACKNOWLEDGEMENTS

The author would like to thank the management and staff of Weatherly Mining Namibia, FTI Consulting, and Mining One for their contributions and support in writing this paper.

Approval for the submission of this paper has kindly been provided by Weatherly Mining Namibia.

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# Slope stability

# Strategic short-term geotechnical design

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## ABSTRACT

The principal objective of Geotechnical Engineers supporting operating mines is to provide safe working conditions and ensure that pit designs are geotechnically optimised to site conditions and pre-defined stability requirements. Pit slope designs are delivered in advance of pit excavations. Making changes to these designs is usually slow to enact and involves a layered sign-off before implementation, leading to unwanted delays in their adoption and subsequently increased production pressures. When unexpected ground conditions are encountered during the mining process, or short-term mining schedule requirements demand it, changes to these designs may not be easily feasible in certain situations. It is when these scenarios arise that Geotechnical Engineers need to be proactive and consider unconventional options to deliver safe operating conditions while satisfying short-term requirements to the mine schedule. Essentially, the geotechnical short-term strategic designs double-pronged approach supports the mine schedule while providing safe conditions and minimising geotechnical risks for people and equipment.

Common short-term strategic geotechnical designs include; slope buttressing, over-steepening cut slopes to increase reserve recovery through unplanned or emergency cutbacks, and accepting risk threat and operating under unstable slopes. When implementing strategic design options, the Geotechnical Engineer must consider the broader implications for such designs. They must think of the plan's long-term effects on the mine schedule and pit wall stability. Initiating aggressive designs to recover additional reserves to satisfy a short-term scheduling demand could result in-pit wall instability or suboptimal pit layout.

Before and during the implementation phase, the Geotechnical Engineer must communicate and engage with relevant stakeholders to ensure that the plan is executed accordingly and geotechnical risks are effectively managed. Proactiveness to ground response and implementation of risk management strategies will improve safety performance and optimise mining schedules.

## INTRODUCTION

The principal objective of Geotechnical Engineers supporting operating mines is to provide safe working conditions and ensure that pit designs are geotechnically optimised to site conditions and pre-defined stability requirements. With optimised slope configuration parameters that satisfy respective safety factor for the project stage, geotechnical designs are developed and approved as official mine plan design before the final design geometry is implemented in the field.

However, due to unforeseen factors or circumstances that can affect the mining production cycle, production needs result in a plan that does not adhere to the standard pit design protocols or, in some cases, a long-term strategic plan. In these instances, Geotechnical Engineers need to assess the individual factors and deliver advice that satisfies the following:

- facilitates the short-term production demands of the mine
- maintain a safe operational environment
- does not overly compromise the long-term strategic requirements of the mine plan.

Combining the above factors means that non-conventional designs that mitigate the developed situations are required. Design options not commonly available to the Geotechnical Engineer may be required. These designs often affect production times and deliverables, requiring various options to solve the problem in the safest and shortest period. The result is that the designs are bespoke, require a detailed understanding of the failure mode and mechanism and a dynamic approach to managing the risk as the slope is excavated. These non-standard designs are short-term strategic geotechnical designs.

## STRATEGIC SHORT-TERM GEOTECHNICAL DESIGN

Strategic short-term geotechnical design is a process of creating pit designs that satisfy a short-term strategic requirement. This requirement may be to solve:

- a latent geotechnical hazard that impacts pit slope stability
- a known geotechnical risk that creates a hazardous working environment for both people and equipment
- an opportunity to improve reserve recovery
- commodities market changes (positive or negative)
- satisfy short-term production needs
- maximise resource recovery for the final void or final 'good-bye' cuts when the pit reaches its final design limits
- improved understanding of rock mass conditions and design assumptions accommodates more aggressive design for the final benches.

As the problem to be addressed is a short-term issue, Geotechnical Engineers often provide bespoke short-term designs that deviate from the more conservative long-term design geometry. Strategic short-term designs are typically more aggressive and may not satisfy all design criteria guidelines adopted for mid- or long-term pit design. Therefore, the Geotechnical Engineer must ensure the working environment safety requirement to ensure the safety of people and the equipment of all workers while following a robust design process (Figure 1) all of the mine's needs are satisfied.

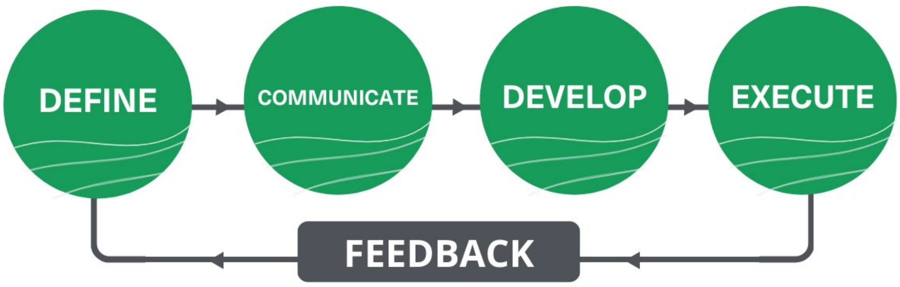


**FIG 1** – Design process for strategic short-term geotechnical design.

### Define

The Definition phase of the process is required to define the intent of design change, often resulting from an observed hazard in the sequence of mining or after a failure event that restricts recovery of reserve ore. The design change is often strategised around the imminent failure or post-failure event, leading to subsequent development of a new plan that circumvents the risk posed by the failure (post-event) or impending failure. The Definition phase flow chart is shown in Figure 2.

#### *Strategic requirement*

The strategic requirement for a short-term design establishes the battery limits and identifies the required outcomes. Without battery limits or the desired results, the Geotechnical Engineer will be unclear what options they have available to develop a design to satisfy the outcomes.

This step needs to:

- Define the purpose of the short-term design, eg stabilise an unstable slope, steepen a slope angle to allow ore recovery, develop a plan to recover ore below a known hazard, etc.
- Consider the long-term mine design or schedule. A short-term plan must not sacrifice the long-term pit schedule, eg if a wall needs to be buttressed, then the buttress must not cause access

to other ore to become unprofitable, or, if ore recovery is going to cause a wall failure, will that failure impact the long-term pit schedule?

- The key stakeholders must be identified early and their requirements considered inthe decision-making process. The requirement for the plan is to maintain sustainable orefeed or ensure Production and shipments are not delayed. The operators are concerned for their safety, while the Environmental department often ensure that the regulatory requirements are not breached. While some stakeholders may need to compromise on their requirements more than others, they must be identified and considered to influence the design outcome positively.


**FIG 2** – Define process definition.

### *Hazard identification*

Like all design assessments, the Geotechnical Engineer must identify all of the latent Geotechnical hazards to be managed in the design. In addition to the failure mechanism, operational experience, engineering judgement, and field-collected data are critical to ensure the optimal return. Critical information available to the Geotechnical Engineer that may be considered in hazard identification include:

- structure model and structural responses due to excavation
- failure mechanisms, ie wedge, planar, toppling, circular etc
- wall stand-up time
- expected failure size (volume)
- typical failure run-out distance
- deformation trends, ie displacement and velocity prior to failure
- groundwater conditions

- geology model.

Numerous influences can impact slope stability, and often, multiple loadings and failure modes contribute to the movement. Therefore, the Geotechnical Engineer must identify all of the relevant loadings and consider them in their assessment. This may require collecting additional field data to provide sufficient confidence in the model and subsequent design process.

### *Risk tolerance*

The prevailing circumstances will strongly influence the mining companies acceptable risk tolerance. A risk assessment should be completed to guide the decision-making process around risk tolerance levels.

According to the Australian Institute of Health and Safety (AIHS, 2019) (AIHS, 2019), there are four types of risk-related decisions, which are:

- Strategic decision
- Tactical decision
- Operational decision
- Contingency decision.

> *The risk will be influenced by how the risk is perceived by those making the decision. Risk perception is not fixed, but is constructed based on an individual experience and situational; characteristics.* (AIHS, 2019) (AIHS, 2019)

Other factors that may affect decision-making are:

- Corporate risk tolerance
- Site history with managing similar types of hazards, eg operator experience working under unstable walls
- Environmental requirements ie mining lease boundaries, environment, social and government (ESG) requirements, cultural heritage, infrastructure etc.

Once all the risk factors have been considered, design acceptance criteria (DAC) can be established that the design must satisfy. Read and Stacey (2009) provides some tables that outline recommended Factors of Safety (FoS) and corresponding Probability of Failure (PoF), shown in Table 1.

**TABLE 1**

Slope design criteria (Read and Stacey, 2009).

| Slope scale | Failure consequence | Acceptance criteria | | |
|---|---|---|---|---|
| | | **FoS (min) (static)** | **FoS (min) (dynamic)** | **PoF (max) [FoS≤1]** |
| Bench | Low–high | 1.1 | NA | 20–50% |
| Inter-ramp | Low | 1.15–1.2 | 1.0 | 25% |
| | Medium | 1.2 | 1.0 | 20% |
| | High | 1.2–1.3 | 1.1 | 10% |
| Overall | Low | 1.2–1.3 | 1.0 | 15–20% |
| | Medium | 1.3 | 1.05 | 5–10% |
| | High | 1.3–1.5 | 1.1 | ≤5% |

## Develop

Once a pit slope design has been implemented, the Geotechnical Engineer has limited options to develop a hazard mitigation design. The development of a suitable design requires three steps, shown in Figure 3.


**FIG 3** – Develop process definition.

### *Concept options*

The Geotechnical Engineer should engage and liaise with other departments, ie Geology and Mine Planning to ensure confidence in the design inputs and assumptions. Additionally, Production Superintendents should also be engaged to ensure that the design being developed by the Geotechnical Engineer is practical, and Production crews are comfortable executing it.

The Geotechnical Engineer should be flexible in their design to accommodate requirements from all departments while satsifying the DAC. This stage can lead to the development of solutions that are non-standard and dependent on the analysis method, they may rely on transient slope conditions that do not satisfy the design criteria or Factors of Safety.

This is the time when the Geotechnical Engineer can be bold in their design recommendations, assuming that necessary steps and engineering sound judgements have been considered to reach any particular course of recommendations. Where non-standard designs are recommended that do not satisfy the DAC, the Geotechnical Engineer must have site-based empirical data to support their design option. This information must ensure a detailed understanding of the historical ground behaviour and responses to different loading conditions. The critical thing to understand during the concept development stage is to consider all options, no matter how unrealistic they may initially present.

Some of the things that may be considered in the development of an appropriate design include:

- excavation sequence to manage loading conditions on the slope

- buttressing to provide passive support to the slope
- rolling buttress to progressively support the slope
- slope cut-back to fully or partially unload the potential failure
- slope de-stacking (unloading)
- slope geometry modification to modify the loading on the slope.

Where multiple options are presented, cost-benefit and sensitivity analyses may be helpful in quickly eliminating some possibilities. Where a negligible cost difference exists or numerous options are considered, these options should be risk ranked based on the assessed geotechnical risk of each option.

### *Design development*

Following the adoption of the preferred concept design, the Geotechnical Engineer must complete a detailed analysis. This stage can be somewhat heuristic when the select option does not satisfy the design criteria established during the 'Define' phase. Engagement with the interested departmental stakeholders is required to modify and optimise the design to develop an outcome suitable for execution.

It is during the design development that the Geotechnical Engineer can be aggressive in their design. A transient FoS of <1.0 may be 'acceptable' in this situation. A detailed understanding of the failure mechanism and failure mechanics is required, eg site experience with slope performance and deformation rates in response to excavation. The site should retain a failure database that includes, where possible, historical deformation rates leading up to and immediately before failure.

The geotechnical engineer should risk rank the remaining options for approval by the appropriate signatory where multiple options remain.

### *Ground monitoring*

A critical aspect of any short-term design is the adoption of a monitoring program. A program should be developed that is bespoke to the hazards and failure mechanisms considered.

When developing a system to monitor operating under an unstable slope, the slope stability radar is the most reliable tactical monitoring tool available to the geotechnical practitioner. The 'quasi-real-time' monitoring system allows for the development of monitoring programs that permit equipment to operate safely under deforming and destabilising slopes.

Where equipment is to operate and unstable slopes, a strategic monitoring plan must be developed. The plan's purpose is to:

1. Ensure the safety of equipment operating in the area.
2. Reduce the downtime of operations when alarms trigger that requires the danger zone (line-of-fire) to be evacuated.

## Communicate

Communication of the plan to all key stakeholders is required to ensure everyone is on board. As Geotechnical Engineers are rarely the people operating directly in the line of fire, it is vital to ensure the plan and justification for the plan is clearly communicated and all concerns addressed. It is crucial to have all personnel buy into the plan and understand the importance of all their observations during mining.

## Execute

Upon acceptance of the design by all stakeholders, the strategy and plan need to be executed. Mining is adept at implementing the Observational Method. Due to the natural heterogeneity and variability of rock masses, a geotechnical practitioner cannot develop a model that is 100 per cent accurate due to the inherent uncertainty associated with respective input parameters. Therefore, the Observational Method is employed to manage the risk during execution. Eurocode 7 (EN 1997–

1:2004; CEN, 2004) outlines the approach to be taken and the parameters that must be developed during the 'Development' phase:

1. *When prediction of geotechnical behaviour is difficult, it can be appropriate to apply the approach known as 'the observational method', in which the design is reviewed during construction.*
2. *The following requirements shall be met before construction is started:*
    - *acceptable limits of behaviour shall be established;*
    - *the range of possible behaviour shall be assessed, and it shall be shown that there is an acceptable probability that the actual behaviour will be within the acceptable limits;*
    - *a plan of monitoring shall be devised, which will reveal whether the actual behaviour lies within the acceptable limits. The monitoring shall make this clear at a sufficiently early stage and with sufficiently short intervals to allow contingency actions to be undertaken successfully;*
    - *the response time of the instruments and the procedures for analysing the results shall be sufficiently rapid in relation to the possible evolution of the system;*
    - *a plan of contingency actions shall be devised, which may be adopted if the monitoring reveals behaviour outside acceptable limits.*
3. *During construction, the monitoring shall be carried out as planned.*
4. *The results of the monitoring shall be assessed at appropriate stages, and the planned contingency actions shall be put into operation if the limits of behaviour are exceeded.*
5. *Monitoring equipment shall either be replaced or extended if it fails to supply reliable data of appropriate type or in sufficient quantity.* (CEN, 2004)

### Feedback

Like all good design processes, the design's performance should be recorded and measured for future reference. Where required, this information should be fed back into the next design to optimise its performance and safety.

Where possible, data that should be recorded for future reference include:

- deformation rates
- slope (geological structure) behaviour
- size of the instability
- run-out distance
- failure volume.

## CASE STUDIES

### Site A – slope buttressing

#### *Site background*

Site A is located in south-east Queensland, and the mine has been in operation for over 20 years. The Site has been through multiple ownerships that have considered different design approaches and several iterations of slope angles. The Site is a single-pit source, utilising conventional drill and blast and load and haul mining methods.

Slope geometry was varied through different ownerships from 5 m bench heights and 5 m berm width to the current configuration 30 m bench height and 15 m berm width. The current bench height is mined through a double bench system, where two blasting episodes are conducted on each 15 m interval (mid-bench) for each production block sequentially. These blasted blocks are then split into two mining flitches each 7.5 m to suit the Site's mining equipment.

The open pit is situated within a complex geological and structural suite with variable rock mass quality. These complex structural networks within the deposit are associated with multiple complex slope failure modes with wedge and planar failures dominating the geotechnical domains (CEN, 2004).

## *Site geology and hydrogeology*

The host rock for Site A comprises mostly dacitic rocks within a clastic rocks package with a robust volcanic association. Thesequence of the volcanic rocks varies entirely from sedimentary to magmatic. Literature review of the Site shows that the host rocks are clast supported volcaniclastic rock probably derived from an eruption and deposited proximal to the vent. The Site has undergone various alteration episodes with a dominant sericite alteration on the open pit's Western slope. This alteration has been associated with the emplacement of the mineralised dacitic dome.

The alteration compounded by the complex structural networks, which are primarily low-angled (30–45°), has been a significant instigator in the ground challenges experienced on the West wall of Site A.

The groundwater at Site A is mostly compartmentalised due to the fractured rock system and low matrix permeability. Aquifer storage and hydraulic diffusivity are also low, implying that the groundwater in the current pit shell's vicinity is exceptionally high and exhibits a steep cone of depression.

### 1. Problem definition

Site A has had ground challenges and slope management challenges throughout its life. These challenges include rockfalls, bench-scale to multiple bench slope failures. Geotechnical risks have been managed categorically depending on the location of failure on the pit slope and pit activities at the time of failure. Three failures in the last three years revolutionised the geotechnical approach towards slope management as the three failures had a significant impact on the mine plan and schedule.

Ground challenges are primarily a result of the deposit's complex geology, the pit's geometry relative to the inherent structural networks and operations issues. A brief description of how a slope failure was managing through the implementation of effective short-term planning is discussed below.

*West Wall Failure*

A failure was picked by the deployed two ground-based interferometric synthetic aperture radars (GB-InSAR). The block displacement was controlled by a steep basal plane and two subvertical planes, shown in Figures 4 and 5 respectively.

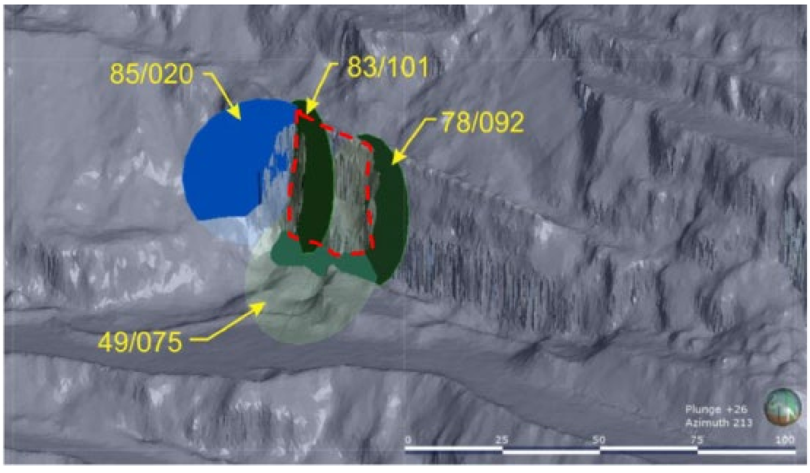


**FIG 4** – 3D model showing the as-built pit shell with the mapped joint planes defining the block (red dotted line).

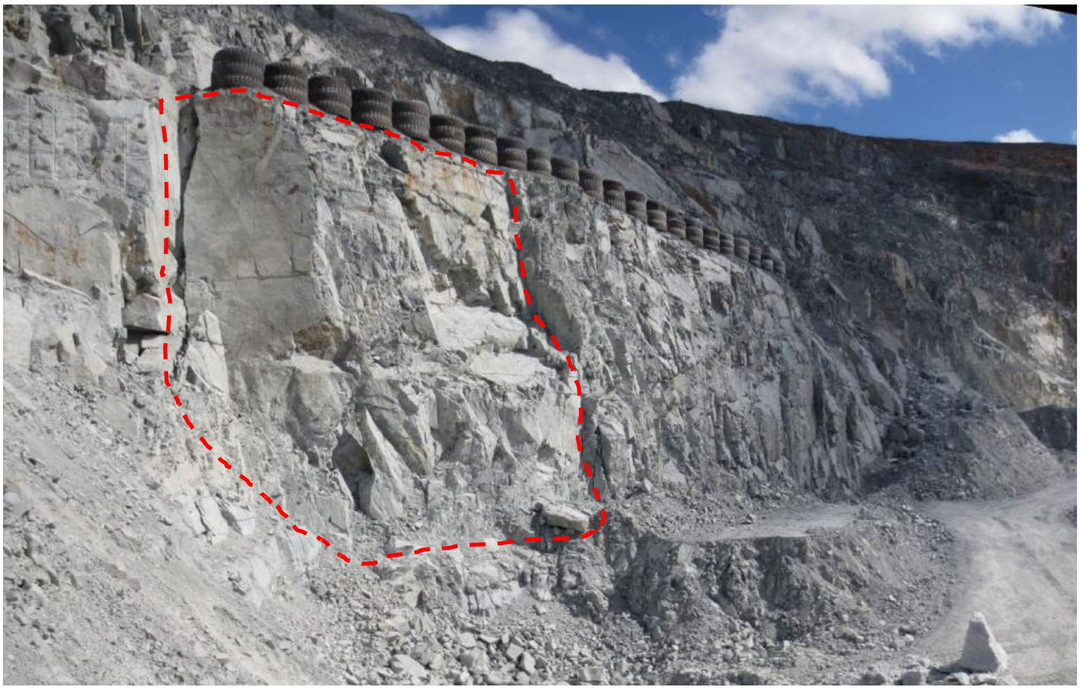

**FIG 5** – Photo showing the block and the controlling structures.

The failure occurred during the final pit shell excavation, and it required an immediate change in short-term mining plans to manage the failure and re-direct Production to other areas to continue production cycles.

**2. Development of strategy (short-term plan inception)**

Some strategies that were adopted to manage the geotechnical risks included:

- detailed mapping of the structural networks
- implementation of trigger action response plans (TARPS) when working proximal to the pit highwalls
- slope monitoring through radars – with two radars installed for full-pit monitoring.
- installation of robust instrumentation – time-domain reflectometers (TDRs), nested piezometers
- remote drilling in high-risk areas
- an integrated risk management framework that incorporates all the strategies.

Monitoring radar information recorded that the inception of the failure was when the last flitch's excavation was begun, shown in Figure 6.

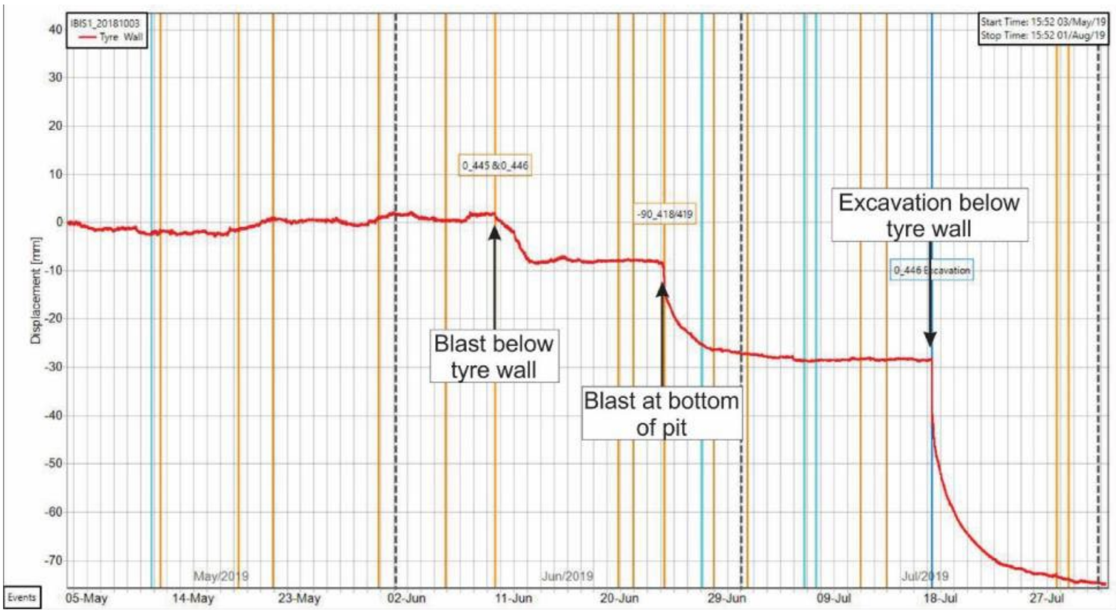


**FIG 6** – Radar monitoring cumulative displacement map.

Short-term plans had to be changed to meet ore deliverables for the plant. The initial plan was to re-design the interim pit shell at the pit bottom and open up benches that would accommodate the production target until remediation of the failure area was completed, which would allow mining to continue on the final pit shell design.

Failure remediation was done by buttressing the failure block by waste rock mined from the pit bottom and the South cutback. Buttressing was done by a remote-operated dozer with dump trucks dumping short on the tip end and the dozer pushing the material to wrap around the failure area.

### 3. Communication of plan

All mining crews were notified of the change in plan and mining sequences. Four sessions were conducted through meetings to cover all four working shifts for both day and night shifts. A presentation was conducted and disseminated to the whole Site alerting all mine personnel of the hazard and the changes to the mine plan.

### 4. Execution of plan

The short plan to manage the failure was executed as per the plan with a remote dozer involved with the emplacement of the waste rock at the toe of the failure to stabilise the failure and built up the ramp to 0 mRL. This process was conducted through haul trucks short-dumping waste material from the bottom of the pit and the remote dozer pushing the material against the wall to mobilise a buttressing effect onto the creeping block.

### 5. Feedback

Feedback on the process was done during the implementation process and post-implementation. The feedback process allowed all the people to be updated on the progress of the remediation process and track the progress of the remediation process against the scheduled time for the remedial works. Considering that this process was critical in re-establishing the long-term mine and getting back on track to open up the pit to the final design wall to access the bottom ore, higher ore grades were anticipated.

## Site B – vertical pit walls

### *Site background*

Site B is located in the New South Wales coalfields of eastern Australia and has been operating for 15 years. The mine employs traditional strip mining and truck-excavator methods. The mine currently produces about 14 Mt of ROM coal per eight from eight pits, covering approximately 2800 ha. The mine limits are constrained by infrastructure and Sandstone escarpments that increase the strip ratio, making mining uneconomic.

### *Site geology and slope geometry*

Slope geometries comprise 5 m to 10 m of alluvium and weathered overburden but excavated at 45°. The underlying fresh coal measure rocks comprise a 30 m thick sequence of interbedded coal seams and fine-grained sedimentary rocks. Maximum slope heights are about 80 m.

The overlying fine-grained sediments are mined as a single pass with 70° pre-split highwalls. Coal passes are mined with 65 m multi-flitch passes with secondary coal parting blasts where hard waste interbeds are encountered.

The geotechnical conditions at the Site are generally relatively benign, with an occasional small wedge or planar failures occurring. Critical geological structure encountered at the Site were persistent jointing and occasional faults.

Groundwater is encountered on-site but does not typically create stability issues within the fine-grained sedimentary units.

#### 1. Problem definition

Due to the pit geometries, the mine regularly mines final walls due to topographic and mine boundary restrictions. The Mining Engineers posed the question to the Geotechnical Department if mining the final highwalls vertically (90°) was possible.

Due to the rapid development rate of the mine, the final highwalls only remain open temporarily for about three months. Backfilling of all pits typically occurs within three months of the completion of coal mining. Hence, the vertical walls will only be left unsupported for a relatively short time frame.

Coal mining blocks are 100 m long by 60 m wide. Therefore, the potential benefit of the option would allow for an additional 100 000 t of ROM coal per block. This equates to about an extra $10M of coal sales per block based on current coal prices. With each pit having between 50 and 70 strips, the potential increased revenue was significant for the mine.

#### 2. Development

Due to the benign geotechnical conditions and well-understood failure mechanisms, the option was progressed to assess its validity. In addition, data requirements and analysis methodologies were outlined to create comfort in the proposed alternative.

As the Site is known to have vertically and laterally persistent joints that subparallel the highwalls in some areas, photogrammetric data for the previous highwall or nearby pit walls was requested to define the presence and orientation of joints. Photogrammetric data for the initial pit wall assessment identified multiple low-risk toppling hazards (Figure 7). As seen in Figure 7, the persistence of the mapped joints is very short, <2 m. Kinematic analysis of the data identified a low risk of toppling failure for the mapped structures (see Figure 8).

A finite element analysis was undertaken for stability assessment to calculate the shear strength reduction factor (SRF) and expected slope behaviour during excavation. The results returned an SRF value of >1.5, and no tensile fracturing or internal shearing was expected.

The results of the stability assessment returned favourable results, and the option was approved.

As ground monitoring instrumentation such as slope stability radar had not previously been used at the site, no ground monitoring instrumentation was employed. Visual monitoring was the only ground monitoring technique adopted.

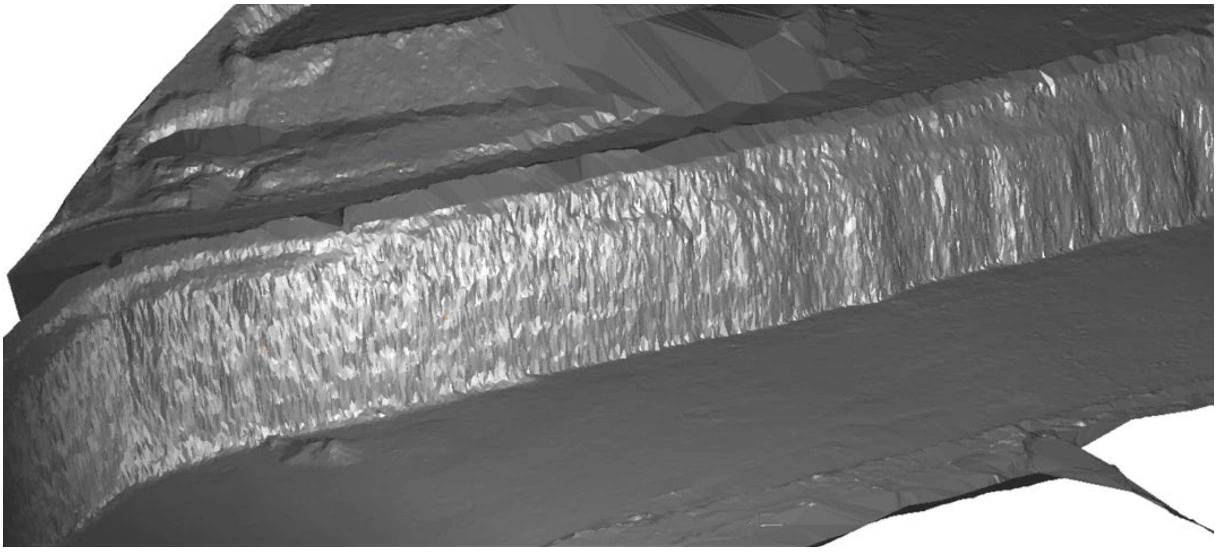

**FIG 7** – Photogrammetric data for pit wall mapping.

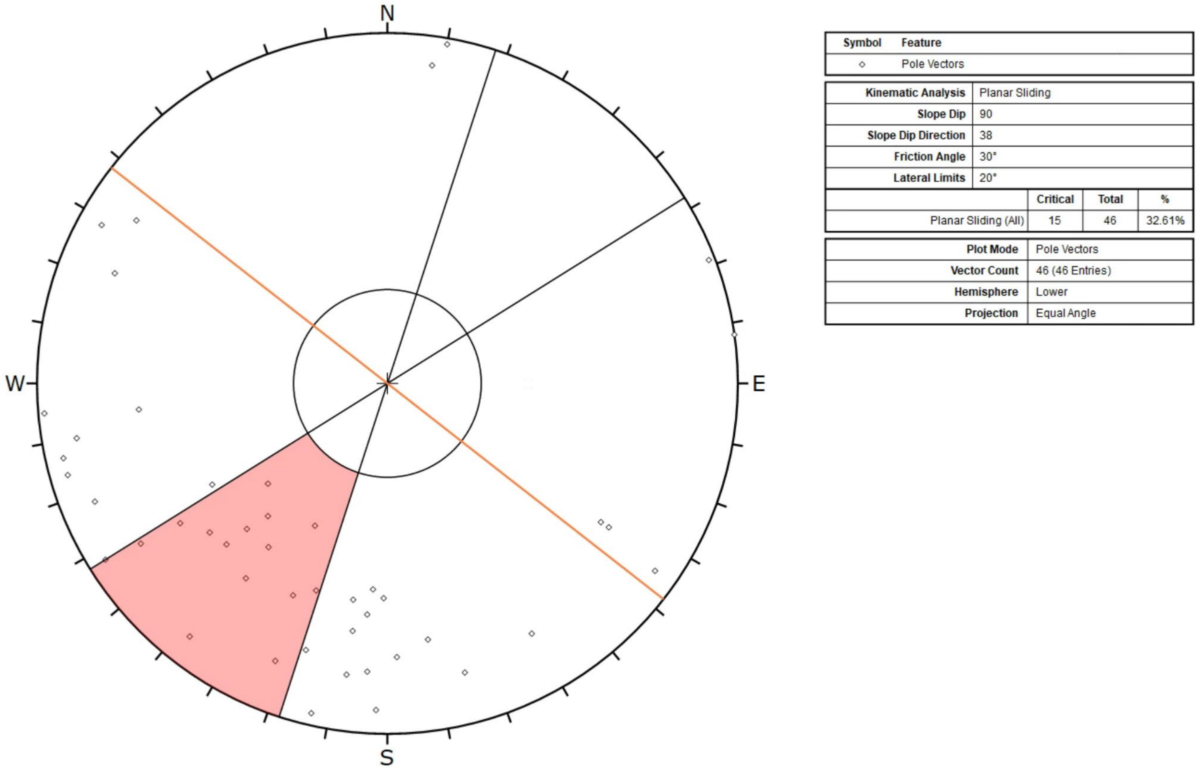


**FIG 8** – Kinematic analysis for the planar failure of mapped defects.

Rockfall analysis confirmed assumptions that the rockfall run-distance reduced, improving the safety conditions for operators when working on the ground.

### 3. Communication

Following approval by the Technical Services department, a Risk Assessment was convened with a diverse cross-section of workers. After completing the Risk Assessment, the plan was rolled out to all mining crews at their pre-start and handovers. The Mining Technical Services department attended these meetings to respond to questions and concerns from the remaining operators.

### 4. Execution

The design was executed using Site standard mining practicesS. The 90° wall was pre-split and mined using a normal mining sequence. No incidents or issues were reported during mining.

The only deviation from design was some minor coal loss at the toe of the highwall. The coal could not be recovered because the excavator's digging arc prevented the bucket from reaching the highwall toe.

### 5. Feedback

Due to the success of the initial vertical wall, the methodology was rolled out across the Site. The data collection and analysis methodology adopted for the first section was followed for other areas.

## CONCLUSIONS

From time to time, a mine requires a strategic design to satisfy short-term requirements. These requirements permit the development of non-conventional designs that may not satisfy the traditional slope DAC. A staged approach engaging all key stakeholders can yield geotechnical designs and mine plans that provide safe operating conditions and significant value for a Site. Geotechnical practitioners should be open-minded in their approach and use site-based empirical data to support the execution of such designs.

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# Should earthquake loading be considered for open pit slope design in Australia?

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## ABSTRACT

The impact of seismic loading on open pit slope stability in Australia is often only considered where the slopes are located adjacent to tailings facilities or other major infrastructure. The debate regarding application of seismic loading in open pit design in Australia is usually founded on limited reporting on significant damage of open pit operations during earthquakes, the common perception that earthquakes do not occur in Australia or that they are unlikely to occur during the mine life of the mine. It is frequently overlooked that pit slopes that have withstood strong earthquakes in highly active seismic regions are often designed under strict seismic guidelines.

This paper discusses the factors that impact earthquake performance of open mine slopes. These factors include topography, slope geometry, rock material and mass properties, water management and intrinsic pit design. Further, the paper presents insights into the characteristics of earthquake activity in Australia such as magnitude distribution, earthquake depths, areas prone of earthquake activity and general seismic hazard that are relevant to pit design.

Guidelines are provided on the conditions and circumstances under which earthquake design in open pit mining should be considered. In addition, this paper provides recommendation on the applicable methods for assessment of the impacts of earthquake loading on pit slope stability.

## INTRODUCTION

There is considerable debate in Australia regarding the need of seismic stability analyses for design of open pit slopes. The reasons for the debate are usually based on the following:

- limited reporting of pit slope instability or significant damage following strong earthquakes
- the common perception that earthquakes do not occur in Australia or earthquakes will not occur during the life of the mine
- management of risk and risk acceptance by assessing failure as having low likelihood and/or consequence.

This paper provides an overview of the reported impact of seismic events on mining operations, particularly the performance of open pits and the factors that may control performance under earthquake loading. The regional seismic hazard of Australia, the characteristics of the earthquake activity and behaviour of seismogenic faults are discussed to provide recommendations on the assessment and inclusion of earthquake loads in open pit mining.

Tailings dams have special considerations that are outside the scope of this paper and are not included in this discussion. There is specialised literature dealing specifically with aspects of seismic design of tailings dams.

## IMPACT OF SEISMIC EVENTS ON MINING OPERATIONS

Read and Stacey (2009) rightly state that there are few, if any, recorded instances in which earthquakes have resulted in significant instability of hard rock pit slopes. Reported damage after strong earthquakes indicate that in contrast to natural slopes, designed slopes such as pit slopes are less susceptible to induced earthquake effects.

Read and Stacey (2009) refer to evidence from a number of mines in highly active seismic zones in Papua New Guinea, Chile and Peru, which although exposed to significant earthquake activity have not reported large scale pit slope failure.

There is limited documentation of earthquake induced failures of open pit slopes. Excluding tailings dams, reported impacts are typically:

- rockfalls
- pit slope failures of limited scale
- damage to mining infrastructure (roads, buildings and other equipment) due to failure of natural slopes.

The consequences of the above impacts to mining operations may include:

- loss of life or injury to personnel
- interruption to operations through a loss of road access, which prevents mobilisation of product and/or transit of personnel interrupting operations
- environmental damage through damage to infrastructure carrying concentrate or mining waste
- impact to operations due to loss of power.

Some examples of impacts associated with earthquakes include:

- In Chile, a magnitude 8.8 earthquake on 27 February 2010 resulted in infrastructure damage and temporary closure of 20 per cent of the mining operations in the region (El Mundo, 08/03/2010). Then in 2015, a magnitude 8.3 earthquake resulted in infrastructure damage and temporary closure of 37 per cent of the mining operations in the region (Nueva Mineria y Energia, 08/03/2016). However, further information of the type of damage was not reported.
- In a study of earthquake disruption of copper mines in South America, Schenebele *et al* (2019) listed the impacts of strong earthquakes on mining operations. The impacts vary from no damage reported, to interruption, closures and mine collapses. Most of mining interruption was associated with road closures, lack of electricity and structural damage.
- On 26 February 2018 a magnitude 7.5 earthquake occurred in the Hela province of Papua New Guinea. A total of 160 people were reported to have been killed and many others badly injured by the event. Impacts of the earthquake to mining operations included:
    - Porgera Gold Mine – suffered damage to its gas and electricity infrastructure (https://postcourier.com.pg/quake-rocks-shela-shp/)
    - Ok Tedi Mine – Large scale failure of natural slopes damaged the road and pipelines connecting the mine to the nearby Tabubil (Figure 1). (https://postcourier.com.pg/landslip-blocks-mines-access-road/)
    - ExxonMobil suffered damage to buildings and infrastructure at the Hides gas field with the facility not operational for eight weeks.

**FIG 1** – 26 Feb 2018 Earthquake in PNG resulting in failure of natural slopes, blocking mine access road: (left) aerial view; and (right) debris pile at access road.

## FACTORS INFLUENCING OPEN PIT SLOPE PERFORMANCE UNDER SEISMIC LOADING

There are numerous factors that may influence open pit slope performance under earthquake shaking. These factors include topography, geometry of the slope, water conditions, rock mass and intact rock properties. Table 1 summarises these factors, their impact and consequence for open pit slope stability.

**TABLE 1**

Factors affecting pit slope performance under earthquake load.

| Factor | Impact on pit slope | Consequence |
|---|---|---|
| Topography | The topography surrounding the pit may amplify or reduce earthquake ground motion. Most studies suggest that topographic amplification greatly increases the earthquake acceleration at the surface of the slope and most significantly at the crest (Ashford *et al*, 1997). Toh (2010) argues that the effect of man-made alterations to topography (ie large open pits) amplifies ground motion at the crest of the pit slopes and ridges. | Topographic effects likely amplify earthquake ground motion and may be detrimental if mine infrastructure is located at or close to the pit crest. |
| Slope geometry | The geometry of the slope (convex or concave slope) has been reported to have a significant impact on amplifying the ground shaking of hills and slope ridges. A focusing and defocusing of seismic waves occurs with curved terrain (Anggraeni, 2010). Seismic waves may be trapped on convex slopes (natural slope shapes) causing amplification. In contrast, seismic waves are scattered in valley-shaped features causing damping of ground shaking. Azhari (2016) and Azhari and Ozbay (2017) list the modification of the natural terrain curvature for open pit mining as a possible factor in improving seismic performance. | Pit slope geometry may scatter the seismic waves reducing the impact of the earthquake shaking. |
| Rock material and mass properties | Rock stiffness and stiffness contrasts play a role in earthquake amplification.<br>Amplification of seismic waves occurs when waves travel from hard rock to soft soil. Pit slopes with a significant stiffness contrast (eg slopes with topsoil layer, weathered material, contrasting lithologies) may experience increases in earthquake shaking and be at more risk of instability. In contrast, open pits in entirely hard rock are not affected by this phenomenon. | Slopes with a large stiffness contrast and/or in weak rock experience earthquake amplification. |
| Water | Increase in pore pressure during earthquakes is a key factor in earthquake associated liquefaction of soils and other phenomena. Similarly, in rocks an increase of pore pressure may increase the probability of failure on a pre-existent failure surface or specific failure mechanism. Azhari and Ozbay (2017) note that although water plays a significant role in different failure modes in rock slopes such as rock avalanches, rock slumps and toppling, with pore pressure control, surface water control and/or dewatering in mining likely to make open pits less prone to failure. | Water management (pore pressure control, dewatering, surface water control) may make pit slopes less prone to structurally controlled failure under earthquake loads by preventing increases in pore pressure. |
| Open pit design | Typically, open pits are designed to have static factors of safety of around 1.2 or greater which may accommodate some level earthquake loading. | Factors of safety for static design may allow some level of earthquake shaking to be accommodated by the slope. |

In highly seismic regions, pit slope design is commonly developed considering seismic loading.

## MINING DESIGN IN HIGHLY SEISMIC ZONES

Minimum damage and/or uninterrupted operation reported after strong earthquakes in mine sites in highly seismically active regions such as Chile, Peru and Mexico are usually quoted as evidence of low risk of slope damage under earthquake shaking.

There is very limited available information and details of mine damage or slope failure during earthquakes. Although damage is seldomly reported, catastrophic failure has not been documented. However, and most importantly, it is usually overlooked that mining operations in highly active seismic regions such as Mexico, Peru and Chile are highly regulated and designed under strict seismic guidelines.

Review of national media accounts regarding mining status after recent strong earthquakes in Chile, Mexico and Peru made the earthquake regulations for mining design apparent.

For example, a quote translated from Spanish in the *Outlet Minero* (a Chilean mining bulleting, dated 29/10/2017) referring to the lack of significant damage after strong earthquakes in mines in Chile states*:*

> *'The mining infrastructure is regulated by a stringent seismic regulation. Additionally, mining companies do not wish to lose production…in countries not used to seismic activity, without the necessary regulations, the consequences may be very different'.*

This quote is particularly relevant for Australia.

Open pit mines in countries such as Chile, Peru and Mexico are required to be designed with appropriate earthquake loads that depend on their site-specific seismic hazard. The pit slopes are designed considering earthquake loads from the national building codes or from mine site-specific seismic hazard assessments (ie Departamento de Seguridad Minera, 2010; Oldecop and Perucca, 2012; Chamorro, 2019). Additionally, these mining operations have detailed safety plans to follow during and after strong earthquake activity.

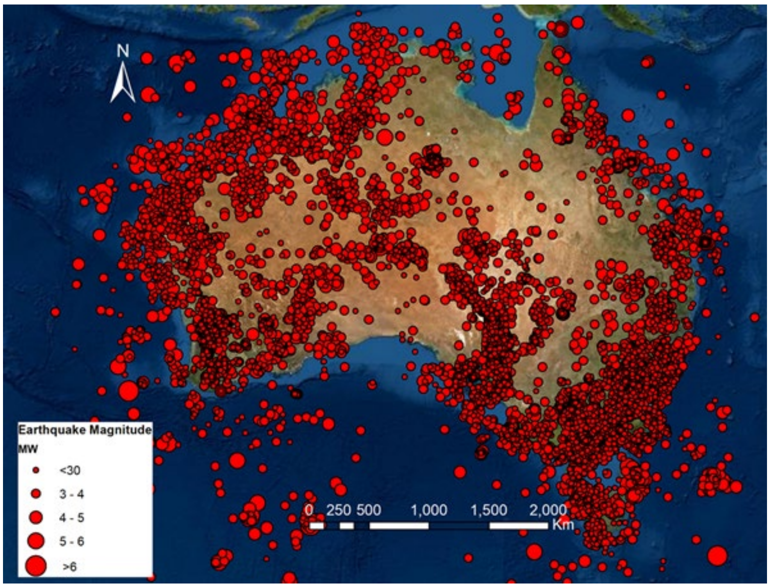
Consequently, the adequate performance of open pit mines in highly seismic regions support the inclusion of earthquake loads for design instead of justifying the contrary.

## EARTHQUAKE ACTIVITY IN AUSTRALIA

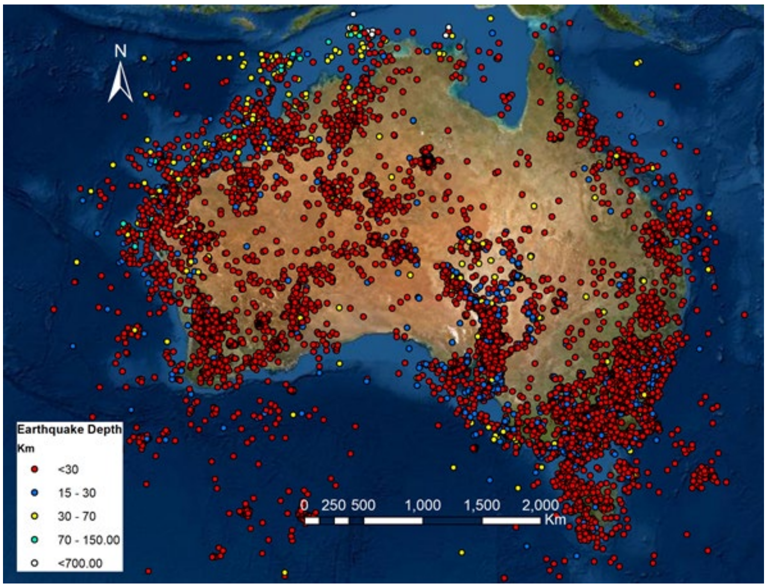
Australia is an intraplate region located away from plate tectonic boundaries where most of the world's earthquake activity is concentrated. Consequently, Australia has much lower earthquake activity and seismic hazard than Mexico, Peru, Chile or other highly seismic regions such as PNG. Despite its intraplate location, earthquakes in Australia occur relatively frequently but their occurrence and characteristics are highly variable across the continent.

Figure 2 shows the location of the earthquake activity recorded in Australia from 1840 to 2018 (earthquakes extracted from the earthquake catalogue of Geoscience Australia). Most of the earthquake activity has been recorded with shallow depths (less than 15 km depth). However, the earthquake locations may involve significant error due to the sparsity of the Australian seismic network.

On average 100 earthquakes of magnitude 3.0 or more occur in Australia every year. Earthquakes above magnitude 5.0 occur in average one to two years and earthquakes of magnitude 6.0 or more occur every ten years, approximately (Leonard, 2008; Geoscience Australia).




(a)




(b)

**FIG 2** – Earthquake activity in Australia. (a) Distribution of earthquake magnitudes. (b) Depth of earthquake activity. Earthquake records from the earthquake catalogue of Geoscience Australia.

Earthquakes can occur anywhere within Australia but there are four regions where current earthquakes are more likely: north-west of Australia, south-west of Australia, Flinders Ranges and south-east Australia (Figure 2).

Australian earthquake records show the seismicity has been steady in the last 100 years in the south-east corner of Australia, the Flinders Ranges and north-west Australia. In the south-west of Western Australia, the seismicity has dramatically increased since 1940 (Leonard, 2008; Estrada, 2013).

South-east Australia and the Flinders Ranges have the highest recorded earthquake activity. However, the largest earthquakes recorded in the last 50 years have occurred elsewhere (Table 2). Note that the Newcastle earthquake with magnitude 5.6 on 28 December 1989 which resulted in the deaths of 13 people is not included in Table 2.

**TABLE 2**

Largest recent earthquakes in Australia.

| Location | Earthquake magnitude Mw (post Geoscience Australia revision in 2016) | Year |
|---|---|---|
| North of Rawson, Victoria | 5.9 | 2021 |
| Off Kimberly Coast | 6.6 | 2020 |
| Tennant Creek, NT | 6.6 | 1988 |
| Meckering, WA | 6.5 | 1968 |
| Simpson Desert, NT | 6.4 | 1941 |
| Tennant Creek, NT | 6.3 | 1988 |
| Meeberrie, WA | 6.3 | 1941 |
| Collier Bay, WA. | 6.2 | 1997 |
| Tennant Creek, NT | 6.2 | 1988 |
| Cadoux, WA | 6.1 | 1979 |
| Peterman, NT | 6.1 | 2016 |
| West of Lake Mackay, WA | 6.0 | 1970 |

## Fault behaviour in Australia

It is well documented that earthquakes occur in geological faults. However, in Australia the association between earthquake and hosting faults is not always evident due to:

- limited knowledge on the character of seismogenic faults
- errors on earthquake location, which prevent the association between earthquake and causative fault
- lack of evidence of fault rupture in the surface after a strong event.

Despite the difficulties associating earthquakes with hosting faults, in the last 50 years, 11 earthquakes have been associated with surface rupturing faults in Australia (Clark, McPherson and Collins, 2011; King, Quigley and Clark, 2019). Figure 3 shows an example of a surface rupturing earthquake: the fault scarp associated with the Meckering Earthquake in 1968. A comprehensive description of the characteristics of historic earthquake surface rupturing features in Australia is provided in King, Quigley and Clark (2019).

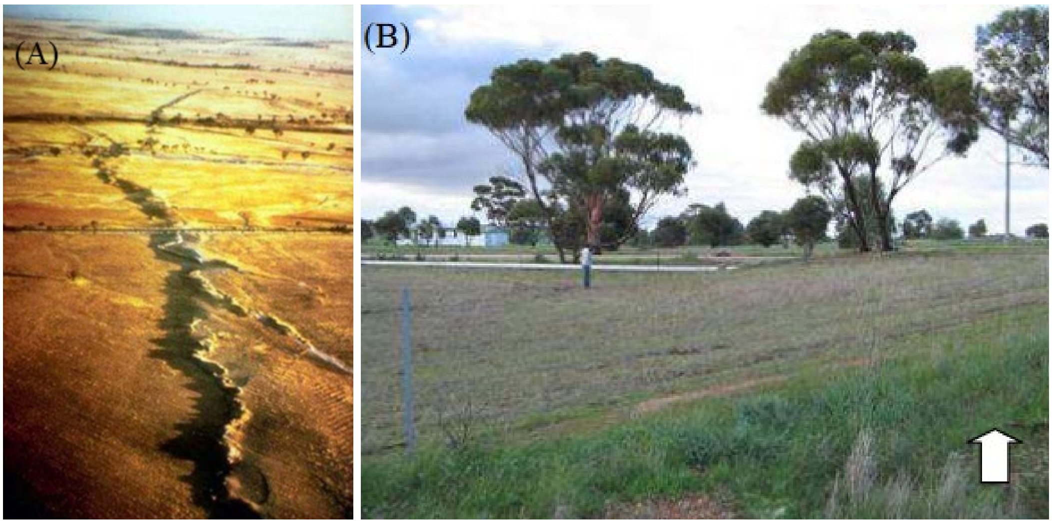



**FIG 3** – Fault scarp associated with the Meckering earthquake in WA. (a) Fault scarp as observed after the earthquake in 1968. (b) The scarp in 2013 (from Estrada, 2013).

Analysis of the earthquake associated surface rupturing faults in Australia has provided insights into the seismic behaviour of the faults. More than 360 geomorphic features with similar characteristics to the surface rupturing faults associated with recent large earthquakes have been recognised across Australia (Figure 4). These features are known or suspected to be associated with earthquakes mostly greater than magnitude 6.0 and are considered likely to have been the source of potentially damaging earthquakes in the recent geological past (Clark, McPherson and Collins, 2011). In addition, these features could again be associated with earthquakes in the feature.

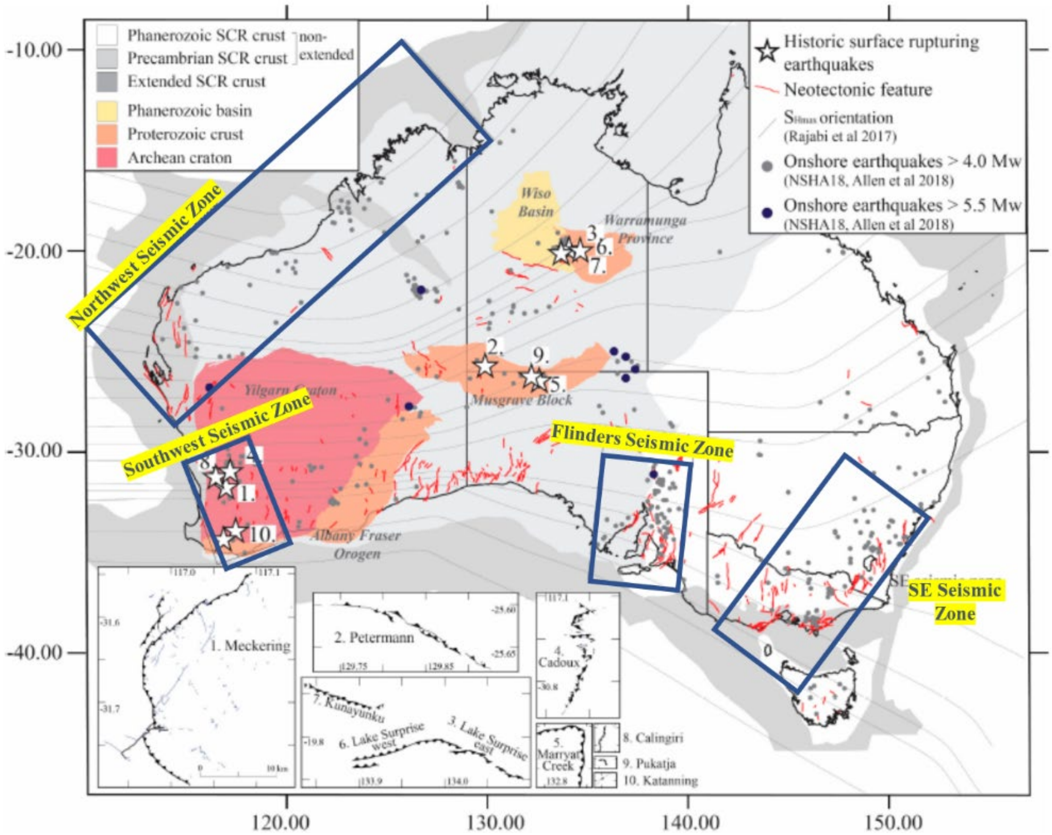



**FIG 4** – Location of recent historic surface rupturing earthquakes in Australia (stars), neotectonic features (red lines), onshore earthquakes with magnitude greater than 4.0 (grey dots), seismic zones, and crustal provinces (modified from King, Quigley and Clark, 2019).

Ongoing palaeoseismological studies of geomorphic features suspected or confirmed to be associated with earthquakes in Australia (ie faults, fault scarps and fault associated folds) suggest that the characteristics and behaviour of the seismogenic faults is variable across the continent. Some of the current understanding of fault behaviour in Australia includes:

- Seismogenic faults in Australia can be classified as active faults (with movement in the last 11 000 to 35 000 years) or neotectonics faults (with movement in the current crustal stress regime, in the past 5 to 10 million years (Clark, McPherson and Collins, 2011)).
- Fault behaviour may be associated with tectonic provinces. Figure 4 shows the tectonic provinces across Australia (ie Phanerozoic Stable Continental Region-[SCR], Precambrian SCR, extended SCR, Phanerozoic basin Proterozoic crust and Archean Craton).
- Palaeoseismic studies across the Precambrian SCR suggest that these faults may have hosted multiple neotectonic earthquakes with long recurrence (>30–70 ka; King, Quigley and Clark, 2019). In contrast, multiple faults in the Phanerozoic non-extended crust of eastern Australia suggest more frequent earthquakes hosted by these faults despite no historic surface rupturing or large earthquakes in this region (King, Quigley and Clark, 2019).

Variable earthquake and fault behaviour across Australia have implications for the regional seismic hazard assessment and the site-specific seismic hazard of open pits.

## SEISMIC HAZARD CONSIDERATIONS FOR OPEN PIT MINES

The seismic hazard of a mine is relative to its location. Although earthquakes can occur suddenly anywhere within Australia, they are more likely to occur in the north-west of Australia, south-west of Australia, Flinders Ranges and South-east Australia.

Considerations for assessing the seismic hazard of a mine include but are not limited to the following:

- If a fault classified as active or neotectonic is located within a few kilometres for the mine, it may control the seismic hazard. This is especially relevant for south-east Australia and the Flinders Ranges zone where palaeoseismological studies (ie Clark, McPherson and Collins, 2011) suggest that more frequent earthquakes are hosted by these faults than in any other onshore Australian region. The probability of activity of the fault, its rate of activity and magnitude distribution must be considered in the seismic hazard assessment.
- Given the highly episodic fault behaviour in the central and western parts of Australia, the seismic hazard in these regions may be better assessed by probabilistic seismic hazard analysis based on randomly occurring earthquakes that are modelled by distributed earthquake sources (King, Quigley and Clark, 2019). A probabilistic analysis of the seismic hazard aims to quantify the uncertainties associated with future earthquake occurrence such as location, size, shaking level at a site (Baker, 2013). Distributed earthquake source model earthquakes as randomly occurring within a specific area.
- Special consideration should be given to mining operations located along the north-west coast where offshore earthquakes and associated phenomena such as tsunami may result in significant detrimental effects for mining infrastructure.

## RECOMMENDATIONS

The assessment of the specific seismic hazard of the mine is necessary to select appropriate earthquake loads for design. However, given the relatively low seismic hazard in Australia, the inclusion of earthquake loads for open pit slopes may not always be required.

There are instances where assessing slope stability with appropriate earthquake loads is recommended, including:

- All slopes where a failure has a high or catastrophic consequence.
- Slopes, excavated or natural, that interact with mine infrastructure (including roads), where failure or damage may result in high financial or social consequences, significant interruption of production and/or injury.

- Slopes with complex or adverse geology or structural features with potential instability issues.

## CONCLUSIONS

There are limited reports of damage of open pit mine during significant earthquake events. Reported impacts to mining operations include:

- rockfalls
- road damage
- limited slope failure
- damage to mining infrastructure such as buildings and mechanical and electrical equipment; loss of electricity.

The lack of significant damage after strong earthquakes in mining operations in highly seismic areas such as Chile, Peru and Papua New Guinea is usually used to justify the avoidance of considering earthquake loads for design. It is frequently overlooked that pit slopes that have withstood strong earthquakes in these regions are often designed under strict seismic guidelines.

The seismic hazard in Australia is lower in comparison with regions located at or close to plate tectonic boundaries. However, strong earthquakes with magnitudes 6.0 or greater are not uncommon. Earthquakes can occur suddenly anywhere within the Australian continent, but there are four regions with more likelihood of earthquakes: north-west of Australia, south-west of Australia, Flinders Range Zone and south-east Australia.

Factors that may affect pit slope performance during earthquakes include topography, slope geometry, rock contrasts, water management, and static factors of safety used for pit designs.

The characteristics of the earthquake activity and the behaviour of seismogenic faults is variable across Australia. Recent palaeoseismological studies suggest that faults in the south-east part of the continent host more frequent earthquakes than in other onshore regions within Australia.

The earthquake occurrence and fault behaviour across Australia have implications for the regional seismic hazard and site-specific seismic hazard of open pit mining operations. Depending on the mine location, its seismic hazard may be controlled by a fault previously recognised as active or neotectonic or its seismic hazard may be more appropriately modelled by distributed earthquake sources.

There are particular cases where seismic design and appropriate selection of seismic loads are relevant for the design of open pit slopes. These cases include but are not limited to:

- All slopes where a failure has a high or catastrophic consequence.
- Slopes (excavated or natural) that interact with mine infrastructure (including roads), where failure or damage may result in high financial or social consequences, and significant interruption of production and/or injury.

The discussion and conclusions presented in this paper are not applicable to tailings dams.

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# Managing uncertainty in large open pit geotechnical modelling

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## ABSTRACT

Uncertainties in large open pit geotechnical modelling has been a major concern for slope stability analysis. The level of confidence in the in-put assumptions, such as structural and rock mass model, dictates the modelling results to some extent. This article is to detail a case study of Kalgoorlie Consolidated Gold Mines (KCGM) regarding a predicted inter-ramp failure on an as-built pit walls by a 3D numerical modelling. Significant concerns were raised across site and multiple preventive plans were considered. After intense data collection work, it was confirmed that the risk of failure had been overestimated. The case study has improved the understanding of geotechnical data in-put and has enhanced the process of creating and updating geotechnical structural models at KCGM.

## BACKGROUND

KCGM Fimiston pit (known as 'the super pit') had a major wall failure in 2018 on Golden Pike cutback east wall. After a thorough geotechnical investigation, one of the contributing factors was determined to be unfavourable major structures developed within the interflow shale sediments. Failure occurred along the weak structures in shale with wedges formed by other persistent joints as lateral releases (Tripp, 2019).

The geotechnical team was concerned about potential other wall failures on the east wall with similar mechanisms. Numerical modelling was conducted by Itasca, a third party consultant company based on the latest rock mass and structural information updated from the east wall investigation.

## POTENTIAL WALL FAILURE

The numerical modelling result showed a low FOS zone (lower than 1.0) on the as-build pit shell (Wine, 2019). This potential failure was about 1 Mt and covered over five benches. The failure mechanism was a similar structural controlled wedge failure. Basal release plane was called Brussels fault (42/270) highlighted in Figure 1, against the east wall facing 265° Azimuth.

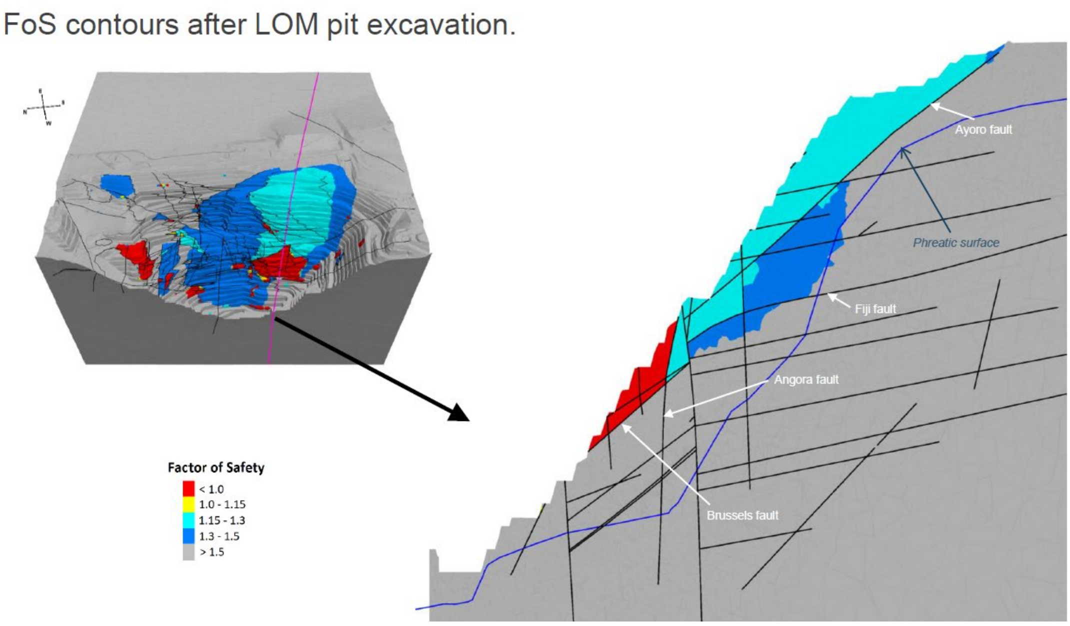


**FIG 1** – Geotechnical modelling result showing low FOS area.

The location of the fault was raised as a significant concern to the design. A recommended preventive plan was proposed to backfill the high-risk wall. However, this plan was relatively costly and also difficult to execute from both safety and production concerns.

## UNCERTAINTY INVESTIGATION

The geotechnical team considered the modelling input information and concluded it to be over-conservative, as monitoring data did not show any considerable short- or long-term movement on the wall. Further investigation was conducted on the critical basal structure – the Brussels fault.

The initial model of Brussels fault was created based on the fault outcrop trace on the lower east wall. The fault model was extrapolated as a relatively planar surface along a few mapping points. The terminations of the fault were cropped by two known regional structures. Although solid evidence of terminating against those regional structures was unclear, this was considered at the time a reasonable and conservative assumption, based on other similar structures and their maximum persistence.

The review questioned two aspects of the original fault model:

1. The planarity of the – suggesting it was over-simplified.
2. The persistence – the structure could also be less than the model showed.

Historical diamond drilling was checked first. A total of two holes intercepted the original Brussels fault model (refer to Figure 2). Core photos indicated there was no major structure around (within 10 m) of the depth of intercepts. This demonstrated the Brussels fault was either not as persistent as the model, or it was a discrete joint other than an open fault, which means the structural strength should be increased.

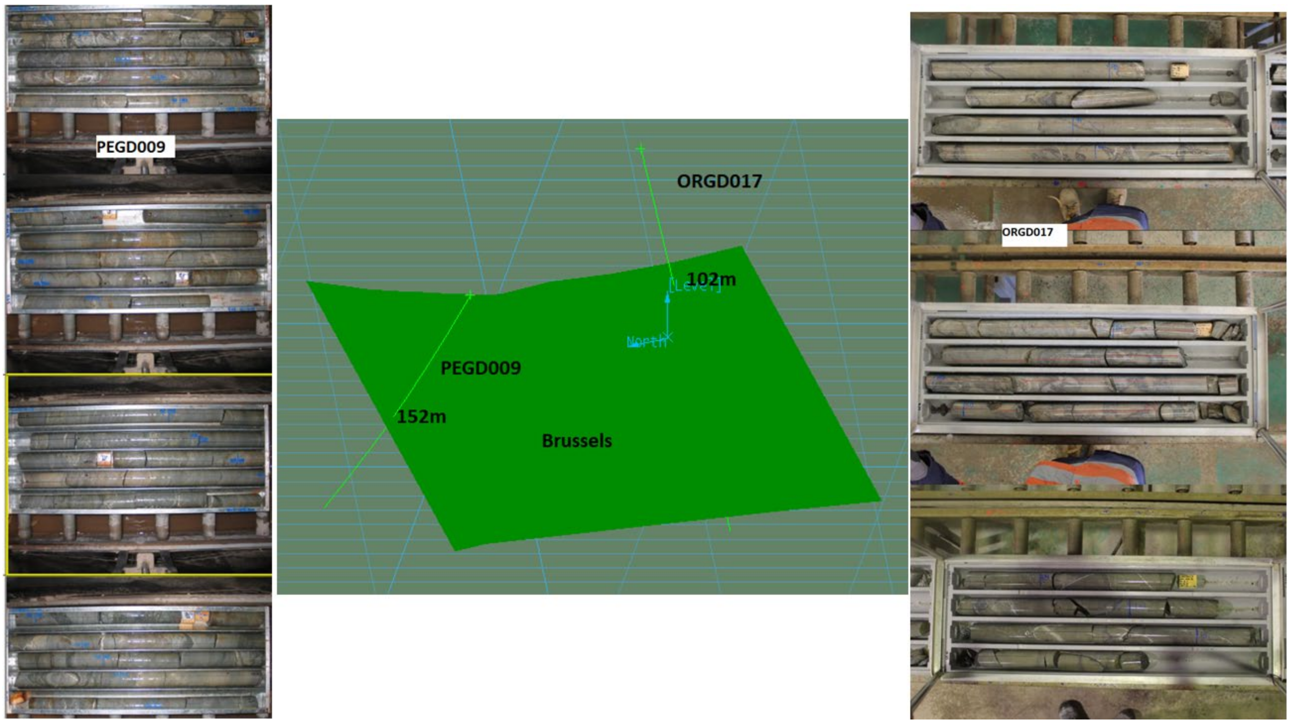


**FIG 2** – Core photos against the structural model.

The next step of investigation was to conduct a detailed drone photogrammetry around the fault outcrops. This was done by a Matrice 200 series drone flying within 100 m of the target wall (refer to Figure 3). Detailed mapping work was then undertaken by geotechnical staff.


**FIG 3** – Photogrammetry model of the target wall.

With the assistance of drone modelling, it was much easier to observe the localised undulation of the Brussels fault (refer to Figure 3). It also revealed that the fault curved and merged into the Oroya fault at the southern end and showed no persistence passing Stanley Crack at the northern end, Figure 4.

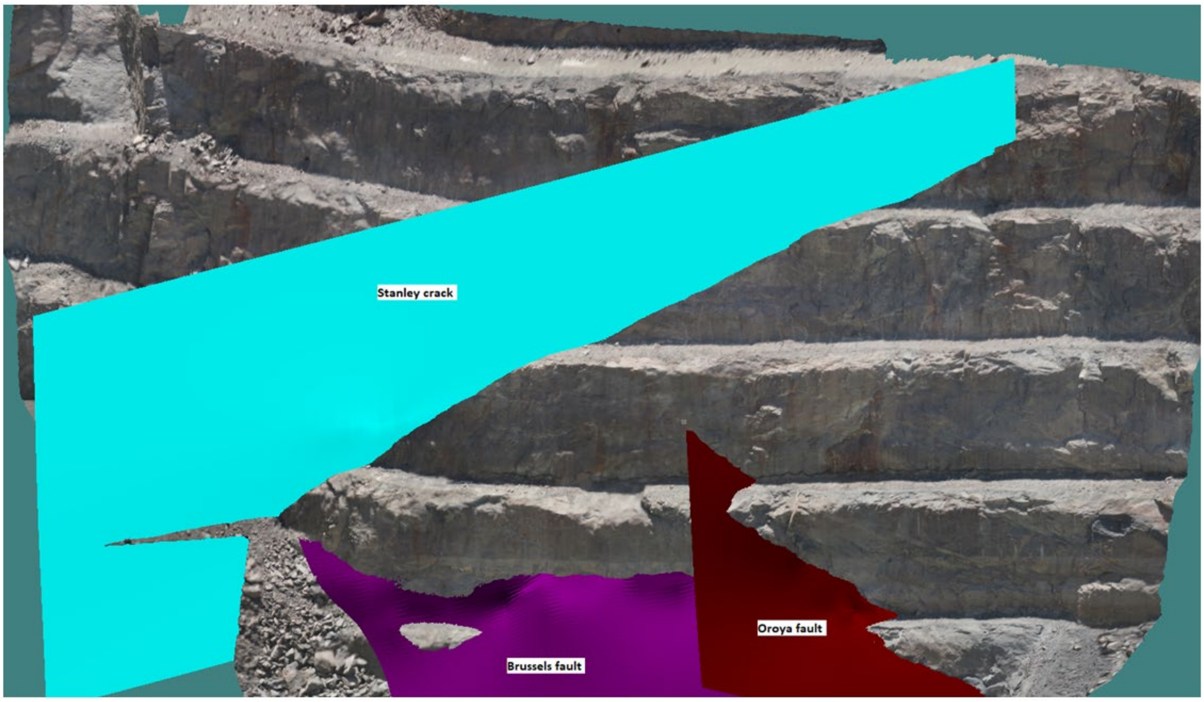


**FIG 4** – Updated model of Brussels fault.

Based on the new photogrammetry model data and the core photo checks, it was confirmed that the actual fault should not be that persistent.

The new fault model was then created based on the new mapping information (Figure 5). The model was significantly smaller and also better represented the undulation and Dip/Dip-direction. This latter was achieved via accurate measurements observed from the detailed drone photogrammetry, which were then used when creating the triangulation.

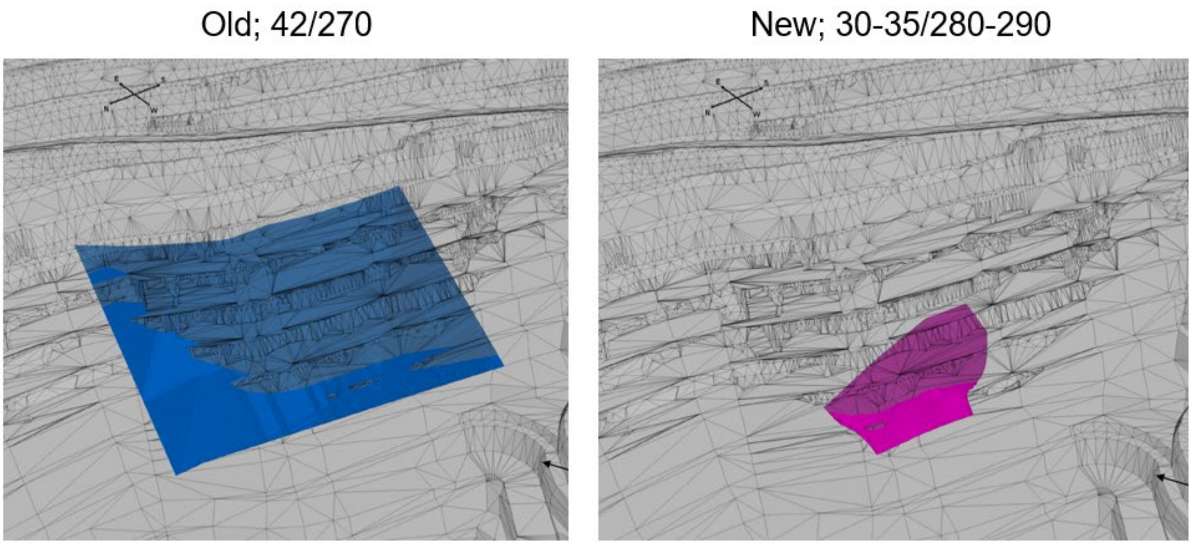


**FIG 5** – Comparison between original and new models.

## NUMERICAL MODELLING ROUND 2

Numerical modelling was conducted, again based on the updated fault model. Modelling result showed a new FOS between 1.0–1.15. The geotechnical team believed the new result should be more realistic although this still indicated some long-term risk of instability due to possible rock mass deterioration over time.

Controlled blasting has been implementing for the shots in the close vicinity of this wall. Monitoring data has been reviewed on a regular basis. At the time of writing this report, this wall is performing to modelled expectations.

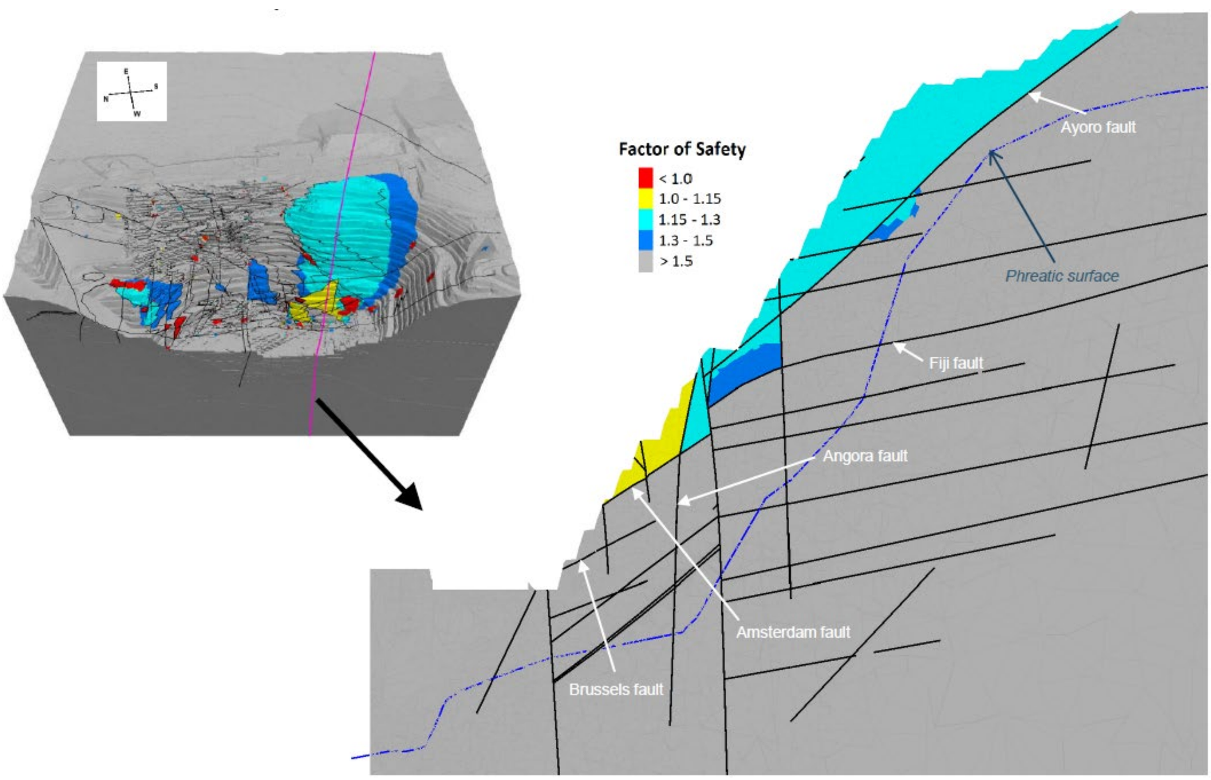


**FIG 6** – Numerical modelling result with the upgraded fault model.

## LEARNINGS

This case study was an interesting lesson for KCGM geotechnical team to understand that modelling is a visualised way to present your geotechnical raw data. Without solid data collection and reasonable engineering judgement, modelling result should always be questioned before making critical decisions. In this case study and improved understanding of structural data improved the modelling results from a FOS <1.0 to an FOS>1.0.

The process of creating structural models at KCGM geotechnical department had been upgraded after this event. Documentations is required for each new or updated structural model. Persistence, terminations, and geological features are to be justified based on either factual data (like mapping or core photos) or engineering judgements/interpretations.

## REFERENCES

Tripp, J, 2019. Fimiston East Wall Structural Model Review, KCGM internal report.

Wines, D, 2019. Itasca 3D Analyses Super Pit East wall, KCGM internal report.

# Learning from and managing a significant saprolite instability

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## ABSTRACT

Two adjacent open pits in Mali hosted by deeply weathered saprolitic materials were mined consecutively over the course of approximately 12 months. During excavation, both pits experienced significant displacement and tension cracking as the result of deeper than expected depth of weathering coupled with relatively steep slope angles and significant groundwater.

In response, geotechnical and operations teams overcame numerous monitoring challenges with prisms and radar to develop a simple and effective monitoring system coupled to a Trigger Action Response Plan (TARP) which enabled continued safe operations while reducing ore loss.

Observations of previous slope performance of these two pits allowed for greater confidence in empirical failure runout calculations that was used to specify the standoff widths for the exclusion zones in the pit.

Back analysis of the slope as-built performance after the first indications of slope distress was also conducted. The back analysis was used to confirm rock mass parameters for modelling while being utilised as a reality check for the stability analysis. The stability analysis indicated that leaving a large step-out or buttress would not prevent the displacement, resulting in some ore being sacrificed to ensure the safety of personnel.

Key learnings from this experience highlight the importance of:

- up-to-date geological interpretation of the depth of weathering
- ensuring slope design guidance is reviewed and updated as latest information (ie geology and monitoring) is received
- having multiple complementary slope monitoring methods to ensure redundancy
- ensuring visual observations and as-built performance are utilised as a reality check to provide greater confidence in stability analysis and design guidance.

## INTRODUCTION

The Tabakaroni gold deposit is hosted by greenstones of the Syama Formation which consist of meta-sediments, mafic volcanics and minor felsic intrusives. Carbonaceous sedimentary units are preferentially deformed/sheared and host the bulk of gold mineralisation in this area (magenta colour in Figure 1). Figure 1 shows a plan view of the geology of the area, with the Namakan and South design pit shapes shown faintly straddling the Namakan shear (ore) zone. The significant instabilities experienced by the Namakan and South pits occurred on the east walls in the volcaniclastics.

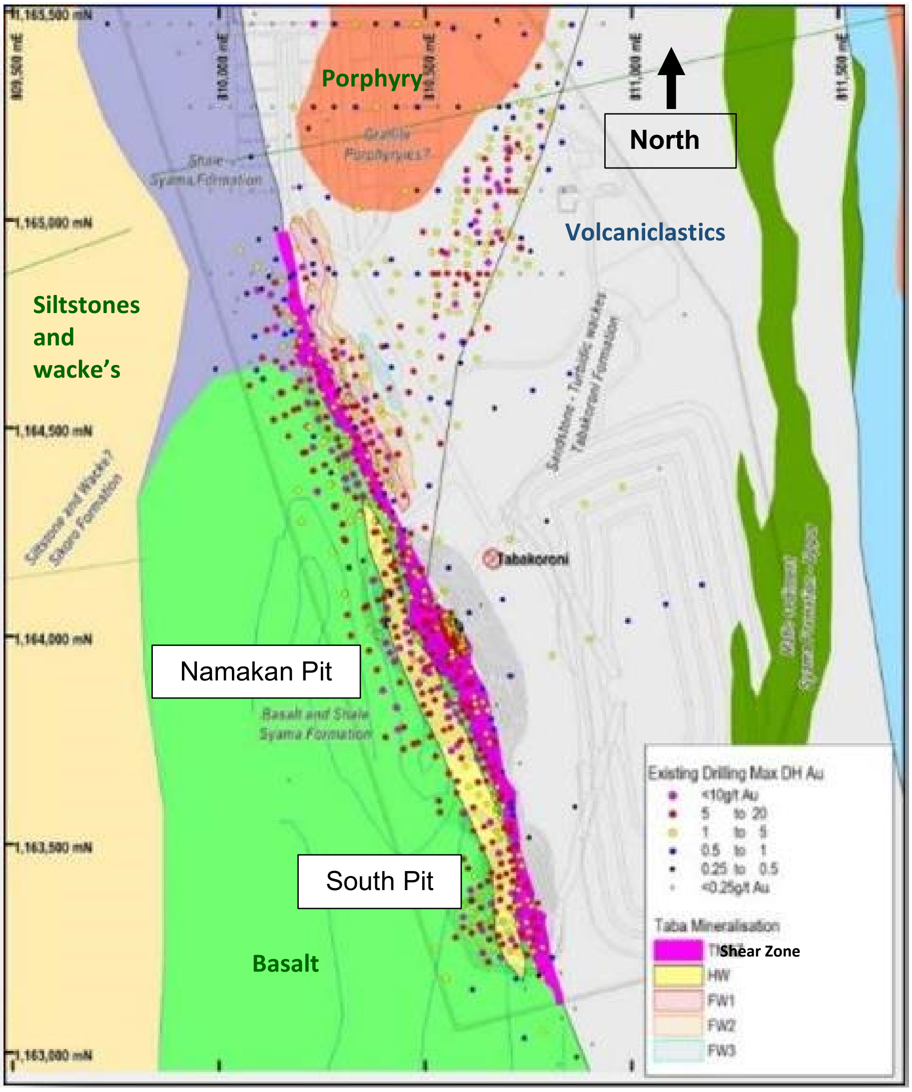


**FIG 1** – Geology plan view with pit and dump overlays (Sharpe, 2019).

Figure 2 shows a schematic geology cross-section through Tabakaroni, with basalt and interflow sediments making up the majority of the western walls of the Namakan and Southern pits. The eastern walls of the Namakan and Southern pits are composed of volcaniclastics with minor porphyry intrusives present in both the eastern and western walls. Also of importance in Figure 2 is the orange dashed line indicating the schematic depth of weathering (Bottom of Complete Oxidation – BOCO) which is considerably deeper in the volcaniclastics on the eastern wall. There is also interpreted folding in the mine area which could not be confirmed in the pit exposures due to the depth of weathering.

The majority of resource drilling was drilled from the west to the east in the basalts to intersect the shear zone that hosts the orebody which dips steeply to the west.

Resource drilling was first undertaken at this series of deposits in 2008–2009 during which a geotechnical consultant was engaged to provide base case slope angles for mine planning purposes.

Visual inspections of available diamond core and core photographs was undertaken, along with 11 Unconfined Compressive Strength (UCS) samples. All the samples failed on pre-existing structures, with UCS values for analysis being taken from average literature/index values.

Geological Strength Index (GSI) and Rock Quality Designation (RQD) estimates in the weathered rocks from core photographs provided GSI estimates of 25–35 and RQD rated at poor to very poor. In the fresh basalt, rock quality was rated as good to very good.

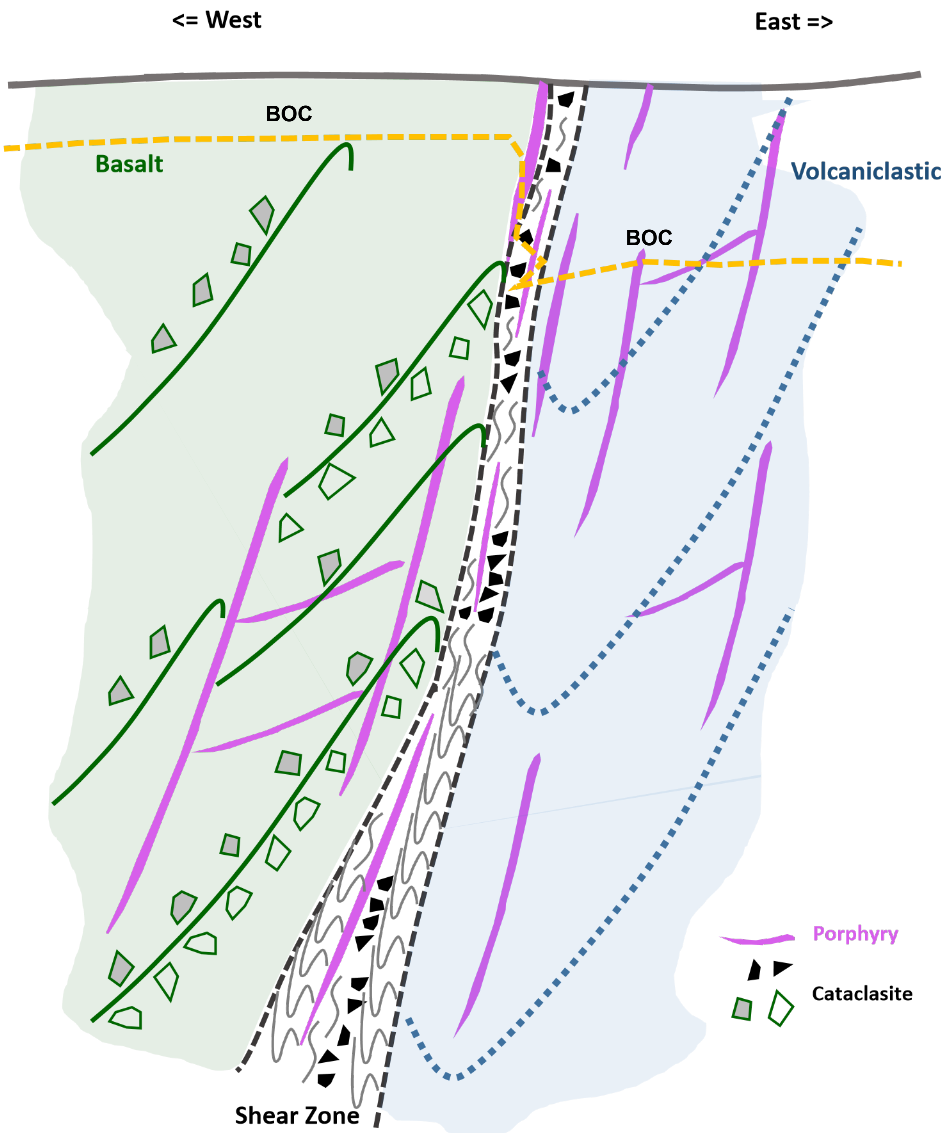


**FIG 2** – Schematic geology cross-section through the Tabakaroni area (Sharpe, 2019).

Resource holes were orientated using the spear method, however the confidence in the orientation marks was poor due to the poor ground encountered and the experience of the drilling team. Structural data for kinematic analysis was restricted to one hole which had high confidence orientations in the basalt on the west wall.

## DESIGN METHODOLOGY

The geotechnical design report in 2009 estimated the depth of weathering from the drilling that had been done at the time as being between 40 to 80 metres.

The failure modes based on the structural data available from orientated drilling, at the time of the design report in 2009 was expected to be bench scale structure related failures in the west wall (Orr, 2009).

The design report recommended that provision should be made for potential horizontal depressurisation drilling be made based on anecdotal evidence of the water table being encountered approximately 40 m below the surface.

From the information available at the time, it was expected that the weathering would be no more than 75 m deep. 2D FLAC/Slope analysis was conducted to evaluate the potential for large scale circular failures using literature values for weathered rocks with poor to very poor GSI and RQD values. The following assumptions were made for the analysis:

**TABLE 1**

Rock mass parameters used for stability analysis.

| Rock type | Specific gravity (kg/m$^3$) | Cohesion (KPa) | Friction angle |
|---|---|---|---|
| Weathered rock/soil | 1800 | 140 | 20 |
| Fresh rock | 2300 | 200 | 25 |

The FLAC analysis and 2D limit equilibrium slope design charts led to the recommendation of base case overall slope angles of 40–45 degrees in 60–80 m deep slopes in weathered rock (Orr, 2009).

These parameters became the base case slope configurations for the majority of oxide pits mined on the extensive leases in the area between 2009 and 2019.

A follow up exploration and resource drilling program was conducted in 2018 as mining commenced in the Namakan pit which allowed for additional geotechnical samples to be collected for laboratory testing of the saprolite (strongly/extremely weathered zone).

The results from this program confirmed the low strengths of the saprolite:

- Cohesion values: 0–10 KPa
- Friction Angles: 28–34 degree
- UCS: <0.1–4.3 MPa
- Specific Gravities: 1600–2200 kg/m$^3$.

Additional resource and grade control drilling collared in the west wall prior to mining led to a digital interpretation of the depth of weathering being produced in 2015 for the Namakan pits which still showed the depth of weathering to be up to 75 m deep.

The majority of the annual rainfall in this part of Mali occurs between May and October with the heaviest falls of 150 to 300 mm per month occurring between July and September. Average annual rainfall is approximately 1000 mm per annum.

The standard process of dewatering for the open pits at this operation consisted of temporary in-pit sumps pumping to adjacent rivers/settling dams or disused pits. Horizontal depressurisation drilling (weepholes) had rarely if ever been used by the operation.

## STAGE 1 NAMAKAN PIT

Mining began in the Stage 1 Namakan Pit in late 2018, with very few geotechnical issues encountered with the exception of minor bench scale structural failures. Stage 1 was completed, and Stage 2 cutback began shortly after in mid-2019.

## SOUTH PIT

Mining commenced in the South Pit in November 2018 and was completed in July/August 2019.

With the exception of some structure related instability below the northern section of the ramp shown as the yellow dashed circle in Figure 3, no multibench scale signs of distress were encountered until June 2019 when the pit reached a depth of 50 m below surface. In June, tension cracks were observed in the lowest benches of the east wall of the pit (see Figure 3).

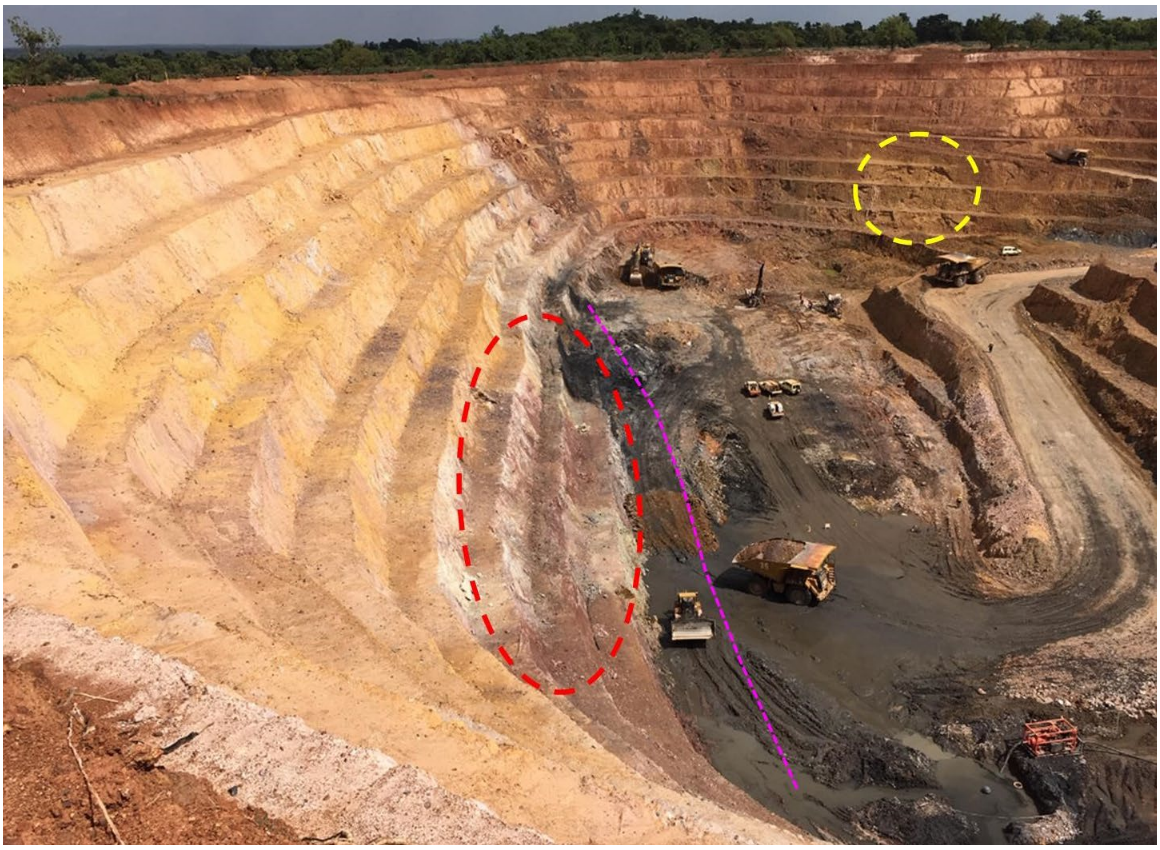

**FIG 3** – South Pit in June 2019 after tension cracks were discovered on the east wall (red dashed circle), yellow dashed circle is a pre-existing structural failure, the magenta line is a large berm put in place as a result of the tension cracks.

Following back analysis and limit equilibrium analysis, a large berm was left to separate personnel from the east wall, quarantine the instability and reduce the overall slope angle of the final design pit shape. The tension cracks in the lower benches continued to extend and dilate with large tension cracks also being discovered 50–70 m behind the crest of the east wall in July until mining was successfully completed in early August 2019 at a final depth of 62 m.

The only survey station at South Pit had been set-up on the east wall with prisms located on the north and east walls. No prisms were able to be read on the east wall as a result. Crack pins were installed when the first tension cracks were discovered in June and read regularly until mining finished in August 2019.

## SLOPE AS BUILT PERFORMANCE

Given the tension cracks and signs of distress being observed in the saprolite slopes on the east wall of the South pit, back analysis of the saprolite as built performance was conducted on the available survey pickups at the time for Stage 1 Namakan Pit and South Pit.

The back analysis cut numerous sections perpendicular to the strike of the survey pickups and measured the height and overall slope angle (toe to crest) of each bench and overall wall of each section. On some sections where ramps were present, additional height and overall slope angle data points were collected at interamp scale in addition to the bench scale and overall wall scale measurements. Those portions of the slopes that were made up of saprock/fresh rock which lie under the saprolite were not used in this analysis.

For sections that showed no signs of distress (no tension cracks or failures), these sections were coloured green, while sections showing signs of distress at the time of the pickup were coloured yellow. The back analysis sections are shown below in Figures 4 and 5.

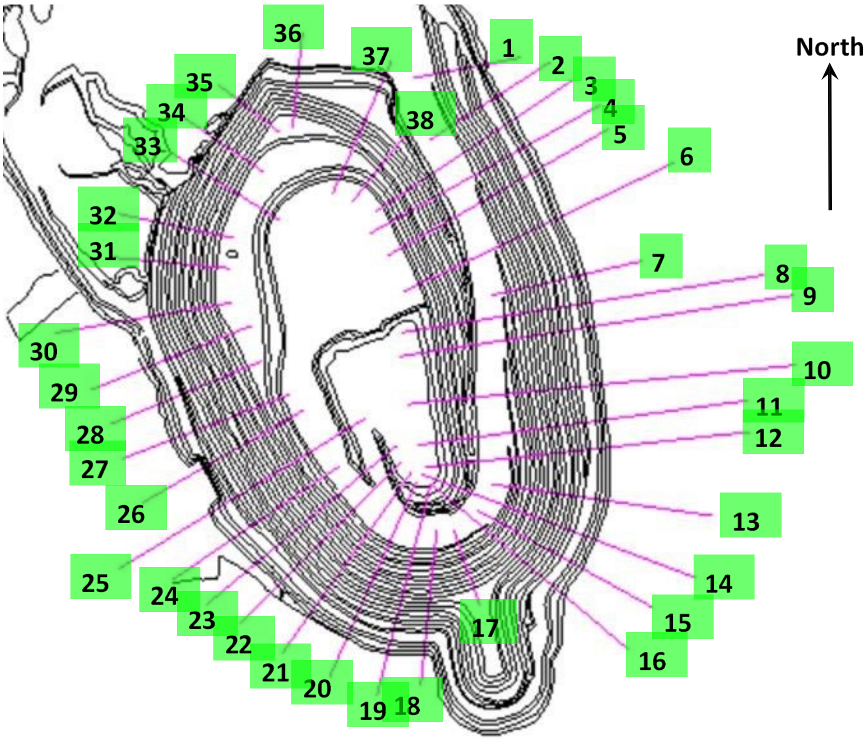


**FIG 4** – Plan view of the survey pickup strings and back analysis sections for Stage 1 Namakan Pit.

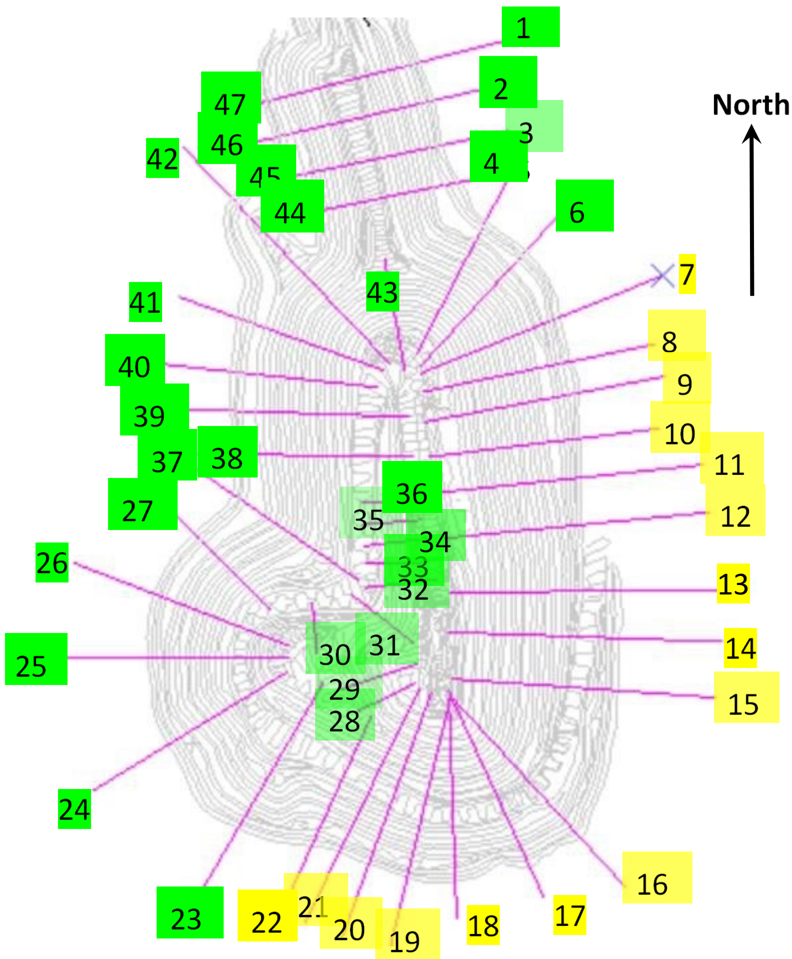


**FIG 5** – Plan view of South Pit at the end of mining with back analysis sections plotted.

Sections 7 to 17 in Figure 5 showed significant signs of distress and tension cracking within the slope and for some distance past the crest. Sections 18 to 22 were part of a complex structure and rock mass style instability and were included in the data set for completeness.

All berm scale data points were given the green colour code even if they occurred on a section that experienced signs of distress as bench scale failures normally occurred during mining and were cleaned up prior to the next bench being taken which is reflected in the as built surveys. All multibench or overall wall data points were given a green or yellow colour code depending on if they occurred on a back analysis section that showed signs of distress (yellow) or no signs of distress (green).

Figure 6 shows the as built performance of the Stage 1 Namakan Pit and the South Pit. At the design overall slope angle of 40–45 degrees, it can be seen that once the saprolite pits get to over 40–50 m deep, the likelihood of instability increased.

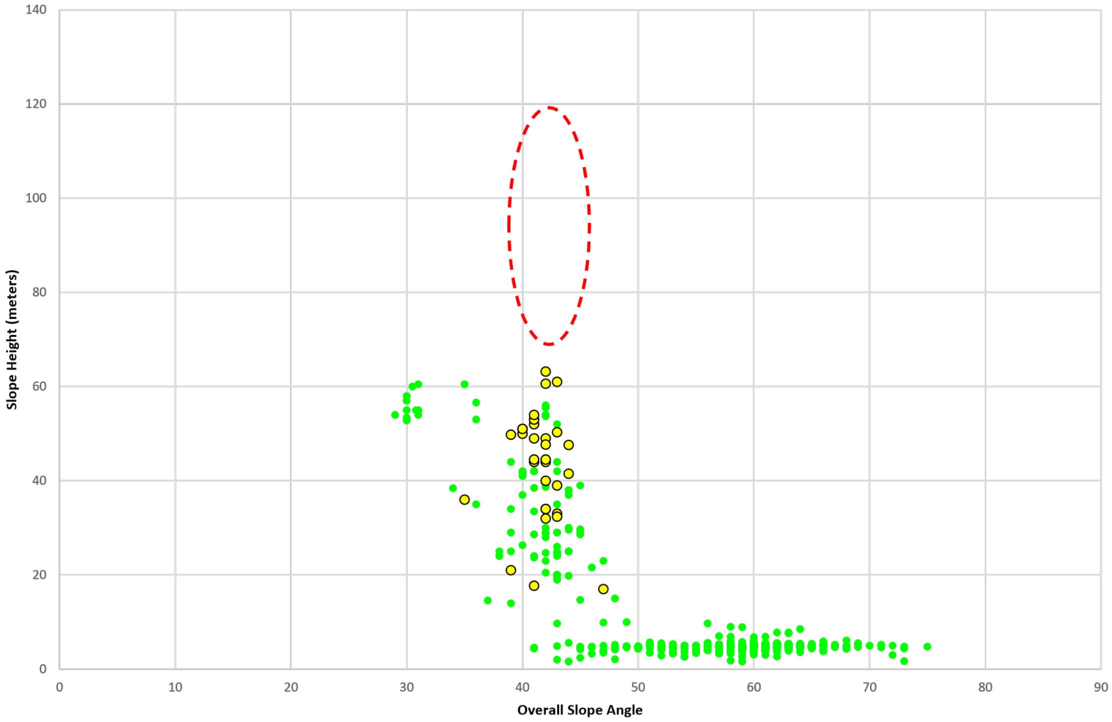


**FIG 6** – Height versus overall slope angle as built slope performance for Stage 1 Namakan Pit and South Pit (Yellow dots are slope sections showing signs of distress, green dots are slope sections not showing signs of distress). The red dashed circle was the planned depth and overall slope angle of Stage 2 Namakan Pit.

The compares favourably with slope performance work using the same technique outlined above, conducted on saprolite pits throughout Africa, Australia and in parts of South-east Asia and South America (Figure 7). The dashed black line in Figure 7 is a best fit line by eye that separates saprolite slope configurations with a higher chance of instability from slopes geometries with a lower chance of instability. This line roughly observes a power law typical of soils and was used in previous studies to back analyse rock mass parameters for partially saturated saprolites (friction angles of 25–30 degrees and cohesion of 20–30 KPa).

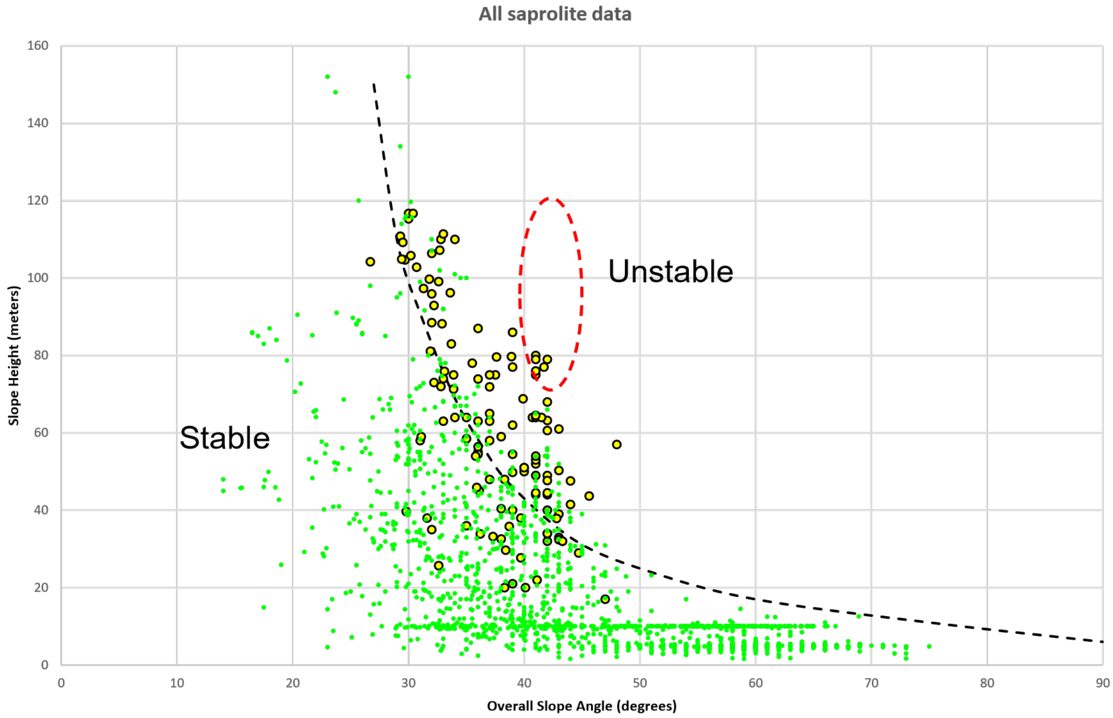


**FIG 7** – Height versus Overall Slope Angle performance chart of saprolite slopes from operations in Africa, South America, Australia and Southeast Asia (Yellow dots are slopes showing signs of distress, green dots are slopes showing no signs of distress). The red dashed circle was the planned depth and overall slope angle of Stage 2 Namakan Pit.

While the South Pit had completed mining, mining had begun on the Stage 2 cutback of the Namakan pit, with the aim of mining to a depth of 120 m using the same overall slope angle configuration of 40–45 degrees that had been used for many years (red dashed circles in Figures 6 and 7).

The geotechnical team became concerned that Stage 2 Namakan Pit may experience instability once the depth of excavation reached 40–50 m.

## STAGE 2 NAMAKAN PIT

Mining recommenced on the Stage 2 cutback of Namakan Pit in June 2019. The geotechnical team began to run scenarios and stability analysis for Stage 2 Namakan Pit in August and September 2019 following the back analysis of the performance of South pit and Stage 1 Namakan pit. The analysis showed how significant the depth of the saprolite (free dig material), pore pressures and wall angle were for stability. The geotechnical team were able to commandeer the reverse circulation grade control drill rig in mid-October to extend grade control holes into the east wall to confirm the depth of the Base of Complete Oxidation (BOCO). This drilling confirmed that the actual depth of BOCO was over 100 m as no saprock or fresh rock was intersected. This was not a favourable outcome given the original interpretation had the depth of BOCO at 60–70 m (Figure 8).

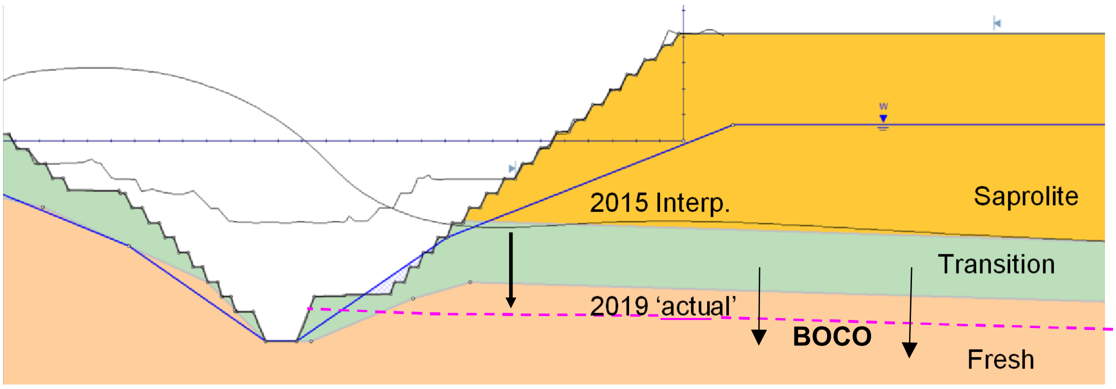


**FIG 8** – Rocscience Slide2 section showing the difference in the depth of the saprolite (BOCO) interpretation following RC probe drilling (60–70 m versus >100 m).

Monitoring during the early part of the Stage 2 cutback consisted of daily manual prism pickups of the east and west walls by the site surveyors. The survey pillars for the prism pickups were placed on the crests of the east and west walls, with the survey pillar on the opposite crest acting as the backsight.

In early to mid-October 2019, anomalous displacement (<1 mm/day) was being measured by the prisms at the toe of the eastern wall. This movement was put down to some rainfall that had occurred during that time.

On 24 October 2019, the shift supervisor discovered hairline tension cracks approximately 70 m behind the crest of the east wall. As the majority of the technical mining personnel used 'WhatsApp', the supervisor was able to send video footage of the cracks immediately following discovery.

The significant length of the tension cracks at that time (100–200 m long) and distance behind the crest appeared to confirm the potential for a very large instability. This also highlighted that the reference point and survey station on the east wall was providing unreliable results.

Crack pins were installed along the tension cracks immediately following the discovery as a monitoring tool while a new survey station and backsight were installed on the north wall to replace the reference pillar on the east wall.

A radar unit from Syama mine was also mobilised to site to add to the monitoring capabilities. Unfortunately, the radar had not been shut down for several weeks leading up to its transfer with numerous problems encountered with the batteries and display units which delayed its ability to provide monitoring coverage until early December 2019. Another significant challenge was the communications link from the radar to the company offices 50 km away and finding a secure way to monitor the radar over a mobile phone-based contractor network. An extraordinary effort by the site electricians, site IT, geotechnical team plus contractors and external service providers was able to overcome the numerous challenges to ensure reliable radar coverage and alarming capability.

An additional challenge was the significant displacement behind the crest of the east wall which had become a hazard for personnel to measure and were stopped in December 2019 (see Figure 9).

**FIG 9** – Personnel conducting crack pin measurements in December 2019.

Table 2 provides a summary of the monitoring challenges encountered and solutions.

**TABLE 2**

Summary of the challenges faced by the Geotechnical and Mining teams when setting up the monitoring systems.

| **Item** | **Challenges** | **Solutions** |
|---|---|---|
| Crack pins | Cracks became very wide and potentially unsafe to measure | Crack monitoring ceased in December 2019 |
| Prisms | East wall survey station/backsight affected by displacement<br>Loss of numerous prisms on east wall due to displacement/localised failures | New backsight and survey pillar installed in the north of the pit<br>High density of prisms installed to cater for this eventuality |
| Radar | Batteries unserviceable<br>User display failure (unable to set-up scan regions)<br>Lack of communications link to company offices >50 km away | New batteries flown in from Europe<br>New display and technician flown in from South Africa<br>wi-fi link set-up to contractor's office on-site<br>Company computer installed on contractor's network |
| 24/7 radar alarming/ monitoring | Limited trained personnel available to monitor the radar on-site | Teamviewer set-up in the emergency services control room to view computer >50 km away at the active mine<br>Emergency services personnel respond to radar alert messages by directly contacting the on call geotechnical engineer |

While the monitoring systems were being set-up, key questions needed to be answered including:

- Could a redesign of the remaining 10 (5 m high) benches stabilise the failure?
- If a redesign could not stabilise the instability, what controls could be put in place to keep personnel safe while extracting as much ore as possible?

## Redesign

The effect on stability from reducing the overall wall angle in the remaining ten benches of the Stage 2 Namakan Pit was evaluated using 2D limit equilibrium analysis and cross-checked with the stability chart.

The limit equilibrium analysis highlighted that even with a large step-off (berm), the instability would migrate down below the step off after two benches had been mined. The stability chart showed that even for a 5–10 degree change in overall slope angle, the overall pit slope would still be in a state of distress (see Figure 10).

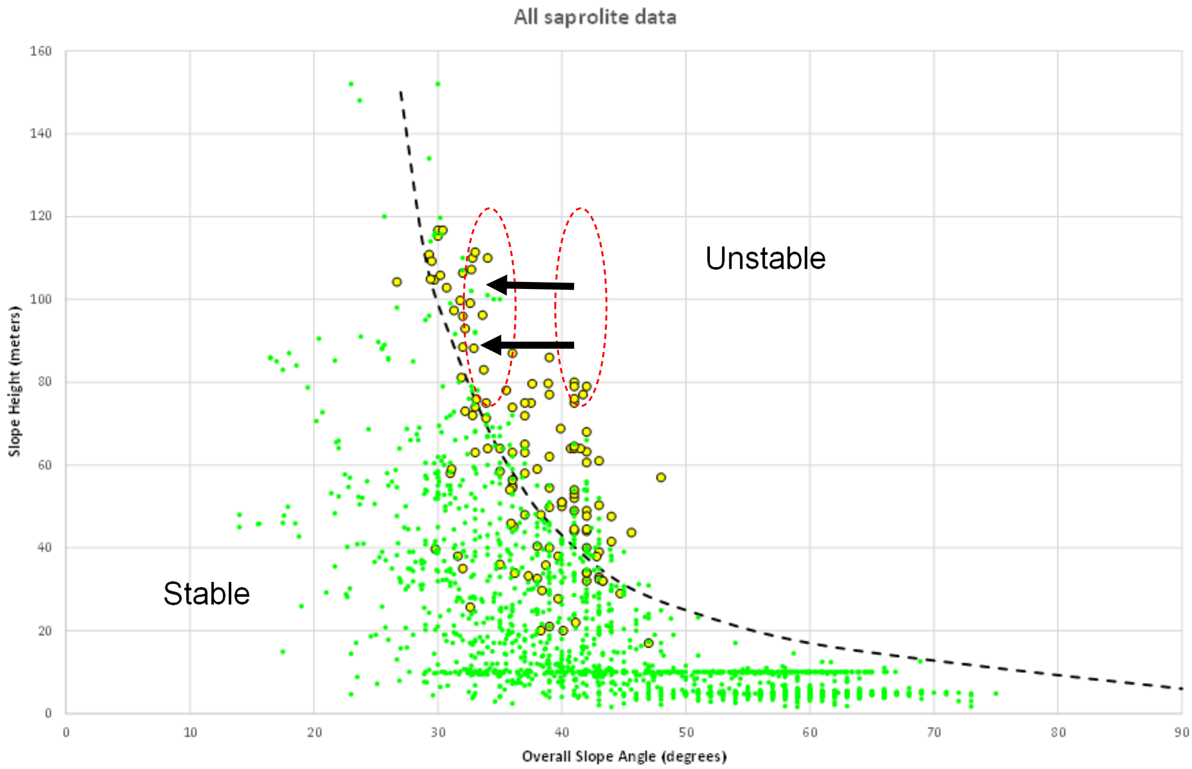


**FIG 10** – Slope performance chart of similar saprolite pits including South Pit.

Limit equilibrium analysis assumed partially saturated conditions due to the inability to safely access the eastern wall to drill horizontal depressurisation holes.

## Managing the instability

With the lack of options to redesign Stage 2 Namakan Pit, the only option was to find a way to safely extract as much of the remaining resource as possible with a modified design that included a 30–50 m wide step off/large berm against the east wall to catch failure material. By this time, it had been accepted that it would not be possible to achieve the design depth of 120 m. The logic behind the controls to manage the instability are summarised by the following points:

- displacement monitoring to understand the size of the potential failure
- runout zone based on the size of the potential failure (high displacement areas of the east wall)
- adaptive exclusion zone along the east wall dependent on the size of the potential runout zone (a function of the size of the potential failure)
- Trigger Action Response Plan (TARP).

The failure mode based on experience in South Pit was expected to manifest as predominantly bench scale failures which progressively increased to multibench scale over days to weeks. Although this was the expectation for Stage 2 Namakan Pit, the geotechnical team maintained a healthy scepticism about the failure mode so as not to become complacent.

## Failure runout distance

A key control for personnel during mining of Stage 2 Namakan Pit was the potential runout distance onto the active pit floor, of any failure. The size of the runout zone would determine the extent of the placement of control zones for the TARP. If the runout zone length exceeded the width of the working bench, this would provide the trigger for complete evacuation and potential abandonment of the pit.

An empirical runout calculation from Whittall, Mitchell and McDougall (2020) was used to understand the range of runout distances relative to potential failure sizes. Figure 11 shows the scale of the potential failures evaluated (black dashed lines) and the estimated dimensions used for the analysis. A conservative triangular prism geometry was used to estimate the volumes of the scenarios shown below.

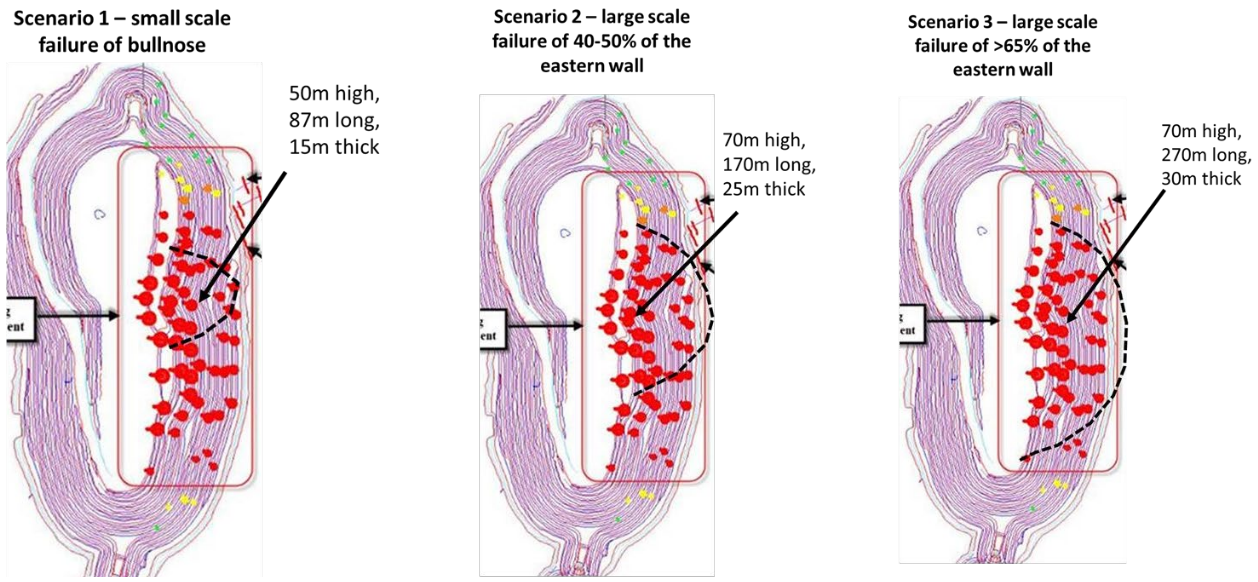


**FIG 11** – Plan view of the three failure scenarios evaluated (the dots in the figure above are prism locations coloured by velocity).

Figure 12 shows the nomenclature used for the runout calculation in Equation (1).

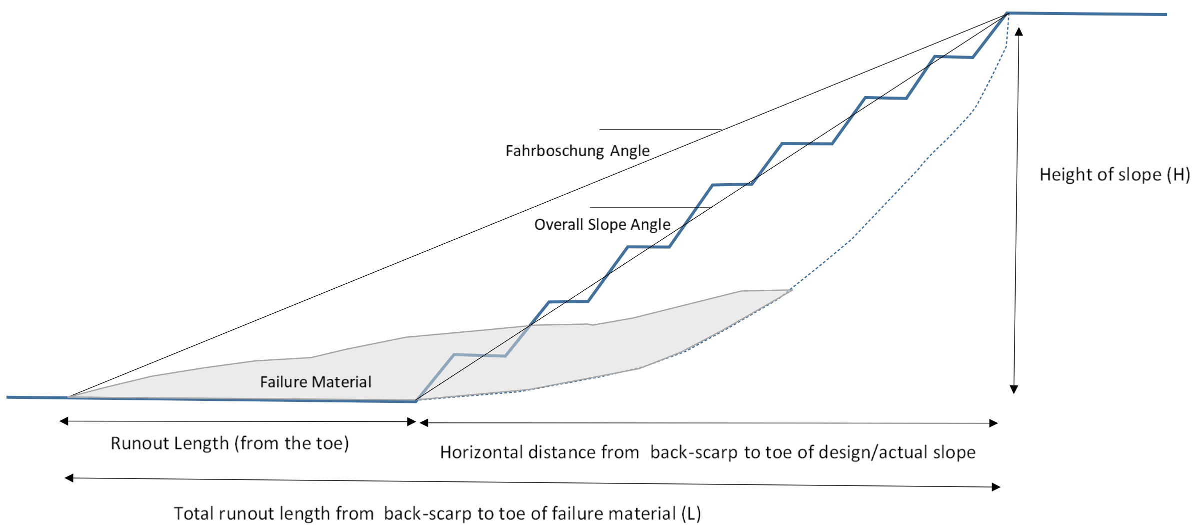


**FIG 12** – Schematic section showing nomenclature used for the runout calculation.

$$H/L = 0.408.\ V^{-0.146} \quad (1)$$

Where:

H = Height (m)

L = Runout Length (m)

V = Volume of the potential failure, in millions of cubic metres ($Mm^3$)

The runout length relative to the toe of the slope was calculated by subtracting the total runout length (L) by the horizontal distance from the back scarp to the toe of the design/actual slope. The results from the calculations for the different scenarios are shown in Table 3.

**TABLE 3**

Runout length for different failure scenarios.

| | Scenario 1 | Scenario 2 | Scenario 3 |
|---|---|---|---|
| **Runout length from toe in metres** | 20 | 55 | 69 |

The eventual multibench failure in the South pit two months after completion of mining offered the chance to provide a reality check (calibration) of the runout calculations. The dimensions of the failure and the estimated runout distance from the toe of the slope are shown in Figure 13.

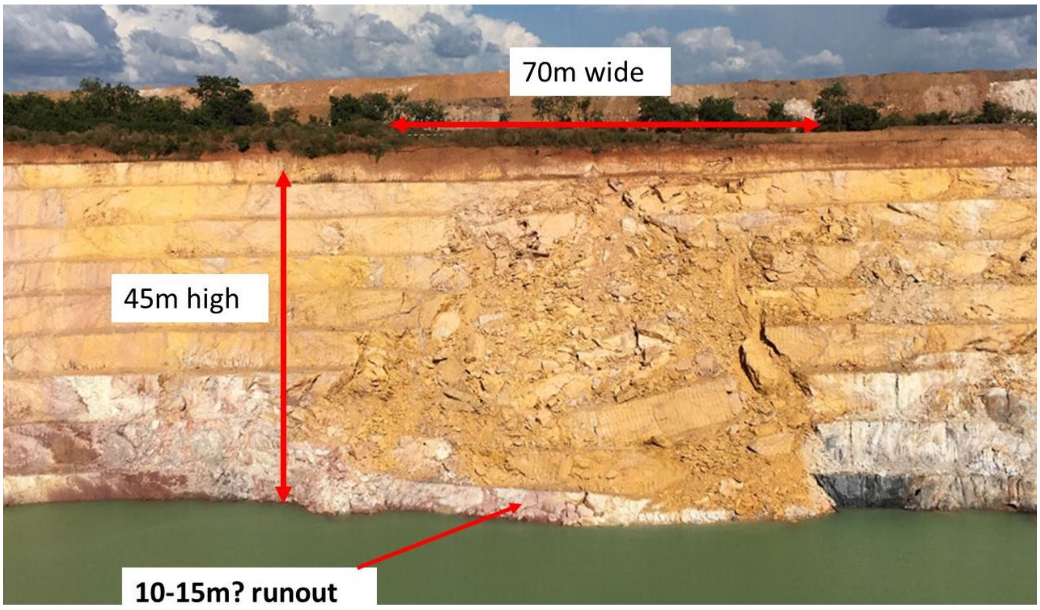


**FIG 13** – Multi-bench failure on the east wall of South pit.

The runout calculation using Equation (1) provided an estimate of 14 m runout from the toe of the slope. This compares favourably with the visual estimate in the field and provided greater confidence in the runout zones scenario calculations in Table 2.

Whittall, Mitchell and McDougall (2020) emphasize that Equation (1) has a 50 per cent chance of exceedance based on the data set used, so the exclusion zones used in the field as part of the TARP for Stage 2 of the Namakan Pit were more conservative than those calculated in Table 2.

## Trigger Action Response Plan (TARP)

The TARP for Stage 2 of Namakan pit used velocity triggers from daily prism and crack pin measurements measured in millimetres per day. Once the radar became available the TARP triggers switched to millimetres per hour to match the nomenclature used by the radar. The TARP triggers for prisms were still retained as a backup if the radar became unserviceable.

The velocity triggers for the prisms and crack pins were kept the same whereas the velocity triggers for the radar were different to the prisms/crack pins due to the shorter averaging time window used.

The radar trigger level for evacuation (Red) in the TARP was 5 mm/h. This was found to be an extremely good trigger level, with velocities greater than 5 mm/h often leading to bench scale failures soon after breaching this trigger level.

The displacement trigger levels and required actions are summarised in Table 4. Part of the TARP was the downgrading criteria to provide a transparent and consistent methodology of deescalating the response.

The TARP was updated every day and communicated via email and a WhatsApp group-chat in the early to mid-afternoon to allow the contractor to make preparations in time for the night shift. WhatsApp was found to be the ideal tool to communicate any sudden changes in addition to the daily TARP status due to the majority of supervisors and management having smartphones on them at all times. WhatsApp allowed for screen grabs of prism/radar plots or pictures from the geotechnical team while also enabling rapid feedback, questions or clarification from the mining supervisors.

**TABLE 4**

Trigger Action Response Plan (TARP).

| **Prisms/crack pins** | **<1 mm/day** | **1–5 mm/day** | **5–10 mm/day** | **>10 mm/day** |
|---|---|---|---|---|
| **Radar** | **<1 mm/h** | **1–2 mm/h** | **2–5 mm/h** | **>5 mm/h** |
| **Action** | Geotech to communicate the change and discuss potential exclusion zone areas as a precaution if velocity increases<br><br>Mining may continue (business as usual) | Geotech to communicate the change and discuss potential exclusion zone areas with contractor and mining manager as a precaution if velocity increases<br><br>Mining may continue (business as usual)<br><br>If data confidence is low, the area in questions becomes ORANGE | Geotech to communicate the change and reassess exclusion zone extent depending on data<br><br>Mining within exclusion on **day shift only** with spotter or alarmed radar in place<br><br>Any rockfall within exclusion zone results in immediate upgrade to RED (evacuate) status<br><br>Contractor to ensure that exclusion zone is in place and enforced<br><br>2× consecutive prism measurement cycles required to downgrade from YELLOW | Geotech to communicate the change and reassess exclusion zone extent depending on data<br><br>Mining within exclusion zone **STOPS** immediately and is evacuated until further notice<br><br>Contractor to ensure hard barrier is in place to delineate exclusion zone<br><br>2× consecutive prism measurement cycles required to downgrade from RED |

Two examples of screen grabs of prism velocity plots of the east wall from WhatsApp have been included in Figure 14. The colour coding is as per the TARP and highlights how the size of the exclusion zone was influenced by the size of the high velocity (Red) coloured areas. The plot on the left is similar to Scenario 1 (runout lengths to 20 m from the toe) whereas the right-hand plot is closer to Scenarios 2 or 3 (runout 55–70 m). The plot on the right in Figure 14 also shows the 30–50 m wide step off put in place as an extended catch berm and defacto permanent exclusion zone. Figure 15 shows an additional WhatsApp screen grab from the radar for one of the daily TARP updates showing the higher velocities in the lower half of the slope which in addition to the prism plots, were used to inform the size of the exclusion zone.

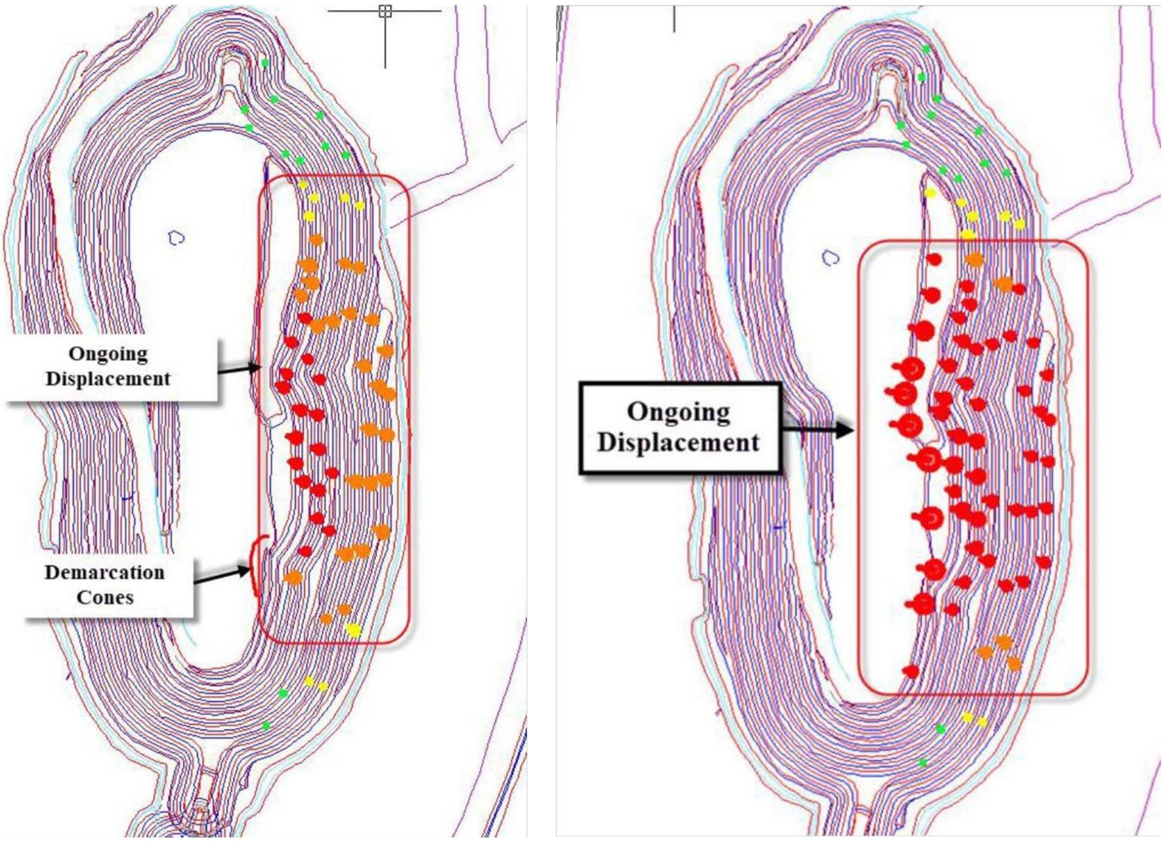


**FIG 14** – Plan view with prism displacement rates (velocity) with the number of high velocity 'red' coloured prisms indicating the size of the standoff zone.

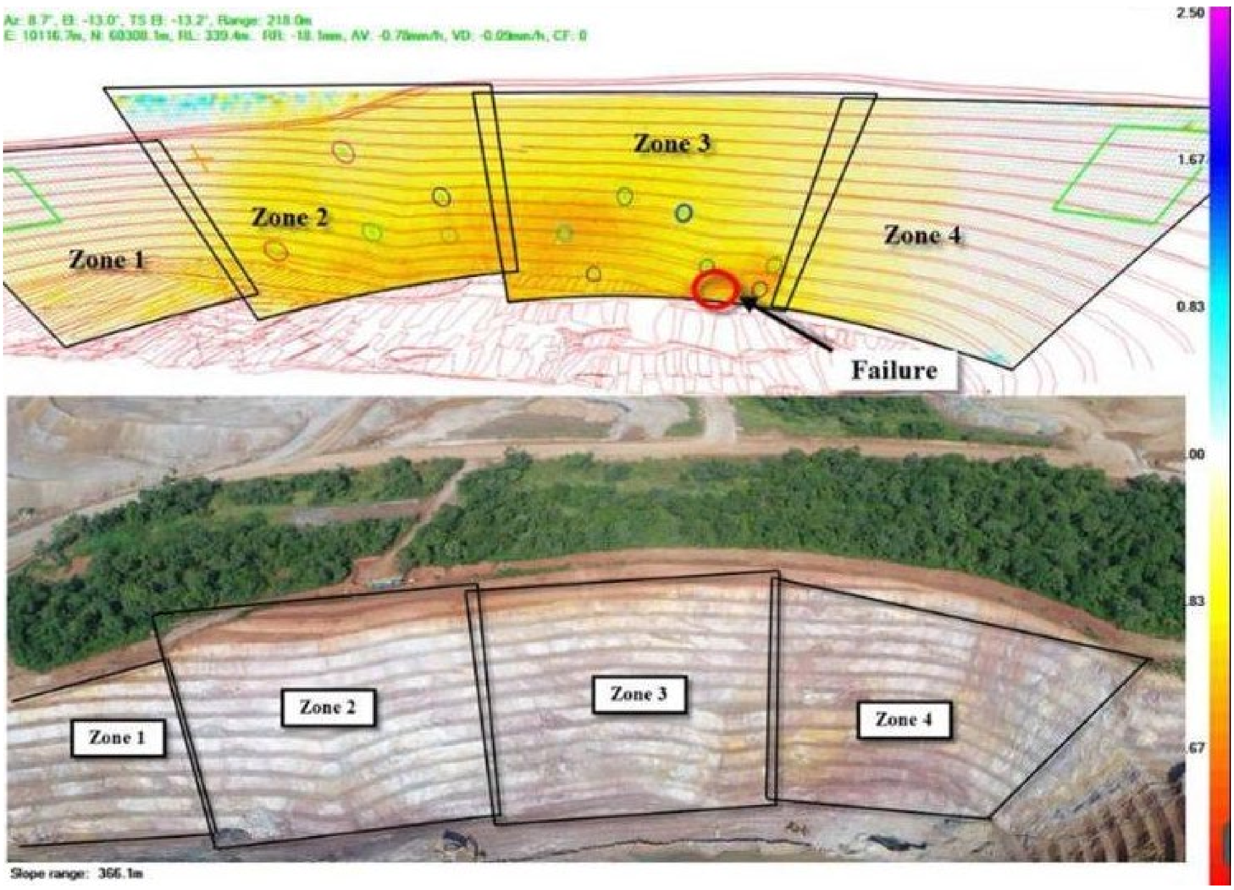


**FIG 15** – Radar screen grabs showing the size of high velocity areas which informed the size of the exclusion zone.

## Performance

While the prism monitoring reference points were being re-established for the east wall, the crack pins became the primary displacement monitoring system for several weeks. In addition to standard displacement over time velocity plots, the crack pins were also presented visually and colour-coded using the same TARP velocity triggers as the prisms (see Figure 16 in late 2019).

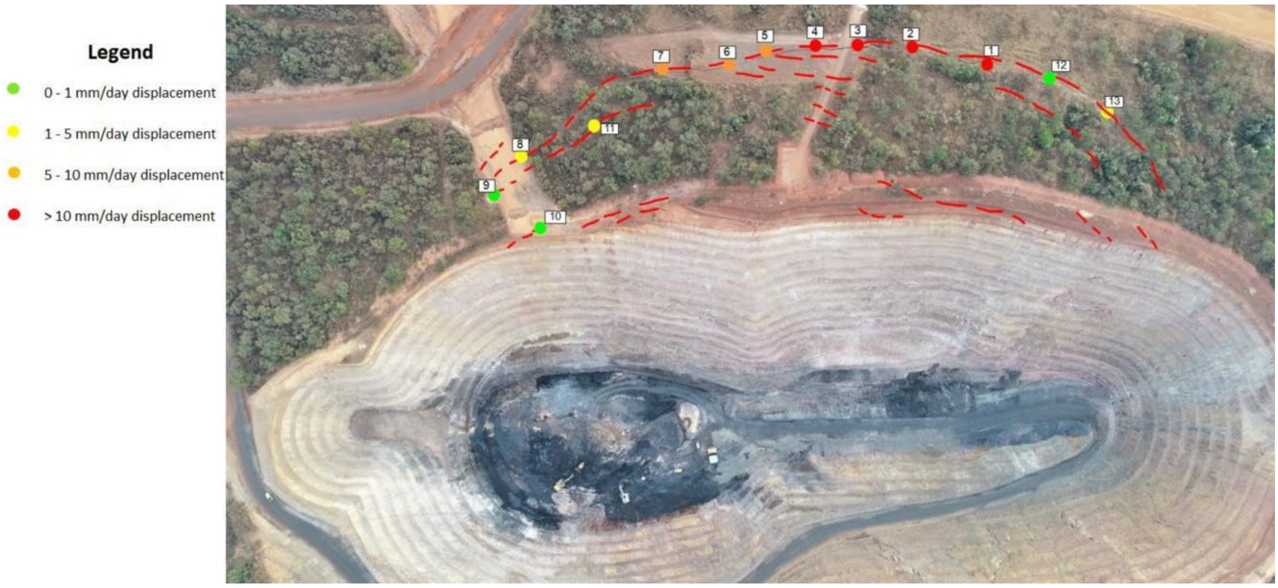


**FIG 16** – Colour coded velocity plots of crack pins.

Once the prism monitoring was reinstated, it allowed for comparisons of prism velocities and crack pin measurements to be made (see Table 5). The two methods had reasonably comparable velocities accepting that the prisms at the toe of the slope were generally moving faster than those at the crest (when measuring slope distance).

**TABLE 5**

Comparison of slope displacement rates (velocity) using different measurement methods over time.

| Working bench (mRL) – surface mRL is 370 | Week ending | Crack pins (mm/day) | Prisms (mm/day) | Radar (mm/h) |
|---|---|---|---|---|
| 345 | 1/9/2019 | – | – | – |
| 330 | 10/9/2019 | – | 0.9 | – |
| 330 | 17/9/2019 | – | 0.1 | – |
| 320 | 1/10/2019 | – | 1.0 | – |
| 320 | 15/10/2019 | – | 0.4–23 | – |
| 310 | 22/10/2019 | – | 0.4–1.7 | – |
| 310 | 3/11/2019 | 1–3 | –* | – |
| 300 | 19/11/2019 | 6–12 | 12–28 | – |
| 300 | 24/11/2019 | 5–15 | 4–17 | – |
| 300 | 1/12/2019 | 0–27 | 21–36 | – |
| 300 | 8/12/2019 | 1–19 | 1–34 | – |
| 300 | 17/12/2019 | 0–21 | 4–33 | 0.4–1.2 |
| 295 | 24/12/2019 | 0–23 | 7–58 | 1.4–1.8 |

| Working bench (mRL) – surface mRL is 370 | Week ending | Crack pins (mm/day) | Prisms (mm/day) | Radar (mm/h) |
|---|---|---|---|---|
| 295 | 31/12/2019 | –** | 7–61 | 0.7–2.7 |
| 295 | 7/1/2020 | – | 44–82 | 1.6–2.7 |
| 295 | 14/1/2020 | – | 17–82 | 1.0–2.5 |
| 295 | 22/1/2020 | – | 12.5–107 | –*** |
| 295 | 30/1/2020 | – | 14–91 | 2.3–3.1 |
| 290 | 11/2/2020 | – | 58–100 | 2.4–5.4 |
| 290 | 18/2/2020 | – | 56–97 | 1.8–4.75 |
| 290 | 25/2/2020 | – | 68–84 | 2.6–4.1 |
| 287 | 3/3/2020 | – | – | 3–4 |

* New survey reference pole installation (cracks discovered behind crest on 24 Oct 2019).

** Unsafe to measure crack pins – monitoring ceases as a result.

*** Radar unserviceable.

Once the monitoring system and TARP had been set-up and were functioning well, the main concern of the geotechnical team was confirming the potential failure mode. Specifically, could fast could a very large scale failure occur given the size and velocity of the instability? Previous experience in South Pit suggested that single bench failures would occur often with several bench failures eventually combining to become a large multibench failure or whole of wall failure over the course of days to weeks.

This was what was eventually experienced in Stage 2 Namakan Pit, with multiple small bench scale failures occurring regularly but rarely resulting in failure material reaching the pit floor. Over time, multiple single bench failures coalesced into a large multibench failure that progressively unravelled around one of the bullnoses (red line in Figure 17).

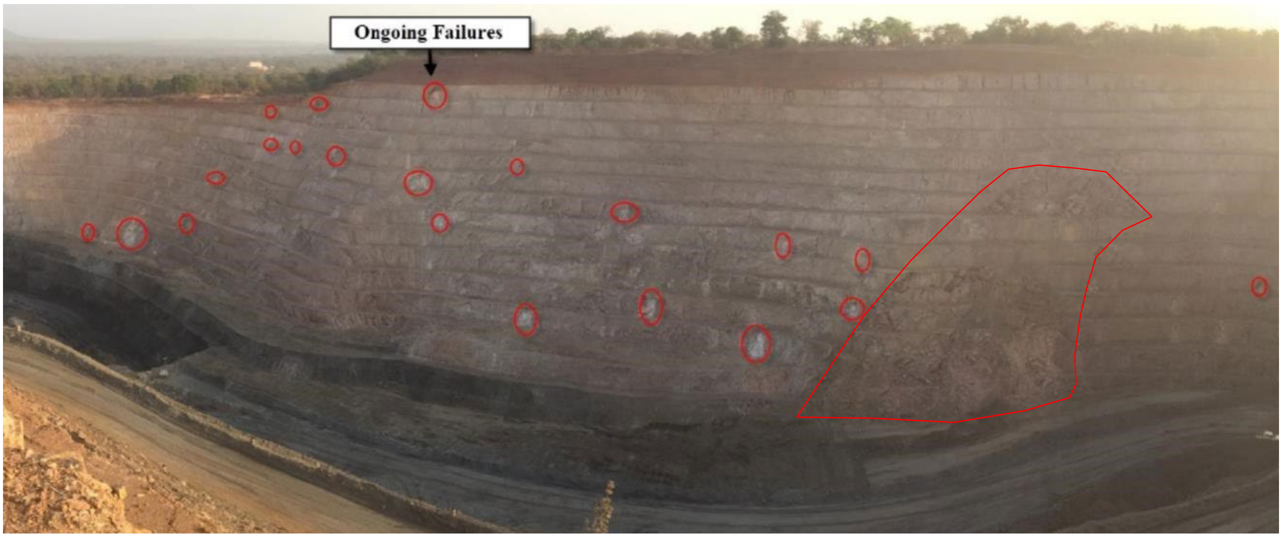


**FIG 17** – Numerous small bench scale failures and the larger progressive multibench failure.

Floor heave became noticeable in November 2019 as can be seen in Figure 18.

**FIG 18** – Floor heave in Namakan pit.

Visualising the movement vectors of the prisms suggested a large-scale circular failure mechanism. Plotting of the change in Z coordinate of all prisms (mRL) on the east wall is shown in Figure 19. This shows prisms at the toe increasing in height over time while prisms at the crest dropped over time, with prisms in the mid slope moving horizontally only.

Mining continued with few interruptions until November, with progressively increasing numbers of shifts where no personnel were allowed in the pit due to a red TARP being triggered. A small pit being mined to the north of the Stage 2 Namakan Pit provided an alternative mining area for the contractor.

Mining of the Stage 2 Namakan Pit modified design was successfully completed in February 2020.

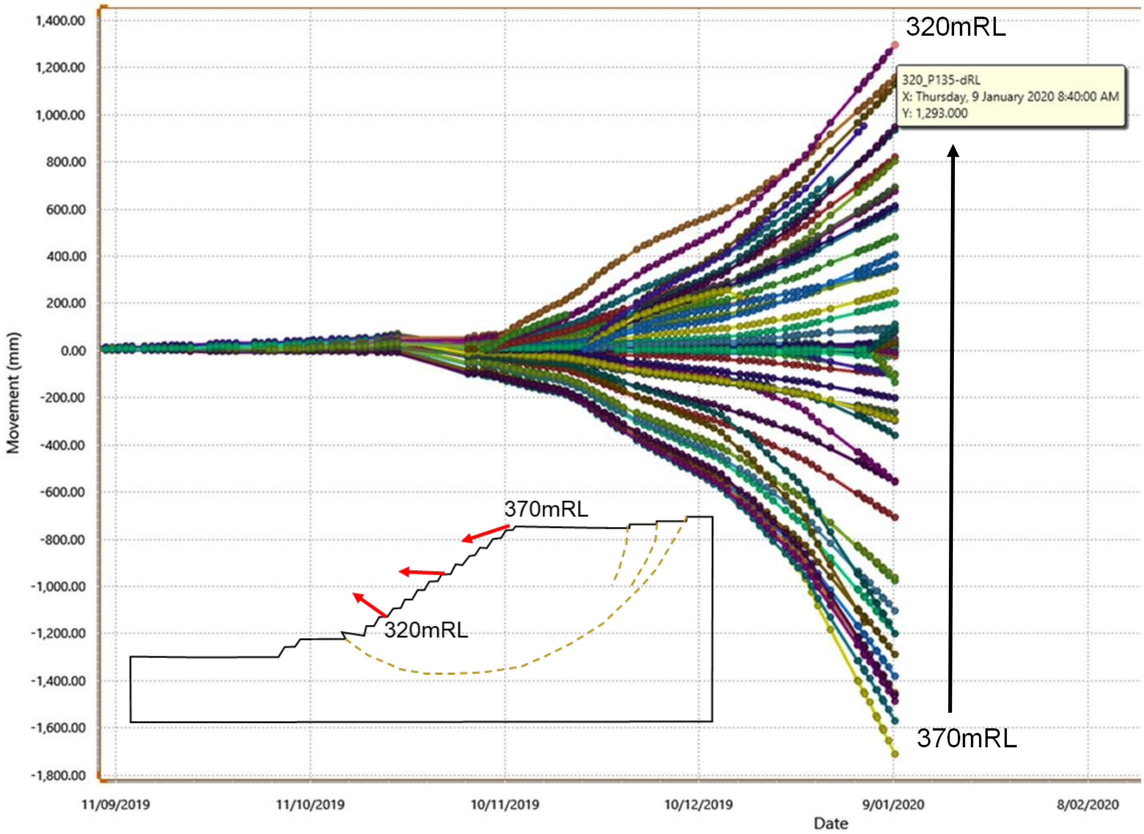


**FIG 19** – Plot of change in Z coordinate of prisms on the east wall suggesting a circular failure mode.


## ACKNOWLEDGEMENTS

The authors wish to thank Resolute Mining for permission to present this work.


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# Slope stability is a critical issue for the open pit mining community

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## ABSTRACT

In recent decades, significant advances have improved the modelling techniques for predicting slope failure. Predicting failures is vital in open pit mining due to the significant risk posed to personnel and equipment from unstable pit walls. Failures not only cause harm and damage to people and equipment but also cause delays to production and subsequently increase operational costs. Coupled with improvements in modelling techniques, utilisation of advanced scanning systems, such as high-definition imagery from UAVs has enabled faster and more precise mapping and modelling. However, one key element required for accurate slope stability modelling and monitoring is the ability for mine geotechnical engineers to have an accurate understanding of the mineralogy and moisture content of slopes, in real-time. To address this challenge, research has been undertaken to utilise hyperspectral imaging across a broad spectrum, combined with high-definition imagery, and LiDAR to scan slopes for minerals and moisture content. In addition to accurately mapping mineralogy and moisture, the data must be collected autonomously and remotely to ensure ongoing safety. The results show that there are many direct and indirect benefits from this new technology, including the ability to monitor slopes at a safe distance and in real-time, improved accuracy of modelling, detection of risks, and improving the automation workflow for geotechnical engineers. The paper presents recommendations for the implementation of the technology in open cut mining environments.

## INTRODUCTION

Recently, there has been a large increase in applications of hyperspectral imaging across multiple industries including environmental monitoring (Stuart, McGonigle and Willmont, 2019), agriculture (Vincent and Dardenne, 2021), clinical research (Pallua *et al*, 2021), agro-food (Lorente *et al*, 2012), microbiology (Gowen *et al*, 2015) and minerals (Ramanaidou and Wells, 2012; Schropp, Knapp and Neubert, 2013; Job, Edgar and McAree, 2017). Advancements in sensor technology have provided opportunities for use at multiple scales. Of particular interest to the mining industry are advancements in portable face scanning sensors which allow for mineral and moisture identification in real-time.

Increasing operating costs and pressures to reduce environmental footprints are contributing factors pushing pit wall design to the upper acceptable safety limits. As pit walls become higher and steeper, slope stability monitoring becomes increasingly important. Slope failures in open pit mines present a significant risk to people and equipment and costs millions of dollars associated with the disruption to operations, clean-up, damage to equipment and loss of reserves (Jele and Dunn, 2019). In the 18 months prior to February 2020, there were five significant open pit wall failures reported in Western Australia, two fatalities in Queensland and Northern Territory and multiple near misses in New South Wales (Government of Western Australia Department of Mines, Industry Regulation and Safety, 2020; Resources Safety and Health Queensland, 2019).

One of the key factors towards the effective prediction and prevention of slope failures is timely, low-cost data that can be collected objectively and integrated rapidly into modelling. An opportunity exists to utilise hyperspectral imaging (HSI), originally designed for ore/waste boundary mapping, for detecting and quantifying minerals and moisture content in highwalls up to 200 metres away from the wall. Such sensors provide a significant opportunity to assist with geotechnical mapping as a real-time, objective mapping tool.

## OBJECTIVES

The objective of this study is to examine the applicability of HSI to geotechnical workflows and the efficacy of the data collected to predict slope failures. There is opportunity to also investigate the potential of the system to fill a gap in the market to provide a live geohazard identification interface for the supervisors in the field, changing the way sites complete their daily inspections.

## HYPERSPECTRAL IMAGING (HSI)

Hyperspectral imaging is a spectral sensing technique that records the light reflected off the surface of materials. The intensity of the light reflected or absorbed by the material will vary based on the atomic and molecular structure of the illuminated surface (Job, Edgar and McAree, 2017). According to Manolakis, Lockwood and Cooley (2016) absorption in rock minerals are due to electronic and vibrational transitions. Information related to the material such as mineral abundances, species, chemistry, and crystallinity can be extracted from wavelength position, symmetry and intensity of specific absorption features that are present in materials' spectra (Laukamp *et al*, 2021). The basic concept of hyperspectral imaging can be visualised in Figure 1.

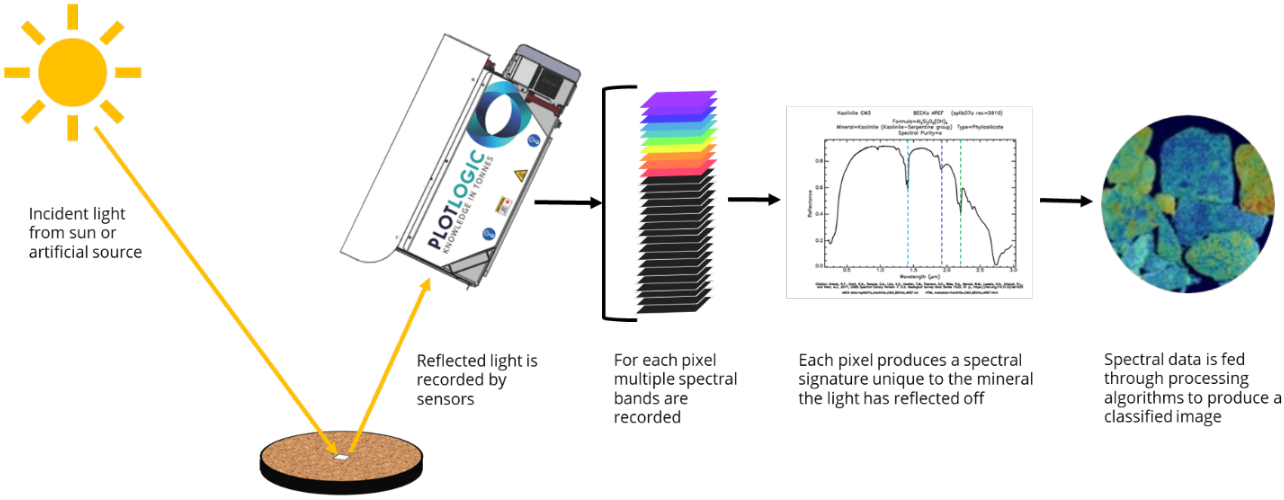


**FIG 1** – Basic overview of the hyperspectral data collection process.

A significant proportion of rock-forming minerals have identifiable features in the visible near-infrared (VNIR) and short-wave infrared (SWIR) spectrum, from 400 to 1000 nm and 1000 to 2500 nm, respectively. Table 1 summarises common mineral groups, their relevant commodities of interest and examples of detection methods used to classify and quantify mineral abundances from their spectral signatures.

**TABLE 1**

Common mineral groups and spectral detection methods used to classify and quantify in the VNIR and SWIR regions of the electromagnetic spectrum (Pattemore *et al,* 2022).

| Mineral group | Commodities | Example minerals | Detection method |
|---|---|---|---|
| Iron Oxides (hydroxides) | Iron | Hematite, Magnetite, (Goethite) | Distinctive features in VNIR; detectable to very high accuracy |
| Simple Oxides | Copper, Titanium, Manganese, Tin | Rutile, Pyrolusite, Cassiterite, Cuprite | Detectable via transitional metals. The signal is generally hidden and requires complex mathematics or machine learning to extract detail. The algorithm can be refined using associated mineralogy |
| Hydroxides | Bauxite, Iron | Gibbsite, Boehmite. Diaspore, Goethite | Goethite is detectable in VNIR; other OH bearing minerals have distinctive features in SWIR |
| Sulfides | Copper, Nickel, Cobalt, Zinc, Molybdenum | Chalcopyrite, Pyrite, Pentlandite, Molybdenite, Cobaltite, Sphalerite | Detectable via transitional metals. The signal is generally hidden and requires complex mathematics or machine learning to extract detail. The algorithm can be refined using associated mineralogy |
| Carbonates | Copper, Iron, Manganese, Zinc | Malachite, Azurite, Siderite, Rhodochrosite, Smithsonite | Distinctive absorption in SWIR. Alternatively, apply semblance or machine learning methods. |
| Rare Earth Oxides | Rare Earth Elements | Monazite (Nd, Sm), Dlorencite (Nd), Goyazite (Sr), Churchite (Y) | Distinctive absorption features in VNIR (excluding La, Lu and Ce oxides) |
| Native metals | Gold | Gold | Detectable in geological context with associated mineralogy and semblance methods |
| Phyllosilicates | Talc | Micas, Chlorite, Serpentine, Clay minerals, Talc | Directly detectable from features in SWIR |
| Inosilicates | | Amphiboles (Actinolite – Tremolite), Pyroxenes | Generally, directly detectable from features in SWIR; otherwise, apply semblance or machine learning methods |
| Tectosilicates | Nickel (laterites) | Quartz, Feldspar group | Detected in geological context with associated mineralogy and semblance or machine learning methods |

## CURRENT GEOTECHNICAL WORKFLOWS

Monitoring is the key to slope instability assessment, management, and risk mitigation (Earle, 2015). The tools available to geologists and geotechnical engineers for data acquisition and interpretation of highwalls vary depending on the site. No design can account for all geological structures, weather events and other geophysical conditions. Therefore, monitoring and detection of failure warning signs is of primary importance for risk management and protecting mine assets and workers (Girard and McHugh, 2000).

Methods of slope stability assessment often combine technologies including photogrammetry, laser, optical satellite images and LiDAR with face mapping of geological structures and mineralogy. Assessment and analysis of this data is commonly manually performed over multiple software packages such as Rocscience (Rocscience, 2022) Seequent (Seequent, 2022) and Maptek (Maptek, 2022).

A common workflow for geotechnical assessment includes the following:

1. Data collection
2. Data analysis
3. Geotechnical design
4. Communication with the planning department
5. Revision and back analysis.

A significant portion of the time and cost of geotechnical monitoring and reporting comes from data collection and analysis. These steps remain predominately a manual process reliant on the quality of the data collected. An opportunity exists for automated real-time mapping of highwalls using hyperspectral technology.

**TABLE 2**

Remote sensing technologies: strength and limitations adapted from Karam and Souza (2017).

| System | Resolution | Limitations | Strengths |
|---|---|---|---|
| Photogrammetry<br>Terrestrial/Airborne | subcentimetre to submetre | Field of view, image resolution, the requirement for surveyed positions. | Rapid, cheap, long-term record, build digital elevation models and see terrain conditions |
| LiDAR<br>Ground-based static and mobile, and airborne | subcentimetre | Humidity, distance limitations, angular limitations, reflectivity, vegetation obscuring, field of view. | Multi return LiDAR allows earth model, very high accuracy, high rate of acquisition, perspective views possible |
| InSAR<br>Interferometric Synthetic Aperture Radar<br>Satellite or ground-based | Millimetre to centimetres | Visibility and shadowing, low coherence in vegetated areas produces unreliable results, limited to slow movement. Can be expensive. | Large area survey, long-term monitoring, return frequency affects the use, comparison or combination of ascending and descending paths, movement measurement, rapid mapping of targeted areas. |
| Optical satellite images | decimetres to metres | Expensive, image quality can be poor due to cloud cover | Landslide surveys, large area coverage, a historical record since 1990s, change detection, rapid mapping of targeted areas. |

These methods are designed to detect rock movement as an indicator of potential failures; however, they generally do not provide information on the underlying cause of potential failures (eg increased moisture, permeability, seals, swelling clays etc).

## APPLICATION OF HYPERSPECTRAL TECHNOLOGY TO GEOTECHNICAL MAPPING

Geological factors affecting slope stability include rock compositions and structure, rock strength, fractures, folds and foliations, geo anomalies, degree of weathering, groundwater, sediment cohesion and *in situ* stress. These factors contribute to the slope strength, which can also be impacted by the moisture, clay abundance and type and permeability of the material. Considering the combination of factors that can lead to slope instability, detection and monitoring of key factors is vital to predicting failures. The capabilities of hyperspectral imaging to detect key factors such as clay types and abundance, moisture content and rock types means it has the potential to be a valuable tool for geotechnical mapping and monitoring of highwalls.

## METHODS

### Fieldwork and data acquisition

Data were collected using Plotlogic's OreSense® system which was used to scan a highwall in Western Australia's Pilbara region. The OreSense® instrument includes hyperspectral cameras, a LiDAR device to build 3D reconstructions and a global navigation satellite system for satellite positioning with Real-Time Kinematic (RTK)for centimetre level spatial precision. The system combines hyperspectral data from two cameras (Table 3) to produce a single image. Each point (pixel) in the data set contains hyperspectral information covering the electromagnetic spectrum from visible light (405 nm) to short-wave infrared (2500 nm). In a standard workflow, the image is georeferenced in 3D space using a LiDAR point cloud with RTK correction for subcentimetre positional accuracy. However, as discussed below, in this case study, the LiDAR data were not used, and the results are entirely based on hyperspectral data.

**TABLE 3**

OreSense® system specifications.

| | |
|---|---|
| **Spectral range** | 408–2517 nm |
| **Spectral channels (bands)** | VNIR – 186 with nominal spacing 3.2 nm<br>SWIR – 288 with nominal spacing 5.4 nm |
| **Spectra overlap VNIR-SWIR** | ~40 nm |
| **Nominal pixel size** | 0.7 milliradians |
| **Image width** | 0.27 radians |

The sensor system is designed to suit a variety of mounting configurations including on the back of a light vehicle (LV), unmanned vehicle, shovel mounted, over a conveyor or gantry set-up and a lab system for high-resolution samples. The LV configuration was used for the geotechnical assessment herein as it provided flexibility for in-pit scanning of highwalls at varying distances. The system is designed to operate in harsh conditions, being built to IP65 specifications with internal cooling. This provides heat, dust and rain protection and the system reliably performed in the hot and dusty open pit mining environment of the Pilbara for a multi-month deployment.

Once data are collected, they are processed using edge and cloud computing. The processing step passes data through algorithms written to detect specific mineral sets and quantify key non-structural geological indicators.

### Data processing classification

Hyperspectral images from the OreSense scanner are calibrated to represent actual light reflectance using one of two methods: (a) a calibration panel; or (b) the MODTRAN commercial product (MODerate resolution atmospheric TRANsmission – an atmospheric modelling approach). The latter was used in this case because of the mining operator's requirements. In addition to calibration. Nearly all mineralogy that is important to the mining operation at this mine site is captured by the hyperspectral sensors, the primary exception being $SiO_2$ which does not have a clearly distinguishing spectral signature within the given bandwidth. Hyperspectral imaging accurately identifies the abundance of all other significant mineralogy, enabling $SiO_2$ to be reliably inferred. A small but consistent measure of the LOI (loss-on-ignition) component exists at the site, but this can be ignored if only a relative $SiO_2$ value is required. The sensor and algorithms provide mineral abundances that are calibrated back to a dry percent weight basis, so consideration of moisture content in calculations is not required. Within the geological context of this site, the inferred relative $SiO_2$ can be reliably regarded as an indicator of unbound granular quartz and therefore, may have a bearing on porosity and permeability if other conditions are satisfied.

Calculating relative inferred quartz content and related prediction of permeability using hyperspectral results is applicable when all other mineral abundances are known and where the quartz is present in granular form in consolidated to unconsolidated strata or stockpiles. Within a suitable geological context, unbound quartz combined with a low clay abundance (taken as <6 per cent $Al_2O_3$ herein) can be used as an indicator of the potential for increased permeability. The risk of algorithm 'confusion' from micellar iron around clay platelets is insignificant because the permeability predictor is only concerned with low clay zones.

Importantly, porosity or permeability are not directly detected; rather, it provides geotechnical engineers and mine managers with a tool for indicating when conditions suggest a probability of increased permeability. This can be used to provide a better focus for sampling programs, or information about otherwise inaccessible sites.

# RESULTS AND DISCUSSION

## Failure back analysis

A failed section of the highwall at an open cut mine presented the opportunity to use hyperspectral imagery to assess the likely cause of the failure. Restricted access for safety reasons presented a challenging imaging environment, as did the absence of previous hyperspectral imagery. Access restrictions resulted in the hyperspectral instrument being placed 400 metres from the face, which is beyond the 200 meter limit of the LiDAR. Therefore, without LiDAR data, the image was projected onto a 2D surface which approximated the actual surface. Additionally, the oblique angle of the sensors to the face required at this site resulted in the left-most area of the scene being significantly more than 400 metres from the camera. As is the case for standard visible light photography, this created difficult lighting conditions. However, despite the various difficulties, the hyperspectral image produced good results for mineral and moisture abundance.

## Non-structural key instability indicators

Hyperspectral images illustrated a thick layer of clay along the lower section of the highwall. The spectral response for this layer has a very clear signature of kaolinite clay. This clay layer is mostly continuous along the length of the wall but with several areas obscured by non-clay debris due to the failure. Moisture is relatively high within clay layers as expected, and moisture can be seen penetrating overlying non-clay debris. Higher in the failure zone are three distinct areas of extremely low clay abundance, which align with areas identified as having potentially high permeability. These three areas of low clay are roughly U-shaped in cross-section (possibly palaeochannels).

## Geotechnical implications

The thick layer of kaolinite detected forms an effective moisture seal which would block water and direct water laterally. Its upper surface also dips slightly (about 5 metres) at the base of the failure zone, indicating that the clay may have also been holding water in a localised perched water table. This clay forms the base of the highwall failure zone. Above this failure point, three areas of extremely low clay and relatively high quartz indicate likely higher permeability. Together, these two features provide the mechanism for both holding water and channelling water into the site from further afield. A localised perched water table located in a depression of a clay layer provides a possible failure mechanism. The mine had received significant rainfall in the weeks before the failure. Failure via such a mechanism would be expected to result in the collapse of a large section of the wall immediately above this point, which is what occurred.

## CONCLUSION

The LV mounted hyperspectral sensing system provides an opportunity to integrate repeated highwall scanning into geotechnical workflows. Geological structures or discontinuities are commonly related to slope failures or highwall movements. However, non-structural geological factors have a great influence on the causes of the movement of such structures. Hyperspectral imaging identified abundant clay underlying the area of instability and a zone of high permeability above the failure which provided a potential path for water to ingress.

Although not a direct measure of porosity or permeability, the relative permeability prediction tested in this study can be used by geotechnical engineers to improve the focus of field sampling programs or provide an indication of conditions that may channel water into a zone of instability. For highwall assessment, this measure combined with clay and moisture contents can help build a more complete picture of a highwall failure risk for otherwise quite inaccessible locations, providing a means of assessment at a safe distance. Beyond the applications considered herein, this technology could be applied to any natural or man-made structure that requires monitoring of moisture levels, various types of clay, or detection of marker minerals in seepage, such as tailings or other dam walls. This study demonstrated that hyperspectral imaging provides a useful method to aid geotechnical slope characterisation.

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# Wall control blasting – the influence of engineering geology on blast design

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## ABSTRACT

This paper is concerned with the influence of engineering geology on design of Wall Control Blasts. In the Introduction, the Blast Management Framework and the Blast Design Requirements classification system (previously developed by the author) is drawn upon to set the context. Blasting objective for all wall control blasts involves both profile and damage control. Four classes of approaches for blast design are outlined and briefly discussed. Five distinct wall control blasting techniques are summarised. The general relationship between geology and blast design is explored in the paper. Influence of geology on pre-split and post-split design and blasting direction is highlighted. Wall damage modelling from the closest row of blastholes is also introduced using the Analytical Damage Model (ADM). The relationship of rock properties and explosive selection and in particular the Mach wave generation is introduced to assist blast designer in this regard. Finally a Conceptual Model that relates vibration, damage and rock mass condition has been presented along with some applications that have been undertaken by the developer. Eight conclusions have been formulated based on extensive work in this area and the current study.

## INTRODUCTION

The blast management framework illustrated in Figure 1 has been previously been published by the author in the European Federation of Explosive Engineers (EFEE) conference series (Little, 2017, 2019). In the previous papers the author showed how the Blast Design Requirements (BDR) classification scheme could be used to improve the first component of the 'Blast design management process'. In the current paper the influence of geology on the second component, blast design and analysis, is demonstrated within a Wall Control Blasting context.

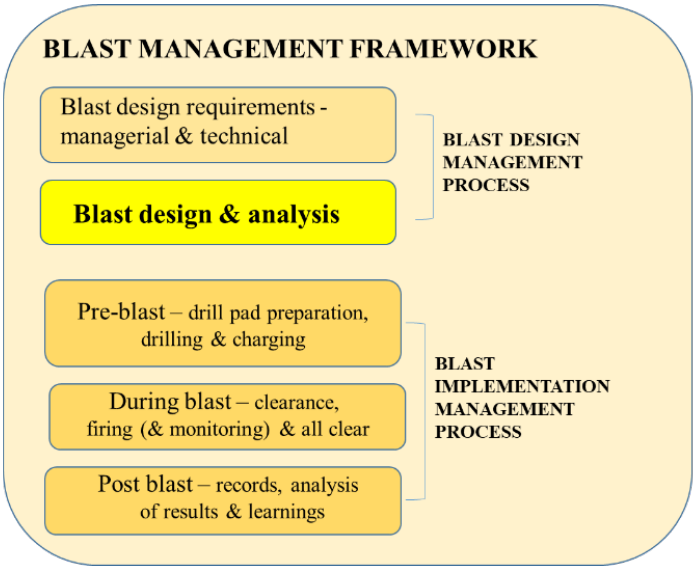



**FIG 1** – Blast management framework showing the component parts (Little, 2017).

The development of the BDR classification scheme began by focusing on grade control blasting (Little, 2015) and was extended to all surface blasting applications (Little, 2017). Further developments to cater for underground blasting operations and to introduces the relatively new concept of 'value-based ore control' were published (Little and Lovitt, 2018). In a recent paper the author (Little, 2019) took the opportunity to provide an update to the BDR classification scheme to include ground improvement as a technical blasting objective. Noting that the overall aim is for the

BDR classification scheme to be relevant to all blasting operations in soil or rock on the surface, underground and underwater.

The BDR classification system uses four classification elements: the blast location relative to the earth's surface, the number of different grades of geological materials, the orientation of the free face relative to the blastholes, and the number and type of primary technical blasting objectives. The first three classification elements are objective physical properties and the fourth relates to technical blasting design objectives.

Blasting objectives have been grouped into two categories managerial and technical, see Figure 2. The premise is that all blasts must satisfy the managerial objectives and hence have not been included in the BDR classification scheme. That is, all blasts must be safe, legal, add value, take into account any special conditions and work within site constraints. Note that blast design is the fundamental engineering control to obtain consistent outputs that achieves all relevant blasting objectives.

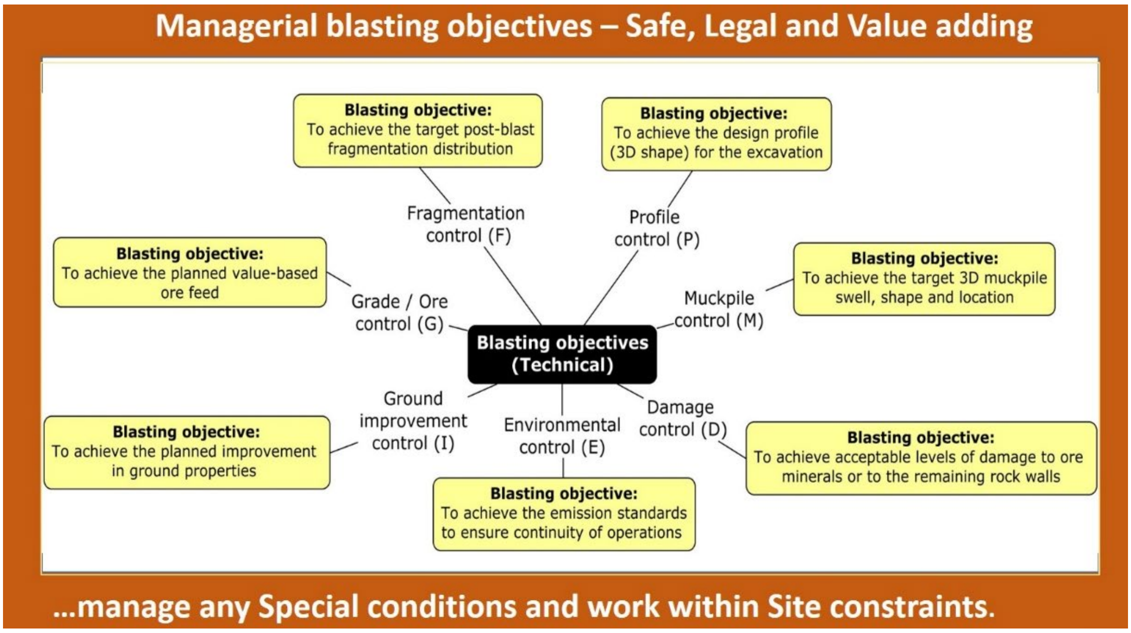



**FIG 2** – Technical and managerial blasting objectives (Little, 2018).

**TABLE 1**

Technical blasting objectives (Little, 2019).

| Technical blasting objective | Interpretation in soil and rock blasting context |
|---|---|
| G Grade/Ore control | Reducing unplanned ore loss and unplanned dilution. |
| F Fragmentation control | Obtaining the desired fragment distribution. |
| D Damage control | Decreasing strength reduction of the remaining rock mass. Reducing damage to valuable minerals. |
| P Profile control | Reducing overbreak, underbreak and bridging. |
| M Muck pile control | Obtaining required muck pile and ore flow characteristics. |
| E Environmental control | Reducing unwanted blasting emissions eg vibration, airblast, fumes, dust, flyrock, water pollution and asset damage. |
| I Ground Improvement control | Increase density and/or shear strength of weak geomaterial zones. Reduce stress in highly stressed rock volumes. Eliminating voids in the rock mass etc |

The four main approaches to blast design are illustrated in Figure 3.

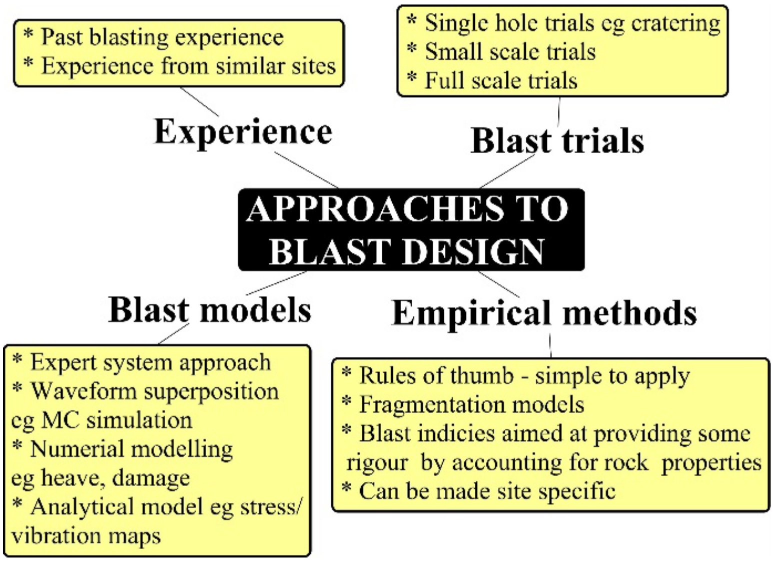



**FIG 3** – Approaches to blast design.

The experiential approach may be acceptable for once-off shotfiring tasks but is not suitable for mine blasting. Benchmarking from similar sites can be a good starting position. Trial blasting is a sound approach to blast design as feedback on the performance of the design is rapidly received (days). It should be noted that design feedback for slope engineering can take years to decades. Empirical methods, such as rules of thumb or blasting indices, can be used as a first pass design or as a broad brush audit check. A number of empirical fragmentation distribution models are available and commonly used. Customised or site specific blastability indices can be developed and used successfully. Blast Model assisted design is the approach considered to be the 'state-of-the-art' and is demonstrated in this article.

In the remainder of this paper the following topics are discussed:

- wall control blasting objectives
- wall control blasting techniques
- geology and blast design in general
- influence of geology on pre-split and post-split design
- wall damage and standoff distance from close blastholes
- rock properties and explosive selection
- the conceptual model for vibration and bulk movement.

## WALL CONTROL BLASTING OBJECTIVES

Wall control blasts involve both profile control and damage control blasting objectives. Using the BDR classification scheme a typical Wall Control blast will take the following form: S_SG_VF_PDO where: S – Surface blast, SG – single grade of material (ie ore or waste), VF – vertical or subvertical free face, and PDO – profile and damage control technical objectives in priority order. Note some wall control applications may set a priority of Damage ahead of Profile (DPO). Figure 4 shows the range of profile control techniques. Pre-split, post-split, and line drilling (and guide hole) will not be discussed here as they are addressed in the next section.

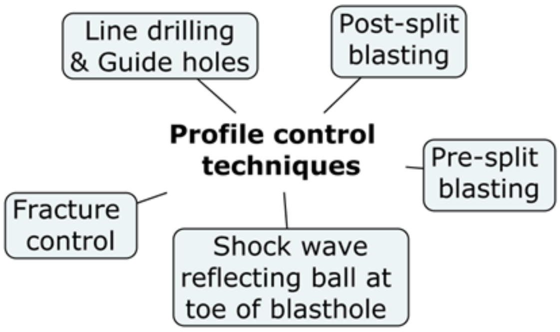



**FIG 4** – Profile control techniques.

Fracture control aims to control the generated fracture direction and length. It can be achieved by modifying the blasthole shape, charge shape or charge configuration. The shock wave reflecting device placed in the toe of a blasthole is a new and novel approach used in contour (profile control) blasting for dam engineering in China (Lu *et al,* 2018).

Figure 5 illustrates the main damage control blasting techniques. These work in association with profile control techniques and it is critical they do not compromise them. For example, if the trim blast takes out the pre-split line the mine is wasting money, may not achieve design and damage objectives and potentially creating a rockfall or mass stability hazard. All these wall control techniques are discussed in the next section of this paper.

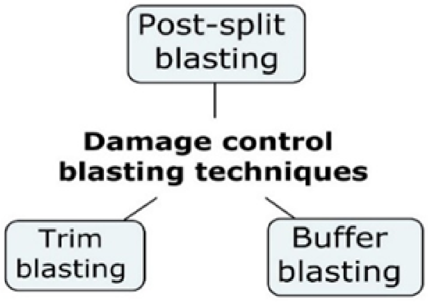
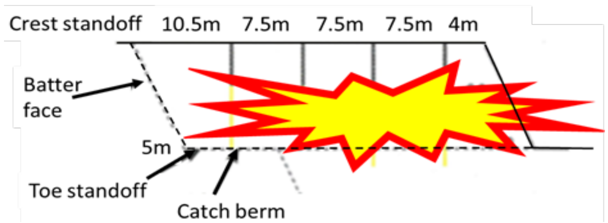
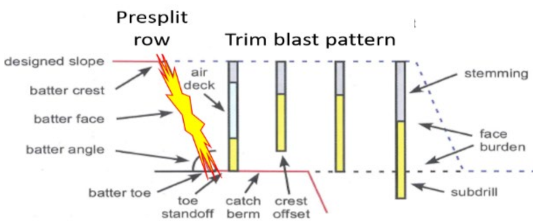
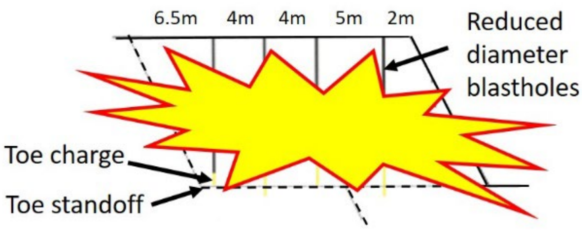
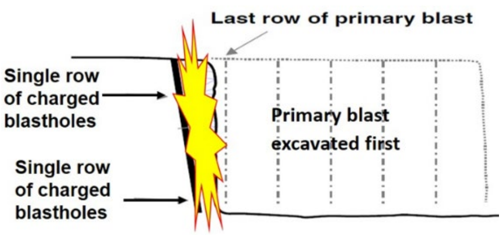



**FIG 5** – Damage control techniques.

## Wall control blasting techniques

Wall control blasting terms are not agreed or uniform across companies, states and countries. Even the term wall control blasting has many aliases, for example, limits blasting, controlled blasting, contour blasting, perimeter blasting and smooth-wall blasting. To ensure readers are clear, the Wall Control blasting terms used in this paper are defined in Figure 6. In short: a buffer blast uses stand-off to avoid damage and the wall is formed by the excavating equipment; a Pre-split blast defines the profile and the thin damage zone attenuates vibrations; a Trim blast attempts to balance the charge diameter and stand-off distance; and a Post-split blast uses a single row of closely spaced holes along the final wall and shots to a free face. Line drilling is an alternative to pre-splitting and is not classified as a blasting technique.

<table>
<tr><th>Method and illustrative cross-section</th><th>Features</th></tr>
<tr><td>Method 1 Buffer (Stand-off) Blasting<br>
</td><td>Buffer blasting can be used in weak rock masses and has the following general features:<ul><li>Toe standoffs is critical</li><li>Blasthole diameter can be same as production holes</li><li>Charge in buffer rows can be reduced</li><li>Excavation equipment digs to hard</li><li>BDR classification S_SG_VF_DPFO</li></ul></td></tr>
<tr><td>Method 2 Pre-split Blasting<br>
</td><td>Pre-split or mid-split blasting works best in massive rock with tight joints and has the following general features.<ul><li>Smaller diameter blastholes and closely spaced</li><li>Loaded with decoupled charges to reduce blasthole pressure</li><li>Blastholes are fired prior (milliseconds or days) to the first hole in the adjacent pattern</li><li>Un-stemmed unless mine is airblast sensitive</li><li>BDR classification: S_SG_VF_PDO</li></ul></td></tr>
<tr><td>Method 3 Trim Blasting<br>
</td><td>Trim blasting can be used in a wide variety of rock masses and has the following general features:<ul><li>Generally uses smaller diameter blastholes and can be used with or without pre-split</li><li>The balance between charge weight and standoff is critical but the balance between charge diameter and stand-off distance is even more critical</li><li>Can be free faced or choked or reverse fired</li><li>BDR classification: S_SG_VF_DPFO</li></ul></td></tr>
<tr><td>Method 4 Post-split blasting<br>
</td><td>Post-split blasting is not often used and has the following general features:<ul><li>Row of closely spaced holes along the final wall</li><li>Light and well distributed charges used</li><li>Blastholes drilled and fired after primary blast is excavated (Dynamic post-split has also been used)</li><li>Critical that damage from primary blast not to extend beyond post-split line</li><li>BDR classification: S_SG_VF_PDO</li></ul></td></tr>
</table>

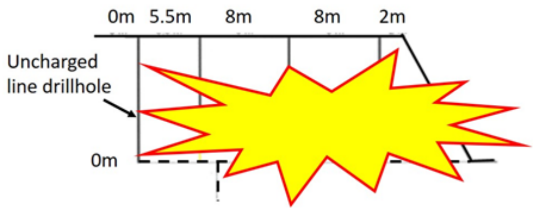
| | |
|---|---|
| Method 5 Line drilling<br><br> | Line drilling can be used in most ground and has the following general features:<br>• Line of closely spaced unloaded holes drilled along the final limit<br>• This weakened zone helps guide the excavation of the slope<br>• BDR classification: Not applicable |

**FIG 6** – Wall control blasting techniques (Little, 2019).

## GEOLOGY AND BLAST DESIGN IN GENERAL

In this section the general changes that can be made to blast design to cater for changes in geology and rock properties are illustrated in Figure 7. It should be noted that these are also relevant to wall control blasting if geological conditions exist.

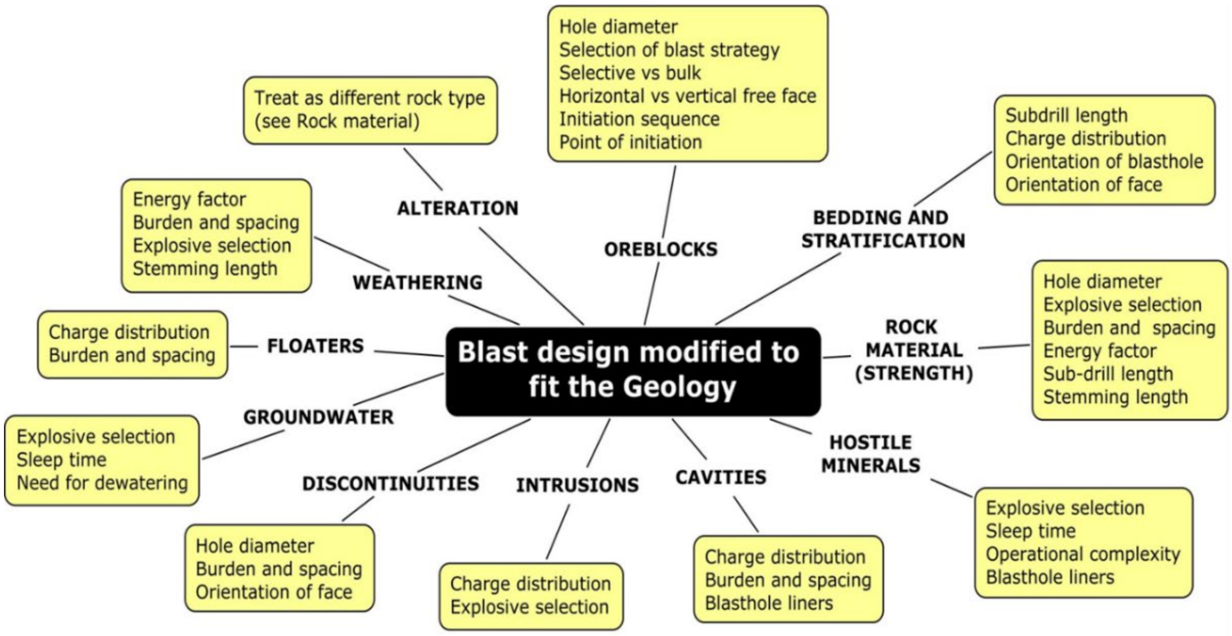



**FIG 7** – Design modifications to fit the geology (Little, 2011).

Some of these geological variations relate to safety (hostile minerals, cavities), strength (alteration, weathering, floaters, intrusions, rock material), geological structures (discontinuities, bedding and stratification), ore grade (ore blocks), and one relates to groundwater.

### Influence of engineering geology on pre-split and post-split design

Geology influence wall control blasting in a number of ways. The impact of geology and geological structures on pre-split and post-split design parameters include the following:

Spacing: The tensile strength of the rock mass and nature and the discontinuities networks will influence the pre-split and post-split blasthole spacing. When it comes to the tensile strength, Chen and Worsey (1986) investigated the influence of rock strength on the success of pre-splitting. It was concluded that tensile failure mechanism played a significant role in fracture initiation and development. By testing in rock of different strengths, it was further concluded that the maximum successful pre-split borehole spacing was inversely proportional to the Brazilian Disk tensile strength of the rock. Regarding discontinuities the spacing should decrease with increasing joint frequency because the number of interfering joints between the pre-split holes has a negative impact on the fracture propagation (increase attenuation).

Firing direction: The preferred pre-split direction, from a damage perspective is influenced by the nature of the discontinuities. Figure 8 illustrates the MCWSM model for vibration/damage pattern. For each of the 67 blastholes of this detonating cord-fired blast radiates according to a DFEM solution for a single blasthole in a material with a single 45° joint set (insert on far left of Figure 8). Firing along the general strike increases the wall vibration (lhs figure). So it is better to fire against the strike of parallel joints (rhs figure).

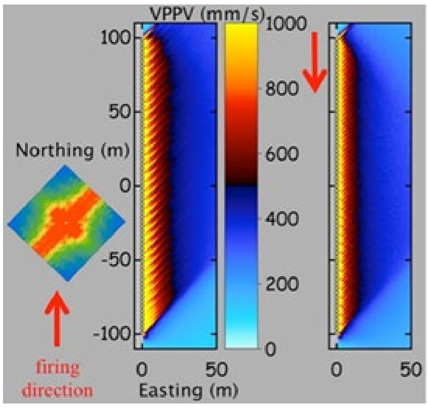



**FIG 8** – Model demonstrating the VPPV depends upon firing direction in structured rock masses (Blair and Little, 2019).

Drill hole deviation: This is an important aspect with regard to achieving the profile and damage objectives. Little (2011) indicated that drill hole deviation is strongly influenced by: blasthole diameter, blasthole length and inclination, and geology and rock mass properties. Also the penetration rate is controlled by the mineral composition and micro-fabric, eg porosity and quartz. It is also controlled by elastic/plastic behaviour, the mechanical rock properties, the rock mass conditions, and discontinuity networks present. Drill hole stability is controlled by the rock strength and sensitivity of the wall rock to the atmosphere, water and/or stress relief.

Performance: The pre-split and post-split performance (quality) is also strongly influenced by the geology and rock mass properties including the groundwater. It should be noted that groundwater has four important influences on wall control blasting. Firstly it changes the rock mass properties, secondly it makes implementation of the blast more difficult and thirdly it limits explosive selection, and finally it influences blast performance. Lewandowski, Luan Mai and Danell (1996) concluded the pre-split quality decreased with increased joint frequency and the pre-split quality decreased with an increase orientation from 5 to 30°. They also concluded that lower joint friction will reduce pre-split quality. Etchells, Sellers and Furtney (2013) used a Hybrid Stress Blast Model (HSBM) for post-split blasting. The model two subvertical joint sets with dip directions at ±50° and dipping at 80° to the face with a third set dipping 10° to the face. The detail of the remaining face shown in Figure 10 indicates the saw-tooth shape shown in the real situation in Figure 9 (Photo), which provided the researcher with some confidence in the model representation.

**FIG 9** – Photo of saw tooth face profile in a southerly direction (Etchells, Sellers and Furtney, 2013).

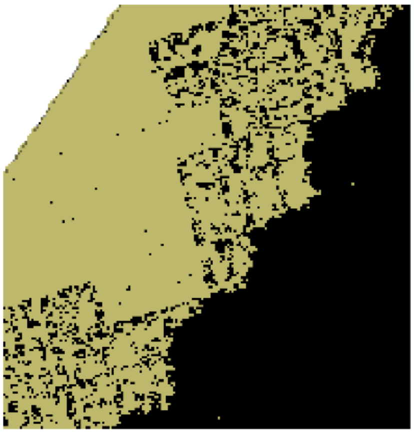


**FIG 10** – HSBM output showing saw tooth face profile (Etchells, Sellers and Furtney, 2013).

Design approaches: Despite what some people may think the design of a pre-split blast is not a simple matter. Table 2 discusses the various approaches to pre-split and post-split design. Generally two or more methods used together provide then best solution along with extensive performance monitoring (beyond the scope of this paper).

**TABLE 2**

Pre-split and post-split design approaches.

| Design approach | Pre-split design comments |
|---|---|
| Experience | • Can use experience and benchmarking from similar site as a design starting position. |
| Blast trials | • Trial and refinement can be used to fine-tune the pre-split design.<br>• Blast trials are needed to confirm designs developed by all other approaches. |
| Empirical methods | • Rule of thumb, for example spacing equals 8–13 hole diameters.<br>• Empirical equations based two dimensional static theory are strictly only valid for static loads but are able to give indication of spacing based on tensile strength of the rock and blasthole pressure. |
| Blast models | • MC waveform superposition – See Figure 8 for an example output.<br>• ELFEN/MBM/SoH modelling – provide guidance for un-stemmed height etc.<br>• HSBM/SMB modelling – see Figure 9 for an example of output. |

Blasting objectives: What are the objectives to be targeted? In a hard rock environment it may be to: achieve the design profile, achieve acceptable damage levels and/or provide a platform for slope steepening. What are the benefits expected? Improved safety by lowering the probability of rockfall, improved stability by reducing rock mass damage. Blair (2015b) showed that the vibration reduction due to pre-split formation was statistically meaningful and a reduction of 40 per cent was estimated. In a throw blasting coal environment the objectives may be to: reduce the likelihood of rockfall/slope failure hazard for mining underneath the face; provide a face that leads to faster dragline recovery during offline key digging or chop cutting; and/or enable a consistent front row burden in the next bench for engineering control (Lewandowski, Luan Mai and Danell, 1996).

## Wall damage from close blastholes

This discussion is applicable to all wall control blasting techniques. This is especially critical for the closest row of holes, but it is also important that the first buffer rows and the first production row of the blast pattern are not damaging the walls. At many mine sites, the charge weight/standoff-distance is set by a trial and refinement process over an extensive period of time. Marklund *et al* (2007) discussed such an approach used at Aitik Mine in Sweden. Figure 11 illustrates the predicted damage zones for a number of different charge diameters for strong and weak rocks. Figure 12 illustrate how this information can be used to predict damage for a wall control blast design.

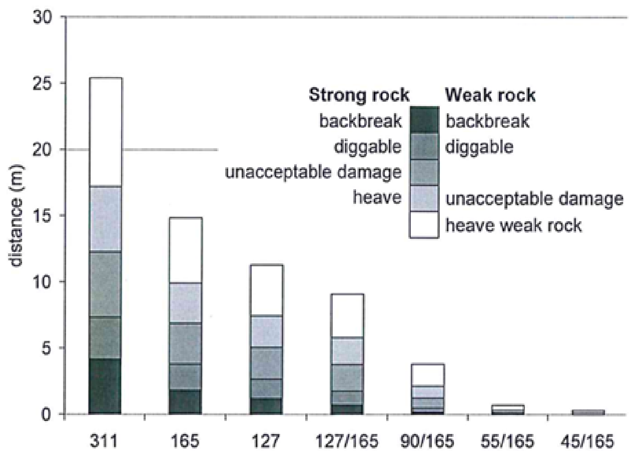



**FIG 11** – Bar chart of charge diameter and damage dimensions for the Aitik Mine (Marklund *et al,* 2007).

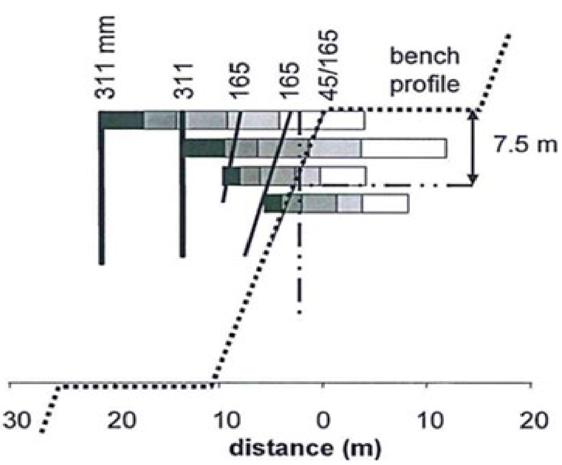



**FIG 12** – Damage prediction applied to a wall control blast design for the Aitik Mine (Marklund *et al,* 2007).

A more efficient alternative to the above approach currently exists. The new approach uses an ‘analytical dynamic stress blast model’ developed and reported by Blair (2015). That work describes the development of an Analytical Damage Model (ADM) based on an exact solution under the assumption of viscoelastic material. Figure 13 indicates that the ADM takes full account of the blast geometry, the explosive properties in terms of pressure-time loading, the rock mass elastic/ viscoelastic properties, and primer location(s).

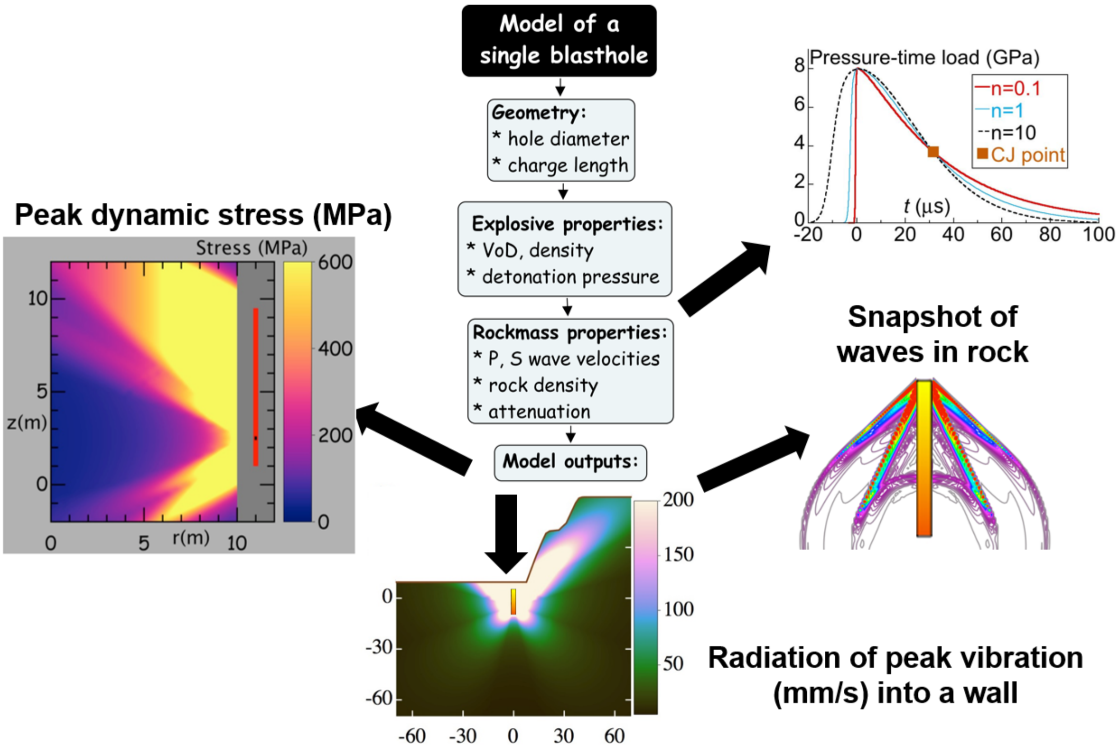



**FIG 13** – Analytical damage model for single blasthole (Blair and Little, 2013).

A very useful ADM output is a prediction of dynamic stress (MPa) radiation pattern for direct comparison with the unconfined compressive strength (MPa) of the local rock mass. Figure 14 shows such a plot, interestingly the zone closest to the primer has the least stress and as expected there is evidence of a run-up VoD on both side of the primer. In terms of damage the batter (face) will not be damaged but the berm (catch bench) may be, it depends on the rock strength.

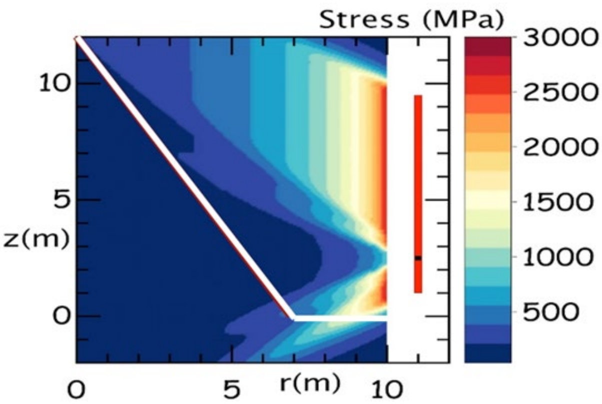



**FIG 14** – Dynamic stress radiation from a single blasthole (Source: Blair and Little, 2019).

According to the ADM, for a sufficient length of charge column in a given surrounding rock mass, the only way to alter the damage standoff distance is to: alter the diameter of the charge column; alter the explosive type; or use a pre-split or any combination of these design elements. It should also be noted that holes in the buffer 1 row (also termed toe or batter row), the buffer 2 row if it is significantly different and also Production 1 row must be examined for wall damage potential. Modelling indicated that the balance between charge weight and standoff is important but the balance between charge diameter and stand-off distance is even more critical with regard to damage. Figure 15 illustrates the influence on ADM calculated standoff distances for blasthole diameters ranging from 165 mm to 311 mm and rock properties ranging from 1–300 MPa (Unconfined Compressive Strength [UCS]). Reading from the chart, in Figure 15, the optimum standoff for a 165 mm blasthole is 2.5 m and for a 311 mm blasthole is 7 m (For a UCS of 150 MPa). It should be noted, that if the Fault zone material is weaker with a UCS of 50 MPa the optimum standoff distance for the 165 mm blasthole increases to 9.75 m, and line drill.

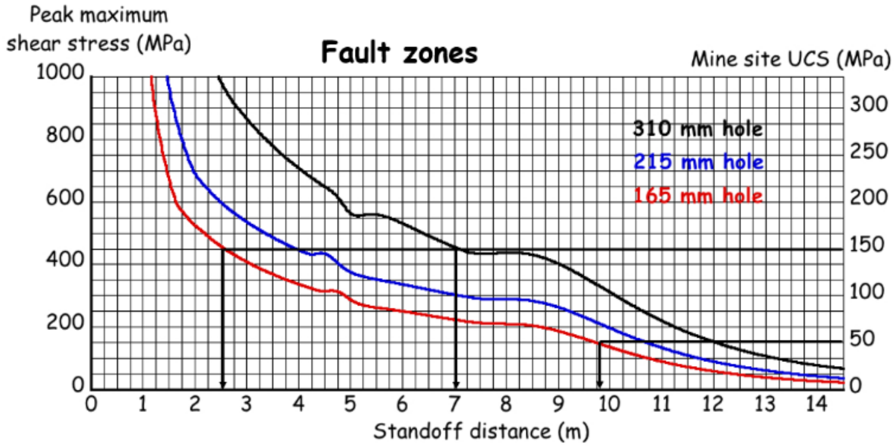



**FIG 15** – Three stand-off distance curves for various rock strengths (Source: Blair and Little, 2021).

The ADM model can be used to determine:

- Toe and collar standoffs for batter, buffer and production blasthole (rows) for any rock type.
- Standoff distances for any blasthole diameter for any rock strength properties (see Figure 15).
- Burden and spacing distances for any blasthole diameter for each rock type.
- Investigate air decking effects for a given rock type, geometry and explosive.
- Appropriate primer location(s) to direct dynamic stress radiation patterns away from weak zones.
- Angled blasthole radiation patterns for each rock type.

## Rock properties and explosive selection

The selection of the explosive type and hence properties is influenced by the blasting objective, the geology and groundwater regime. The amount of explosive required in terms of distribution of charge in space and time for optimal results is also influenced by the rock properties. In turn the rock mass stiffness and geometric design influences the explosive performance and the blasting results.

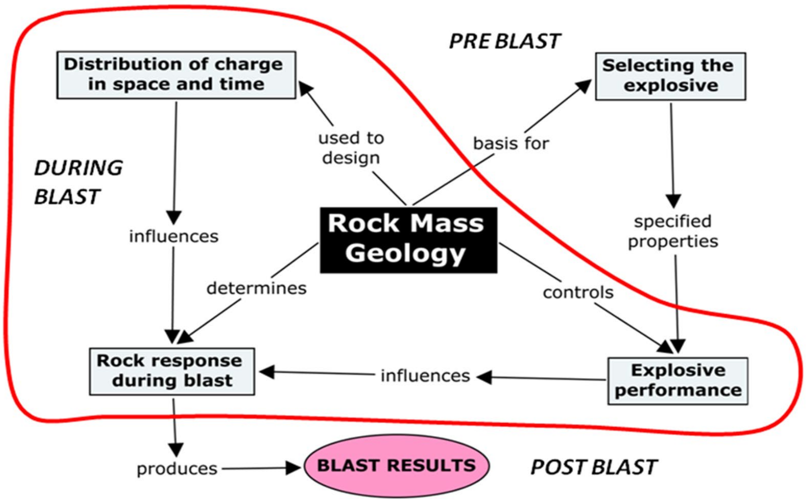



**FIG 16** – Geological controls on explosive selection and the explosive-rock interaction (Little, 2011).

Blasting results indicate that rocks respond differently to explosive loading rates (VoD). The following modelling examples illustrated in Figures 17 to 18 explains why this is the case. The ADM is used to examine the blast stress induced in hard, medium and soft rock. All the vertical blastholes have a diameter 114 mm and length 15 m. All blasted with ANFO at density 810 kg/m$^3$. The rock properties and the resulting explosive properties are listed from strong to weak. Seismic Q is the dimensionless viscoelastic attenuation factor. Figure 17 indicates that different strength rocks respond very differently to the same explosive charge. For strong rock no Mach waves are generated for ANFO charges and very poor fragmentation is expected. For medium strength rock an S Mach wave forms and fragmentation is expected to be reasonable for 2 to 3 m if the rock mass strength is less than 70 MPa. In the weak rock model both an S and a P-Mach wave form, giving rise to the best conditions for maximum fragmentation. However, due to the low seismic Q the stress zone greater than the 200 MPa stress zone is reduced to approximately 1 m from the blasthole axis.

*Strong rock* typical of very competent underground rock: $V_P$ = 7000 m/s, Poisson's ratio = 0.15, Density = 2500 kg/m$^3$, Seismic Q = 100. Then: VoD = 3736 m/s; $P_{VN}$ = 6.08 GPa; $P_{CJ}$ = 2.80 GPa.

*Medium rock* typical of competent open cut rock: $V_P$ = 4500 m/s, Poisson's ratio = 0.15, Density = 2000 kg/m$^3$, Q = 50. Then: VoD = 3446 m/s; $P_{VN}$ = 5.10 GPa; $P_{CJ}$ = 2.32 GPa.

*Weak rock* typical of some coal measures and weathered surface material: $V_P$ = 2500 m/s, Poisson's ratio = 0.15, Density = 1800 kg/m$^3$, Q = 30. Then: VoD = 3412 m/s; $P_{VN}$ = 4.99 GPa; $P_{CJ}$ = 2.34 GPa.

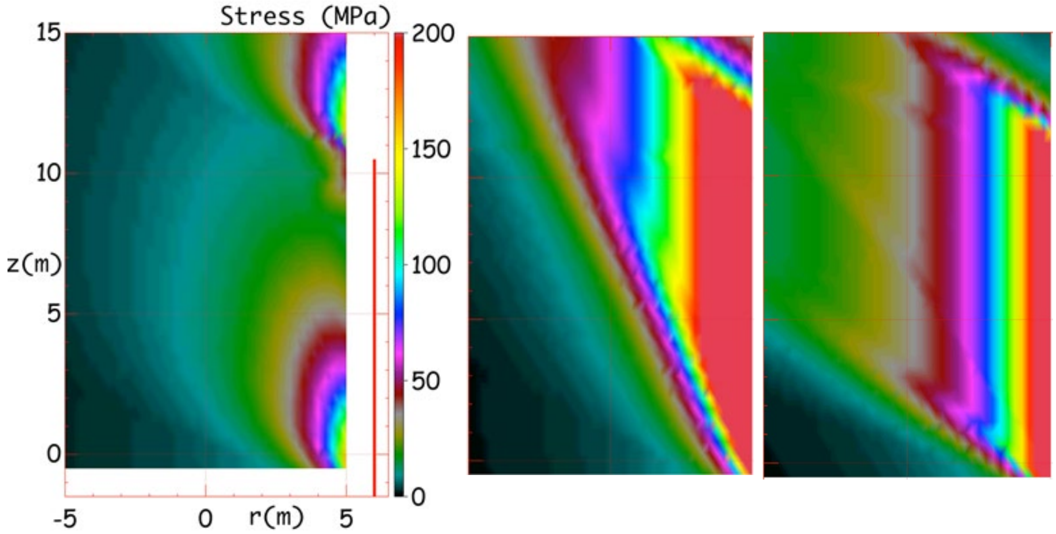



**FIG 17** – (a) Wave radiation for strong rock, Q100 (VoD < VP,VS) P-, S-waves decouple from VoD, No Mach waves. (b) Wave radiation for medium rock, Q50 (VS < VoD < VP), Forms S-Mach wave. (c) Wave radiation for weak rock, Q30 (VoD > VP,VS), Forms P and S-Mach waves. But low Q reduces stress (Source: Blair, 2011).

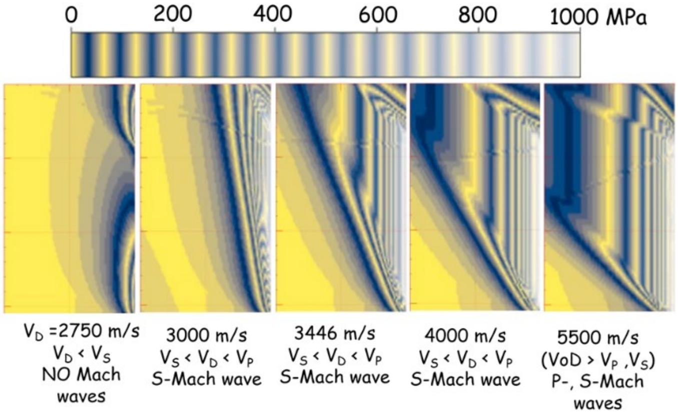



**FIG 18** – Development of Mach waves in medium rock with changing VoD (After Blair, 2011; using updated graphical presentation).

The selection of explosive type and hence properties is also influenced by the geology and the blasting objective. If the blasting objective is profile and damage control then explosive select, in terms of VoD, can be used to control the Mach waves (fragmentation/damage potential) generated. It should also be noted that the location of primers and the associated run-up zone can also be used to direct energy to where it is needed or away from weak or damaged zones.

## The Conceptual model

Blaster designer can also benefit from having a sound understanding of how different rock types respond to dynamic loading such as in a blast. A very good insight into this behaviour was formulated as a Conceptual model (Blair, 2015). The Conceptual model relates vibration, damage and rock mass condition. Figure 19 illustrates this simple model, in which identical spherical masses of steel are dropped from identical heights onto a flat steel surface as well as a flat level of sand. The model demonstrates two obvious facts. Firstly, for a fixed input energy, weak (damaged) ground will suffer more bulk motion than strong (undamaged) ground; secondly, the weak ground will significantly attenuate the resulting wave motion whereas the strong ground provides much less attenuation.

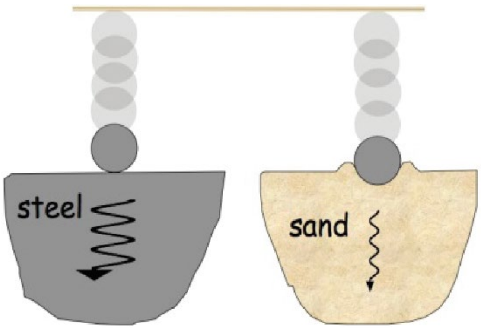



**FIG 19** – Conceptual model for bulk motion and vibration (Blair, 2015b).

Figure 20 encapsulates these ideas with regard to blasthole initiation in undamaged and damaged ground; the broad arrows indicate the physical displacement of the ground and the oscillatory motion shows the wave propagation. The indicated amplitudes of wave propagation and displacement are a direct consequence of the conceptual model.

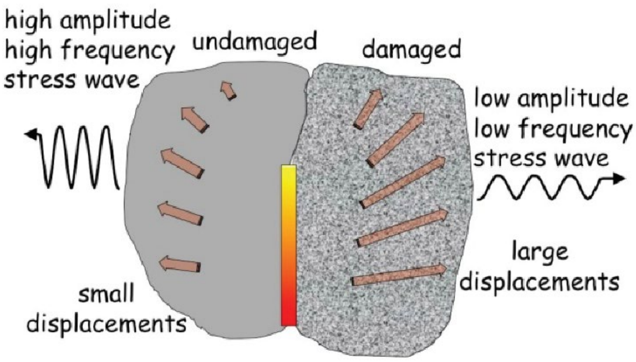



**FIG 20** – Conceptual model for blasthole initiating in different ground conditions (Blair, 2015b).

This simple conceptual model has been used to demonstrate:

- ground heave for a centre-lift blast
- the muck pile shape for weak zones in a strong rock mass
- why blasts in weak ground produces a lower vibrations than a blast in strong ground
- how power troughs form
- the concept of reverse firing.

## CONCLUSIONS

Based on the current study, the following conclusions can be drawn:

A Blast Management Framework developed has been used to put wall control blast design and analysis into context. The framework is simple and generic and its use by others is encouraged.

The BDR Classification Scheme has been developed to improve the blast design process. Blast design is the fundamental engineering control to obtain consistent output that achieves the results demanded by the relevant managerial and technical blasting objectives.

The design of wall control blasts must explicitly address both profile and damage control objectives. This could be achieved over a period of time by trial and refinement. However, the approach outlined in this paper, using blast modelling has proved to be quicker, less prone to misinterpretation, and importantly it allows for optimisation based on engineering geology.

The four main approaches to blast design are: experience based, blast trial based, empirical methods, and the use of blast models. In reality, a combination of these approaches is often used.

Pre-split and post-split blasting is not straightforward and for optimisation it requires a combination of: site investigation, design analysis (modelling) effort and field trials to assist with design refinement.

An Analytical Damage Model (ADM) based on an exact solution under the assumption of viscoelastic material can be used to determine optimum stand-off distances for blastholes close to final walls.

The explosive selection for particular rock mass properties and in particular the generation of Mach waves was demonstrated with a modelling demonstration.

Blast designer will benefit from an understanding of how rocks respond to blast loadings and a Conceptual model was presented to assist with that understanding.

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# Dredger drop impact on slope stability – Hazelwood Mine

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## ABSTRACT

ENGIE, the current owner and operator of the Hazelwood Open Cut coalmine, are in the process of rehabilitating the mine site in preparation for mine closure. Bucket wheel excavators (dredgers) used during coal mining operations are in the process of being demolished, which involves the controlled felling of dredger components using explosives. With the mine located in close proximity to the Morwell township and other critical public infrastructure, it is crucial that the use of explosives and the possible implications for mine stability is well understood. The felling of Large Mining Equipment (LME) will apply an impact load immediately to the underlying coal benches with the adjacent coal batters potentially at risk also of adverse slope stability impacts. This paper presents the 3D stability modelling methods employed to successfully assess the potentially adverse stability implications associated with LME collapse.

## INTRODUCTION

Hazelwood Mine began operation in the 1950s under the management of the State Electricity Commission of Victoria (SECV). Over the following decades the operation was privatised, with ENGIE being the current owners. The coal mining operation was ceased by ENGIE in 2017 and mine rehabilitation works commenced shortly after. The rehabilitation works included removal of the various Large Mining Equipment (LME) present on-site. At the time of closure, three of the largest Bucket Wheel Excavators (BWE) (Dredgers 9, 10 and 11, abbreviated to D9, D10 and D11) were located on the North Field Northern Batters (NFNB) of Hazelwood Mine (as shown in Figure 1).


**FIG 1** – Hazelwood Mine and locations of LME on the Northern Batters.

Two BWEs, D11 and D10, were felled using explosives on 15 and 28 October 2020 respectively. D9 has not been demolished as at February 2021. The controlled felling of D10 and D11 applied dynamic loading on the coal batters, with the potential to adversely impact batter stability. 3D modelling was undertaken on 'critical' stability sections (ie N10N, N11 and N12 – see Figure 1) of the NFNB domain of Hazelwood mine which coincided with the locations of the three bucket wheel excavators outlined above.

The intent of the modelling was to assess the potential for the felling activities to impact batter stability, and determine suitable treatment measures, as applicable.

The authors have worked with ENGIE over many years, conducting extensive studies undertaken by GHD (2019a, 2019b) and detailed assessments into the stability of brown coal batters informing the ENGIE (2017) Ground Control Management Plan (GCMP) and rehabilitation and closure strategies. This has included assessments of the properties, behaviour, terminal batter geometry and variation of the coal and interseam geological units present within the mine, and the influence of groundwater on them. This paper presents the results of the stability analyses conducted and the findings post-demolition.

# METHODOLOGY

## LME loading

The impact loads caused by the felling of the LME onto the brown coal batters were required for the batter stability assessment. The collapse sequence modelling for the LME is summarised in Table 1:

**TABLE 1**

Hazelwood Mine LME demolition parameters (Delta Group, 2019).

| Item | Assessment | D9 | D10 | D11 |
|---|---|---|---|---|
| 1 | Maximum energy impact (MJ) from centreline of rotation | 58.3 | 46.4 | 44.6 |
| 2 | Maximum impact load (MN) | 49.6 | 39.4 | 37.9 |
| 3 | Impact velocity lower counterweight chords on impact (m/s) | 11.8 | 11.2 | 10.8 |
| 4 | Location where maximum energy impact occurs (counterweight boom side from centreline of rotation) (m) | 24.9 | 31.9 | 32.5 |
| 5 | Location where maximum energy impact occurs (from bucket wheel boom towards top of batter on north side) (m) | NA | 9.8 | NA |

Note: NA – Not Applicable.

The impact loads from the proposed LME collapse on the coal batters were modelled for the purposes of assessing stability implications on the coal batters, using the following loading arrangements:

- **Static loads** from the LME (ie BWEs) based on technical specification drawings for dredgers D9, D10 and D11. This load was modelled as a uniformly distributed load underneath the dredger tracks.
    - Dredger D9 – applied ground bearing pressure: 106 kPa
    - Dredger D10 – applied ground bearing pressure: 113 kPa
    - Dredger D11 – applied ground bearing pressure: 113 kPa.
- **Impact loads** from the largest total force applied by the collapsed components (ie counterweight boom, counterweight boom ballast, mast etc). This was modelled as a line load over the impact area, in addition to the design static weight of the entire dredger, to produce the most conservative ground movement assessment.

Figure 2 displays the application of loads in the 3D model.

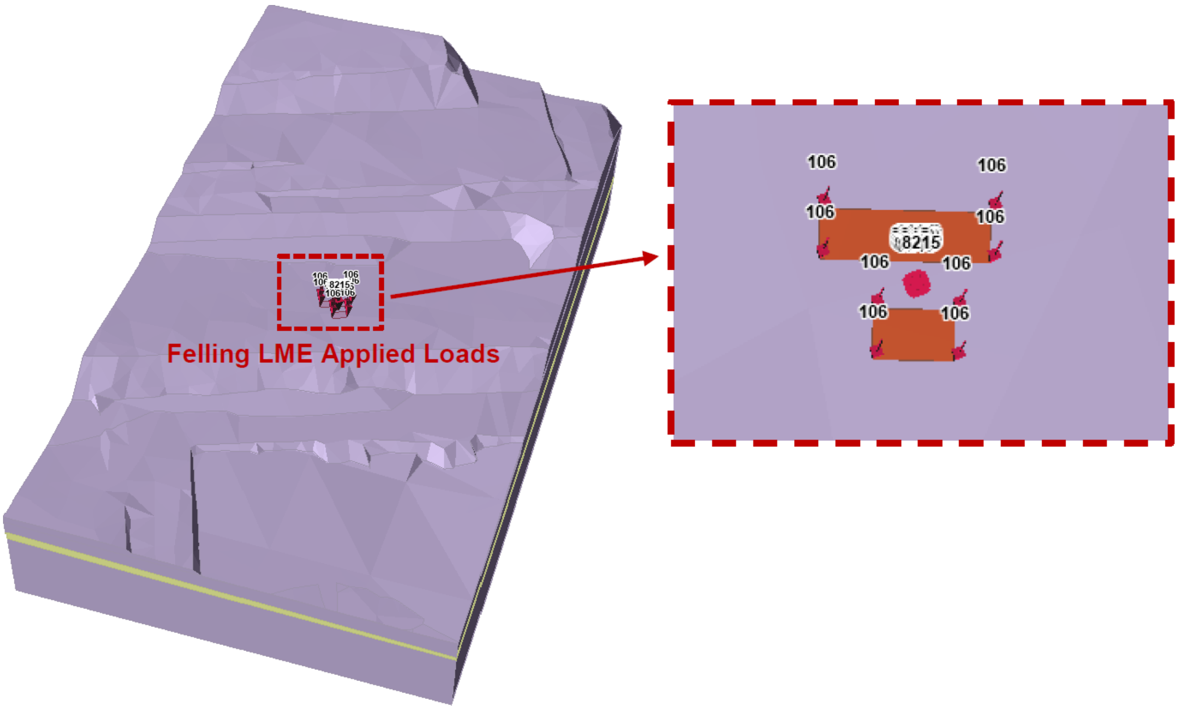


**FIG 2** – Impact loads for Dredger D9.

### *Material properties*

The material parameters (see Tables 2 and 3) adopted for the assessment were statistically derived based on historical and contemporary investigation and laboratory testing completed by the authors over the course of rehabilitation studies for the Hazelwood Mine. A schematic profile of the geological units is presented in Figure 3.

**TABLE 2**

M1 coal shear strength – Mohr Coulomb parameters.

| **Material** | **Unit weight (kN/m$^3$)** | **Cohesion c' (kPa)** | **Friction angle ϕ' (°)** |
|---|---|---|---|
| Coal | 11.5 | 150.0 | 35.0 |
| Residual coal (ie joints) | 11.5 | 0.0 | 35.0 |

**TABLE 3**

M1 clay interseam shear strength parameters.

| **Material** | **Unit weight (kN/m$^3$)** | **Shear strength function** |
|---|---|---|
| M1 interseam | 18.0 | $\tau = \sigma_n \tan\left(18 + 10.8log_{10}\left(\frac{314.5}{\sigma_n}\right)\right)$ |

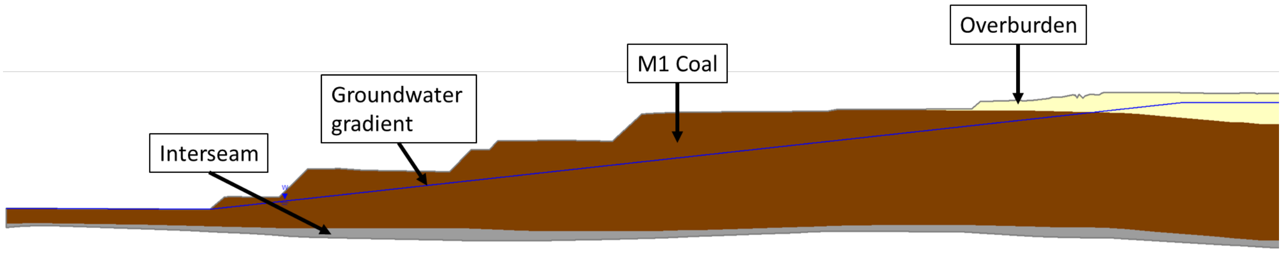


**FIG 3** – Schematic of geological units.

## 3D model methodology

A 3D surface (in the form of a Digital Terrain Model (DTM file)) of the NFNB domain was created in Geovia Surpac 6.7 (Surpac) using 1 m contours. Surpac utilises standard triangulation techniques to create a surface mesh from the contours, however, residual defects (eg breaks and irregular shaped polygons) can remain. Rocscience Slide3 modelling software is equipped to service the importation of DTM files and provides a function to remove/repair residual defects. The repaired surface layer is presented in Figure 4.

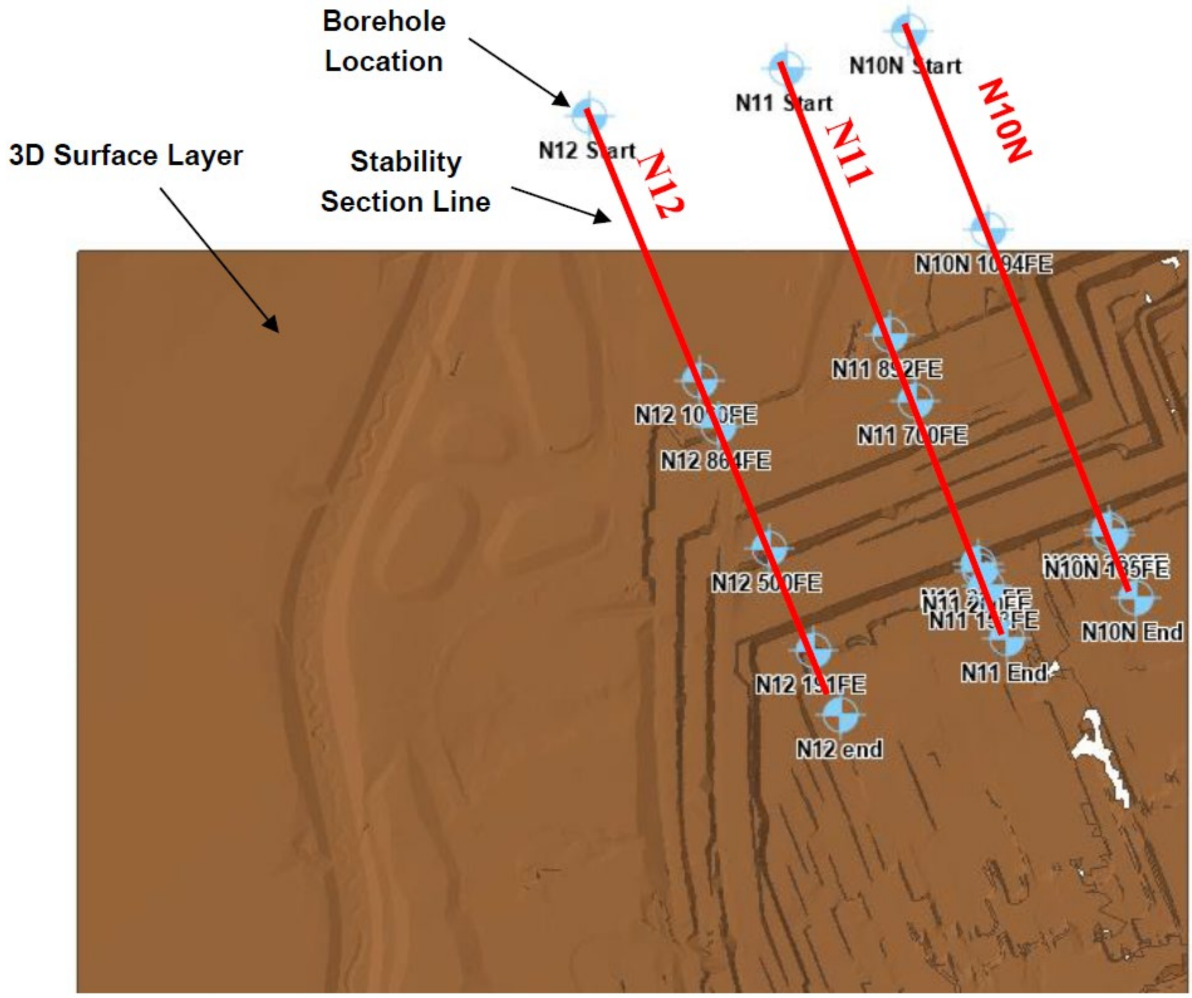


**FIG 4** – Plan view of Hazelwood Mine NFNB depicting 3D surface layer.

In order to incorporate a 3D geological model of the NFNB, subsurface layers (ie M1 clay interseam, M1 coal and overburden) were required. This was undertaken using Slide3 as follows:

- Definition of stratigraphy boundaries (ie depth to top of coal) using vertical borehole data obtained from 2D models along key stability sections of the NFNB and adjacent domains.
- Importation of borehole information into the model for at least six (6) boreholes along each key stability section.
- Interpolation of the subsurface layers within the localised areas of consideration (ie Sections N10N, N11 and N12) were applied using imported borehole information.

In addition to stratigraphic data, it is possible to assign the depth to water table and therefore the groundwater profile, for each borehole.

The interpolated subsurface geology and water table layers are depicted in Figure 5.

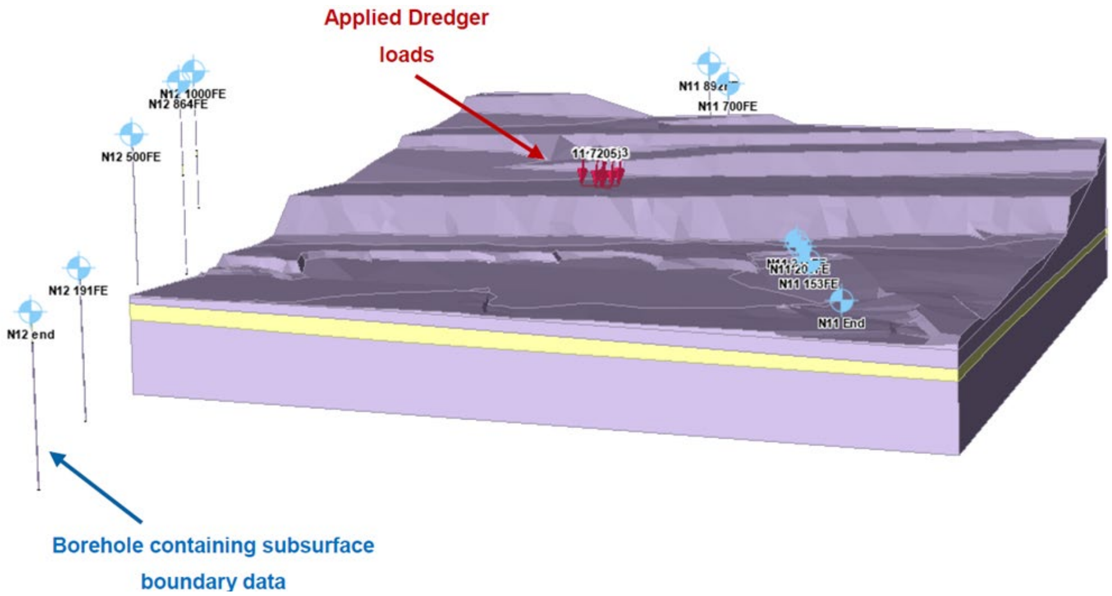


**FIG 5** – 3D model section (Dredger D11) constrained by external boundaries.

### *Definition of calculation boundaries*

In order to undertake stability calculations for the dredger collapse, an external boundary was first defined within the overall model extents. The external boundary created a closed system and provided a method to reduce the extent of the model being analysed. It is typically acceptable to have localised boundary conditions within a broader model extent, provided that the (localised) boundary limits do not impact the results of the stability assessment (ie overlook the critical surface(s)). The external boundary (see Figure 6) for each of the proposed LME collapse sequences was restricted as follows:

- 200 m vertically
- 300 m laterally (ie either side of the dredger centre line).

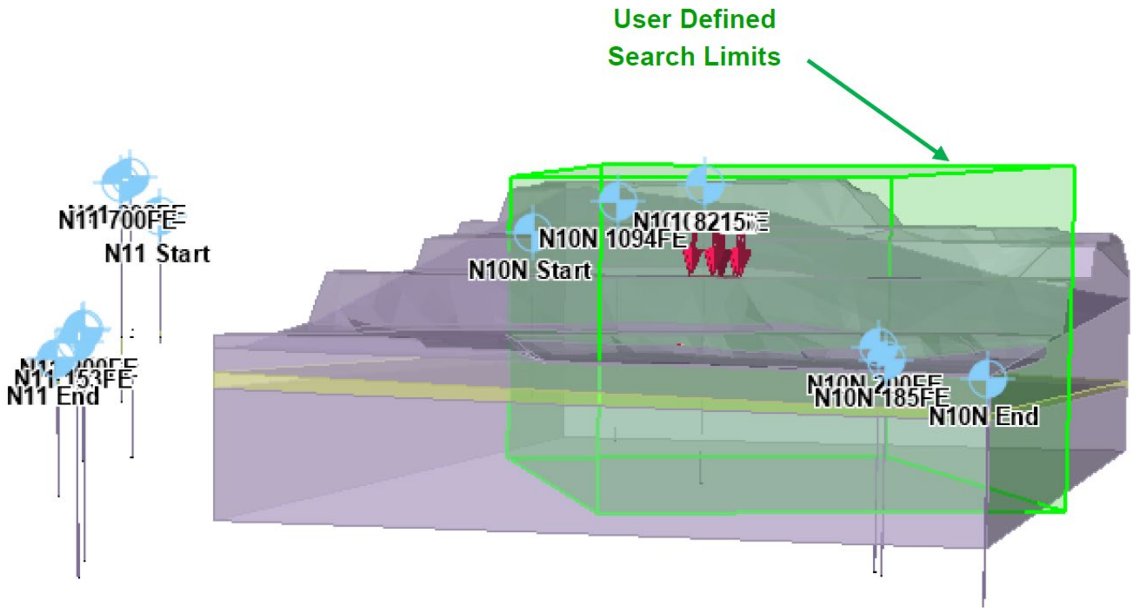


**FIG 6** – 3D model depicting search limits around Dredger D9 (along section N10N).

In addition to the above, a focused search was employed by adjusting the model search limits to approximately 100 m either side (laterally) of each dredger location (refer Figure 4).

Three-dimensional stability modelling was performed using Rocscience's Slide3 2019 software package.

## Surface safety maps and critical slip surface search

### *Surface safety maps*

Surface safety maps are used to represent areas of potential weak zones (see Figure 7), where areas highlighted in red indicate an increased likelihood of instability and those highlighted in blue a reduced likelihood. Figure 7 depicts an example, where it can be seen that the induced felling of D10 resulted in an increased likelihood for instabilities in a localised zone directly below the Point of Impact. The surface safety maps are intended to depict the zones of high and low risks.

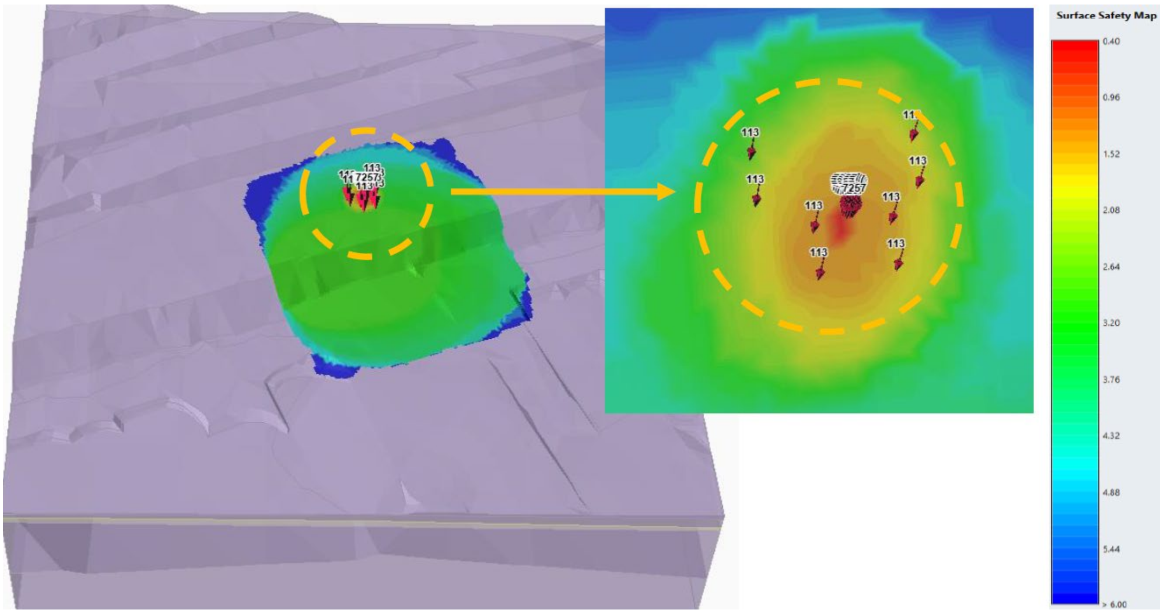


**FIG 7** – Surface safety map (ellipsoid) – Dredger D10.

### *Critical slip surface determination*

In calculating critical instability modes, the impact of each collapse sequence using the ellipsoid (ie deep seated, pseudo block sliding mechanism) and wedge (localised unravelling of coal blocks) surface functions were assessed. Critical slip surfaces for the ellipsoid function were calculated using the Cuckoo search method and surface altering optimisation (see Figure 8).

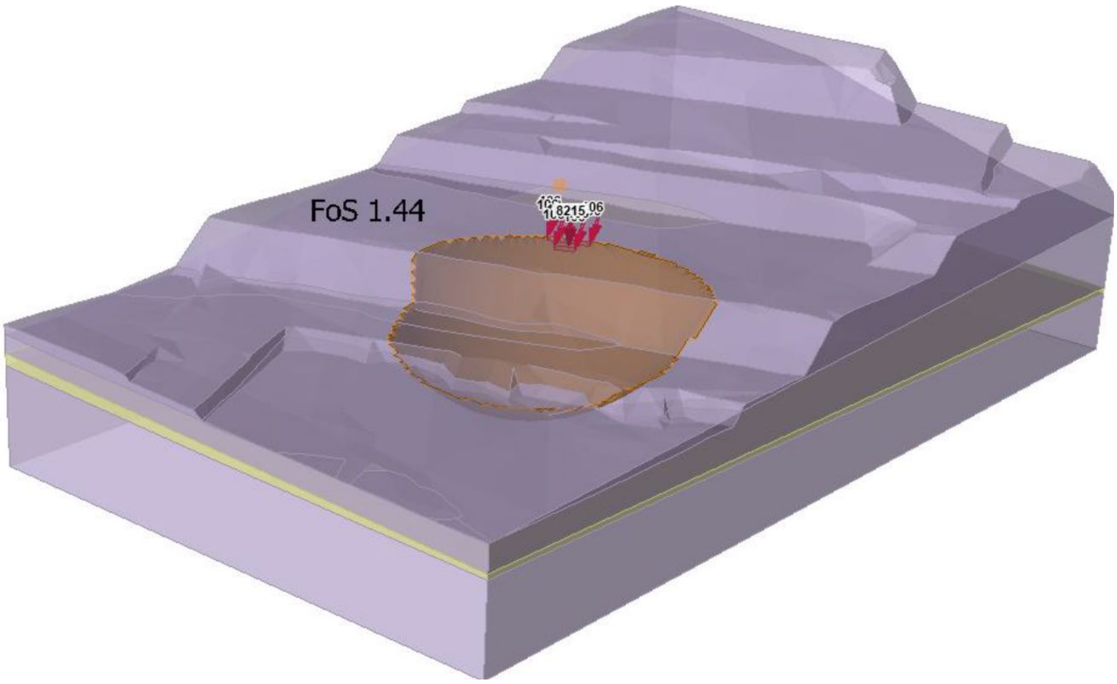


**FIG 8** – Ellipsoid instability surface – Dredger D9.

In order to generate a wedge surface (see Figure 9), geological defect information mapped within the respective coal batters was incorporated. The critical orientations of the defect sets are summarised in Table 4.

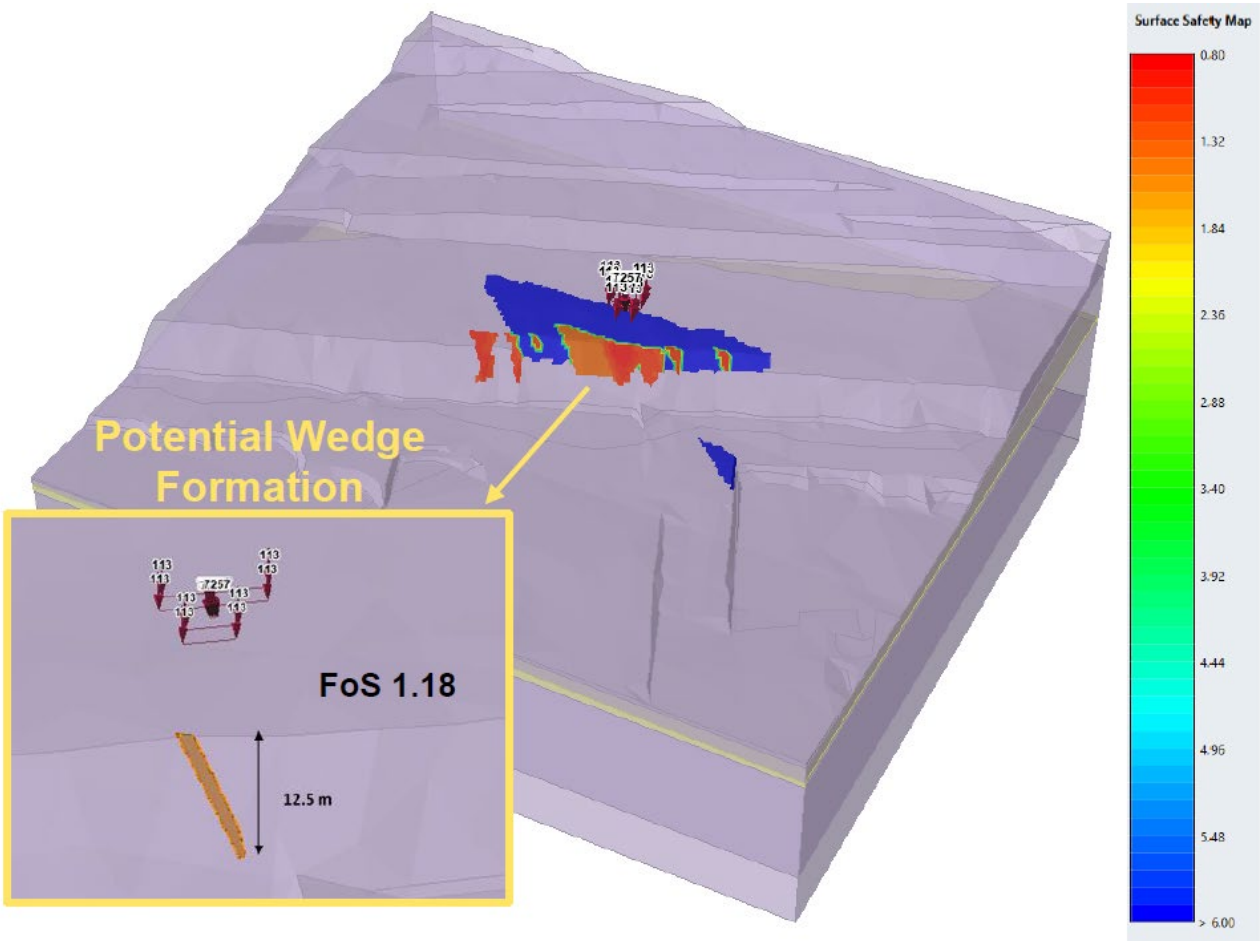


**FIG 9** – Wedge instability formations – Dredger D10.

**TABLE 4**
Summary of joint set properties.

| ID | Dip angle (°) | Dip direction (°) |
|---|---|---|
| Joint Set 1 | 87 | 42 |
| Joint Set 2 | 49 | 177 |

### Nominated design acceptance criteria

The nomination of a suitable design acceptance criteria for impact load assessment can be challenging. Accordingly, the authors undertook a literature review to determine suitable precedents, and guidance was provided in Guidelines for Open Pit Slope Design (Read and Stacey, 2009).

Subsequent to the above, a resultant Factor of Safety (FoS) of greater than 1.1 was nominated for the impact loading assessment of the LME collapse, given the temporally restricted nature of the felling activities.

## RESULTS OF MODELLING

### Dredger D9

Presented in Table 5, Figures 10 and 11 are the results of the 3D stability modelling for the demolition of LME D9. The results are provided for both a deep-seated instability (ellipsoid) and local instability (wedge).

**TABLE 5**

Summary of stability analysis results.

| Dredger number | Factor of Safety (LME collapse) – FoS | | Design acceptance criteria – FoS |
|---|---|---|---|
| | Ellipsoid | Wedge | |
| D9 | 1.44 | 0.89 | 1.10 |

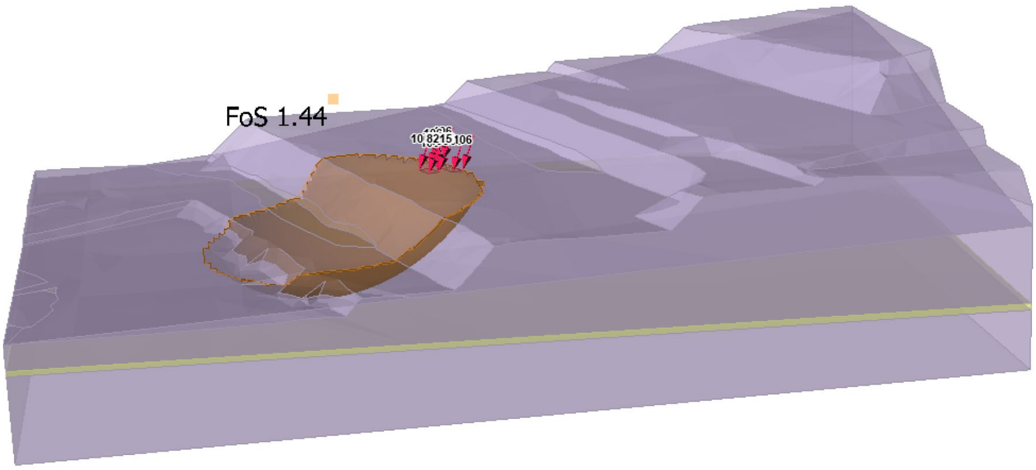


**FIG 10** – Critical slip surface (ellipsoid) – Dredger D9 (FoS = 1.44).

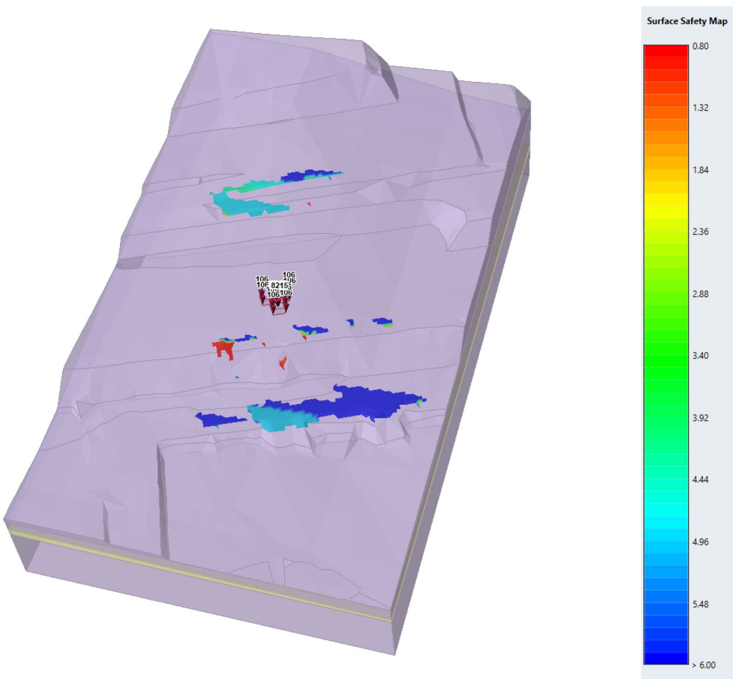


**FIG 11** – Potential wedge surface – Dredger D9 (FoS = 0.89).

Results of the 3D stability analysis indicate that:

- When large scale (deep seated – ellipsoid) and wedge instabilities were calculated, the design acceptance criteria (FoS 1.1) was satisfied for the felling of LME D9.
- There are some minor wedges with FoS less than 1.1. However, these represent volumes less than 15 $m^3$ and were considered immaterial. Therefore, substantively, the DAC was satisfied.

## Dredger D10

Presented in Table 6, Figures 12 and 13 are the results of the 3D stability modelling for the demolition of LME D10. The results are provided for both a deep-seated instability (ellipsoid) and local instability (wedge).

**TABLE 6**

Summary of stability analysis results.

| Dredger number | Factor of Safety (LME collapse) – FoS | | Design acceptance criteria – FoS |
|---|---|---|---|
| | Ellipsoid | Wedge | |
| D10 | 1.51 | 1.18 | 1.1 |

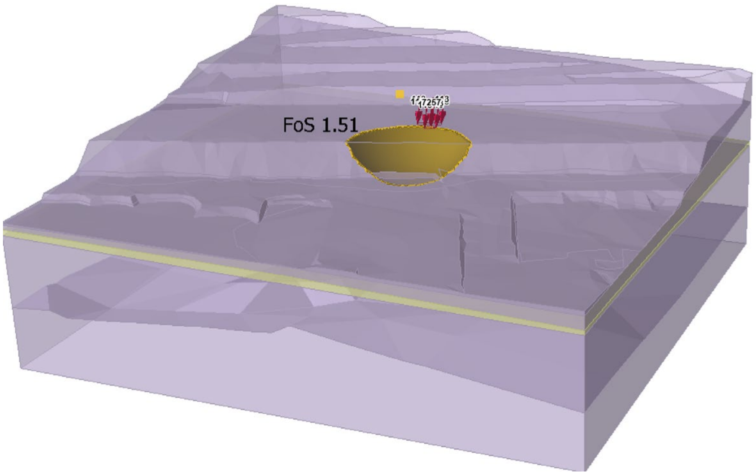


**FIG 12** – Critical slip surface (ellipsoid) – Dredger D10 (FoS = 1.51).

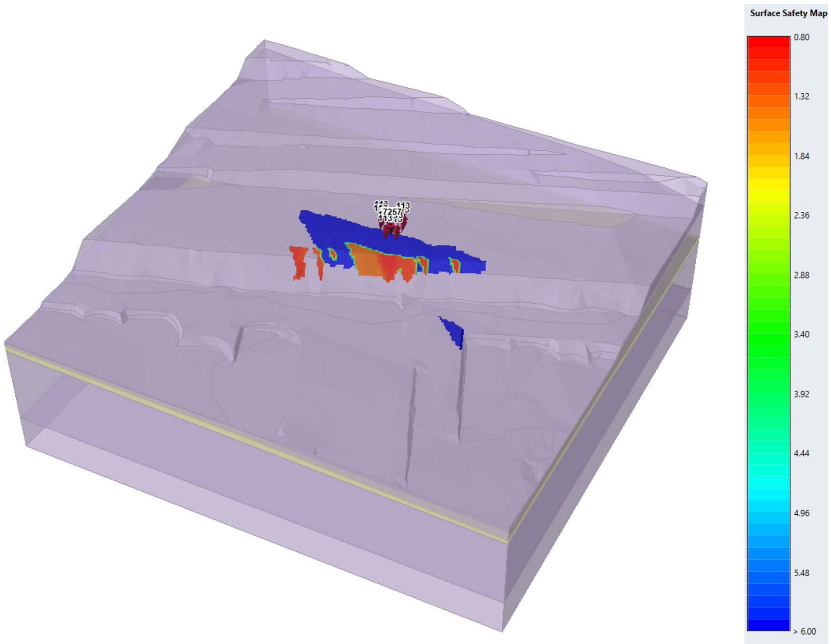


**FIG 13** – Potential wedge surface – Dredger D10 (FoS = 1.18).

Results of the 3D stability analysis indicate that:

- When large scale (deep seated – ellipsoid) and wedge instabilities were calculated, the design acceptance criteria (FoS 1.1) was satisfied for the felling of LME D10.

## Dredger D11

Presented in Table 7, Figures 14 and 15 are the results of the 3D stability modelling for the demolition of LME D11. The results are provided for both a deep-seated instability (ellipsoid) and local instability (wedge).

**TABLE 7**

Summary of stability analysis results.

| Dredger number | Factor of Safety (LME collapse) – FoS | | Design acceptance criteria – FoS |
|---|---|---|---|
| | Ellipsoid | Wedge | |
| D11 | 1.61 | 1.19 | 1.1 |


**FIG 14** – Critical slip surface (ellipsoid) – Dredger D11 (FoS = 1.61).

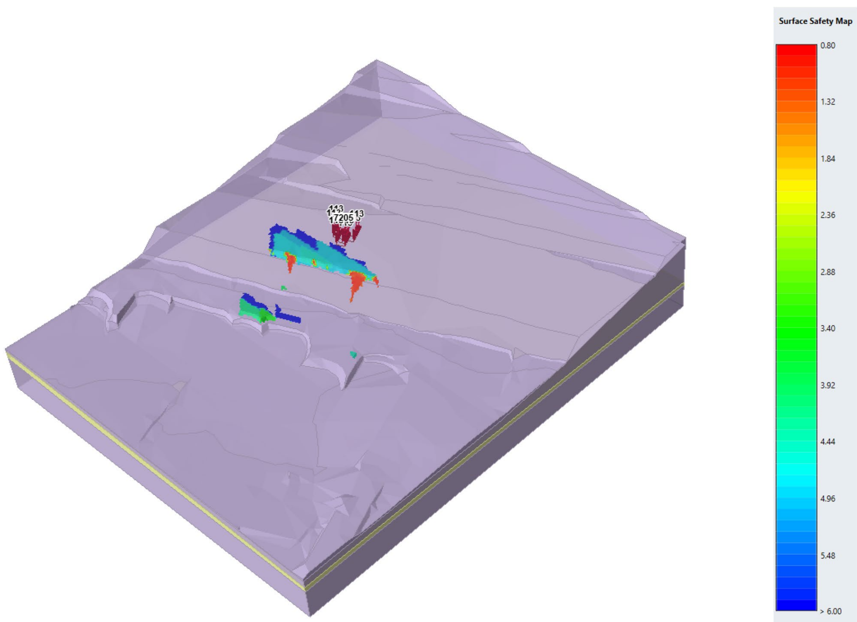


**FIG 15** – Potential wedge surface – Dredger D11 (FoS = 1.19).

Results of the 3D stability analysis indicate that:

- When large scale (deep seated – ellipsoid) and wedge instabilities were calculated, the design acceptance criteria (FoS 1.1) was satisfied for the felling of LME D11.

## Discussion of results

The key findings from the 3D modelling are summarised below:

- Surface safety maps considering deep seated instabilities indicate that some localised effects surrounding the point of Impact are likely to occur, which are expected to be in the form of indentation due to the weight of collapse.
- There is an increased likelihood for wedge instabilities to manifest as a result of dredger demolition, where FoS for wedge instabilities were all calculated to be lower than the deep-seated instabilities. Wedge instabilities for D10 and D11 were found to satisfy the design acceptance criteria (FoS 1.1), whereas the FoS for D9 was calculated to be less than the design acceptance criteria (FoS 0.89). The potential wedge instability calculated for D9 is less than 15 $m^3$. This potential volume is not considered to be a significant stability implication.
- Owing to the outcomes of the 3D stability modelling, apart from the formation of minor wedges that could dislodge from the impacts of the felling, no major instabilities are anticipated. However, this identified the potential for surface indentations to result. Accordingly, a loose layer of sand (0.5 m thickness) was placed to cushion the felling zones.

# POST-BLASTING OUTCOMES

Blasting was carried out on 15 and 28 October 2020, subsequent to which a rigorous geotechnical inspection was undertaken, and the following outcomes were noted:

- Prism survey undertaken prior to and following the impact did not indicate any significant batter movements.
- Real-time GPS GNSS units also did not indicate any adverse or significant batter movements.
- Visual inspections of the dampening pads installed beneath the drop points remained in relatively good condition (ie no significant/unintended damage to the pad – see Figures 16 and 17).
- Long-term monitoring of proximate prisms has not indicated any significant movements, except those typically expected during the mine rehabilitation phase.


**FIG 16** – Pre-demolition inspection – Dredger D11.


**FIG 17** – Post-demolition inspection – Dredger D11.

## CONCLUSIONS

This paper outlines how 3D stability modelling was utilised to assess the potential batter slope stability implications as a result of a relatively complex task of LME felling, which imparts both static and dynamic loads.

It was important to ensure that the process was adequately modelled, so as to determine, with confidence, any adverse implications as a result of felling activities, as they could have resulted in unforeseen ground movements, resulting in potentially adverse stability outcomes.

The process undertaken by the authors demonstrated how this complex mechanism could be modelled, to arrive at recommendations that were implemented in the field to result in a successful outcome post felling.

## ACKNOWLEDGEMENTS

The authors thank ENGIE for providing the opportunity to publish this paper and GHD for providing the necessary resources. In particular, the authors would like to give thanks to Mr Martin Slavkay and Delta Group for their insight and guidance during the planning and execution of this work.

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# Back analysis of pit slope performance in a saturated alluvial sediment

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## ABSTRACT

The material properties and hence performance of saturated alluvial sediments slope can be incorrectly forecast if the water content is not preserved, and materials properties not correctly determined. The occurrence of saturated sediments in the Western Australia (WA) Goldfields, while not common, should be a significant investigation focus. Geotechnical engineers new to the area may be unaware of this and hence the importance of correct testing and determination of the material strength. Application of appropriate analysis to determine stable pit slopes to an acceptable level of risk, is not possible unless the appropriate strength and hydrogeological parameters have been established at the outset.

A back analysis is presented of a pit that had extensive slope failures in alluvial clay-rich sediments with discussion on the approach adopted by the original design engineers to assess pit wall stability and consequent poor slope performance. Learnings from a full pit 3D back analysis are presented, with an approach to geotechnical and hydrogeological investigations and analysis for slopes in similar materials.

## INTRODUCTION

Low permeability, saturated, clayey sediments pose challenges in-pit slope design, which if not properly accounted for can lead to undesirable outcomes. Clay-rich lacustrine sediments deep in the overburden sequence are not common in the WA Goldfields and as a result design engineers may not be fully aware of the risks posed to pit stability. This is especially the case where engineers have a mining geotechnical background rather than civil.

A case study is presented, of a pit with such an example of a deep, saturated clay-rich layer which has dictated the performance of the pitwalls. The test work and analysis methods used by the original pit design engineers are presented along with a summary of pit slope performance.

Results of a back-analysis of the pit design using 3D inelastic finite element methods (not available to the original study) are presented and conclusions drawn about the results. Potential improvements from the learnings of the study are proposed.

## BACKGROUND

### Geology

The geology of the open pit area comprises 5–15 m of laterite cover, underlain by up to 40 m of transported alluvial sediments (Figure 1). These sediments comprise gravels, sands, silts, clays, siltstones, claystones and ironstones. Beneath the sediments, the Archean bedrock comprises east-dipping sequences of metasediments, mafic and ultramafic rock types. A transitional weathered zone exists from the base of the alluvial sediments to a depth of ~80 m below surface.

The geotechnical domain of concern for this study are the alluvial sediments which occur down to depths of ~55 m from surface, and in particular a lacustrine clay layer near the base of the sequence Figure 1.

The outcrops of the respective geotechnical domains on the original pit design are shown in Figure 2.

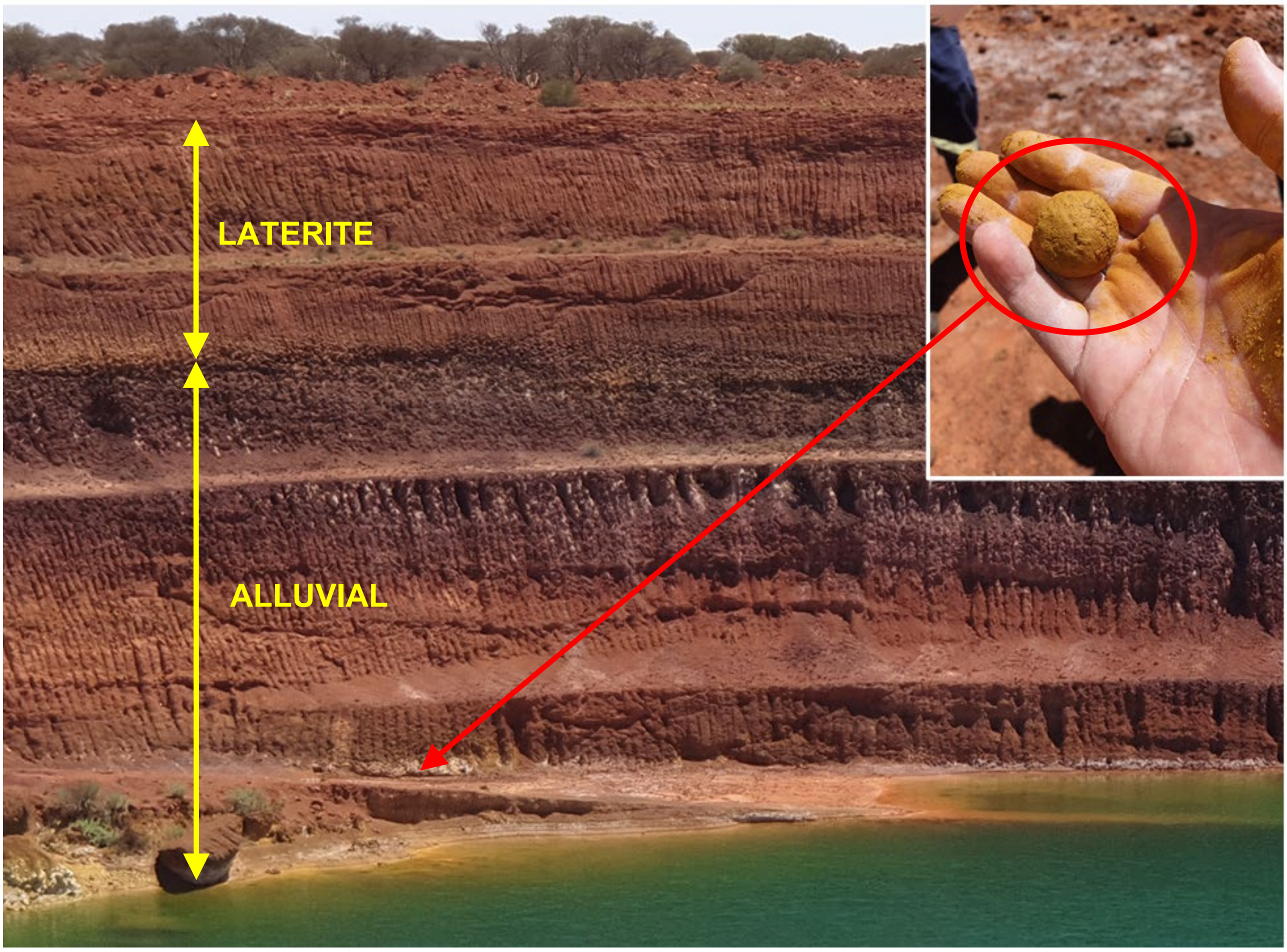



**FIG 1** – Deep saturated clay layer near the top of bedrock.

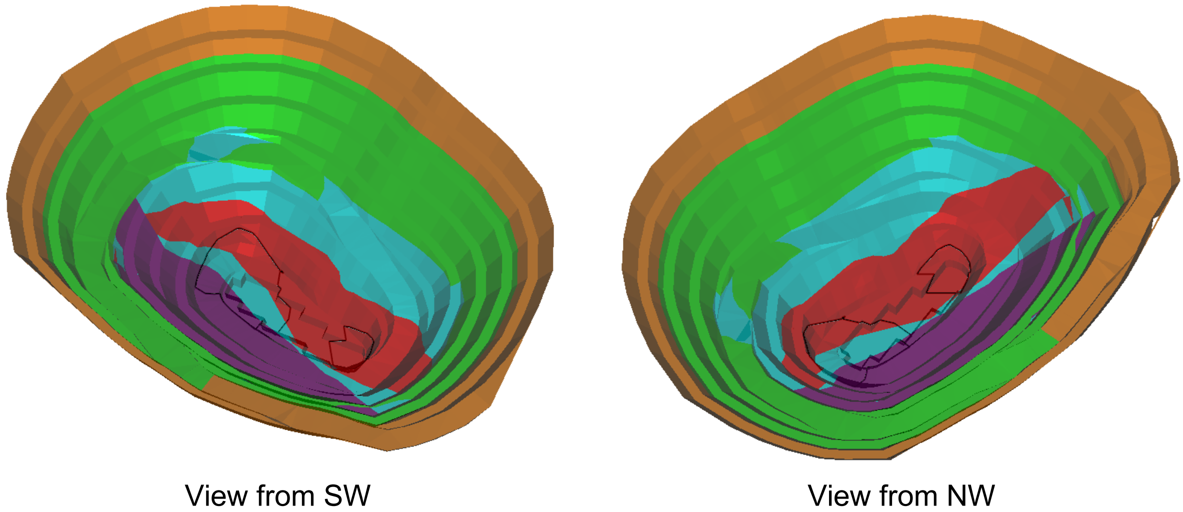



**FIG 2** – Geotechnical domain outcrops on pit design (brown = laterite, green = alluvials, red = ore, purple = ultramafic, cyan = mafics and sediments.

## Hydrogeology

Hydrogeological field investigations that took place prior to mining included:

- drilling of eight exploration holes to be used as monitoring bores, within the area of the pit
- installation of monitoring bores to the north and south of the pit, to measure groundwater levels
- permeability testing of all ten bores using airlift or slug testing methods.

The field measurements and monitoring indicate a depth of the water table coinciding with the base of the laterite domain.

Permeability testing focused on the fractured bedrock rather than the alluvial sediments and thus the values of between 6.8e-7 m/s and 5.0e-9 m/s obtained are not representative of this critical domain.

Horizontal drain holes were recommended in the original design study, but it is not apparent that they were installed and if they were, they were not effective in controlling instability.

Postmining monitoring of the installed piezometers around the pit indicates that the piezometric pressures recorded have recovered at a very slow rate in the years subsequent to cessation of mining (Figure 3). These observations support an assumption of low permeability for the alluvial sediments.

In the original geotechnical design study, slopes were assumed to be depressurised for an arbitrary distance of 20 m from the pitwalls. This was based on the premise that a combination of horizontal drain holes and pit dewatering would achieve this condition.

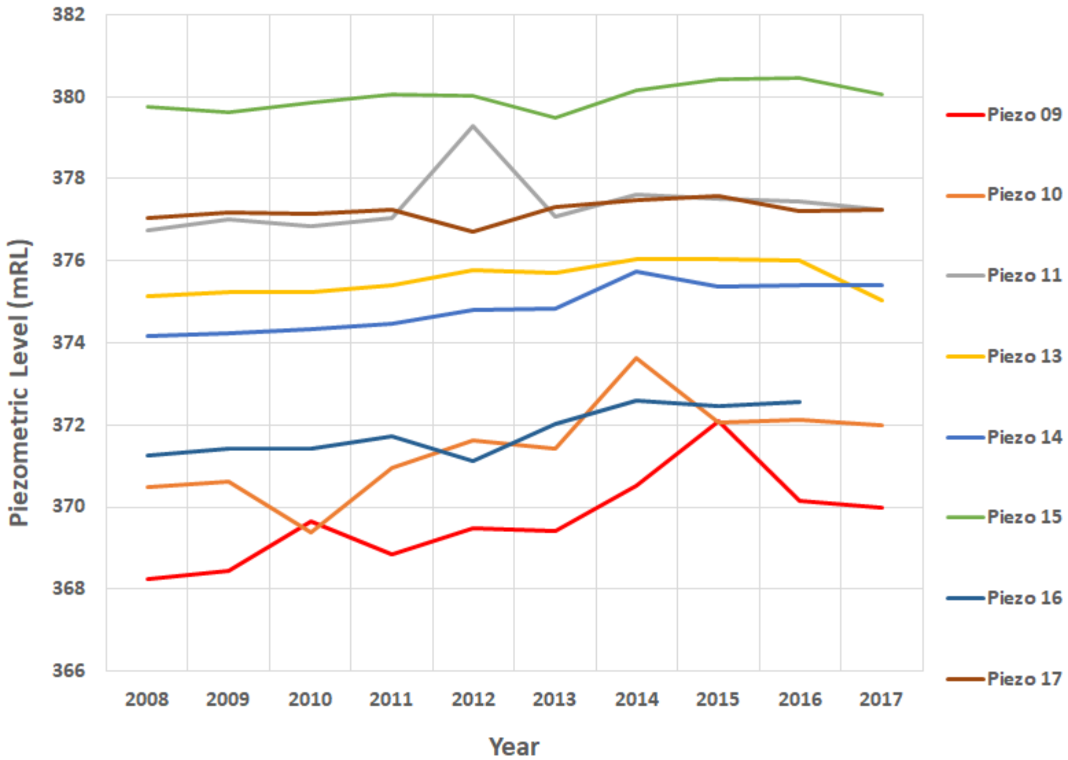



**FIG 3** – Piezometric levels recorded in monitoring bores installed around the pit periphery. The surface elevation is 398 mRL and the final pit depth is ~450 mRL.

## Geotechnical data

Three sets of laboratory testing focused on the alluvial sediments, with no testing carried out in the laterites, nor in the transitional and fresh bedrock.

The first was of five multi-stage direct shear box tests were carried out on what appear to have been 'saw-cut' surfaces on samples selected from core which apparently had not had the natural moisture preserved by sealing at the drill rig. Samples appear to have been tested 'dry', without inundation or consolidation during the testing process. Consequently, the results obtained from these tests are considered highly questionable for use in slope stability design and are ignored.

The second was two consolidated, drained multistage triaxial tests were carried out in addition to the above shear-box tests and the results presented in Figure 4 and Table 1.

The third was a further two direct shear tests carried out at another laboratory and involved inundating and consolidating the samples and shearing at a slow rate for drained strength parameters. The results of this test program are presented in Figure 5 and Table 1.

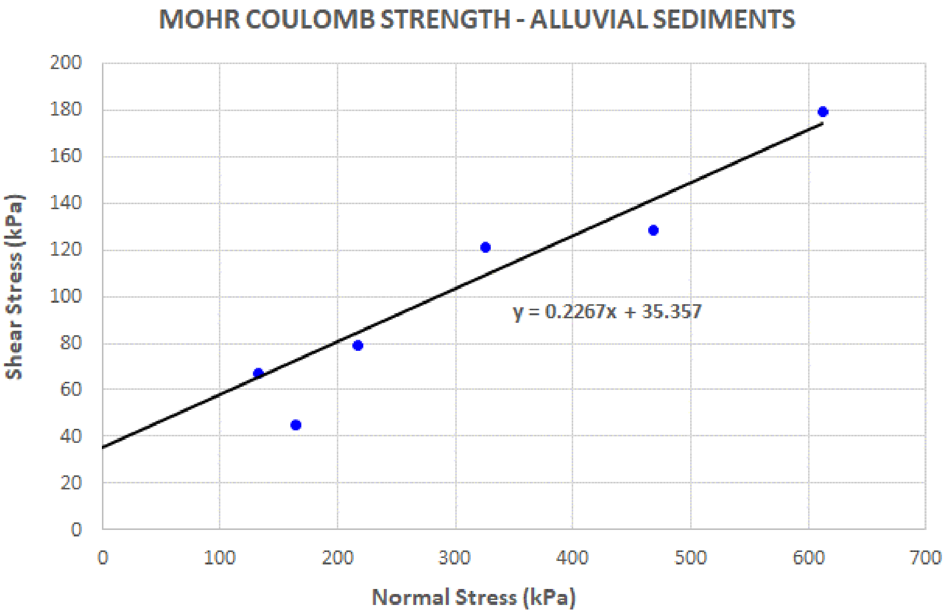



**FIG 4** – Mohr–Coulomb strength envelope from multistage triaxial tests.

**TABLE 1**
Mohr–Coulomb strength summary.

| Data set | Cohesion | Friction angle |
|---|---|---|
| Consolidated, drained multistage triaxials | 35 | 13 |
| Consolidated, drained multistage shear box | 47 | 16 |
| Assumption used in original study (pre-2000) | 150 | 18 |
| Assumption used in current study (2019) | 47 | 16 |

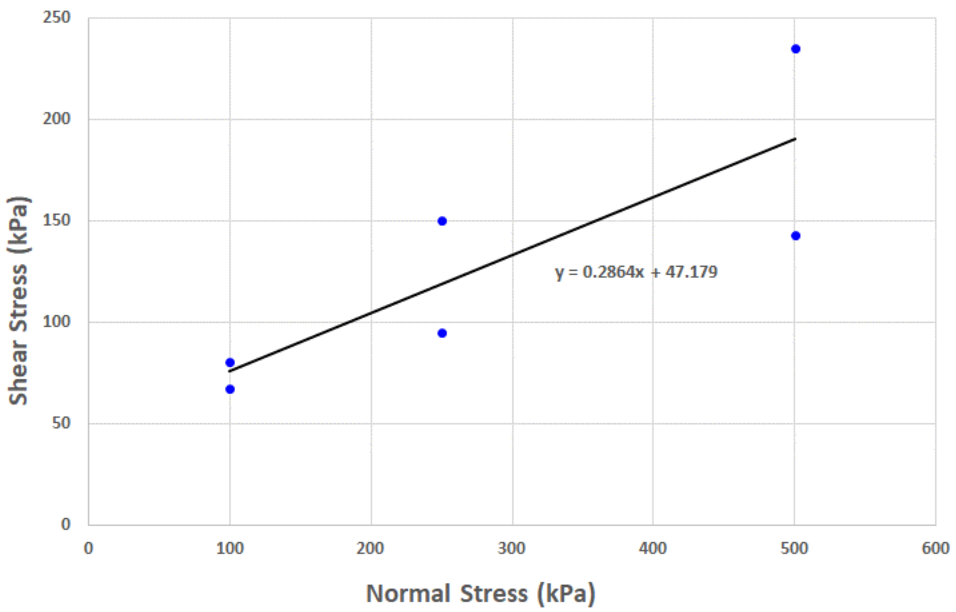



**FIG 5** – Mohr–Coulomb strength envelope from multistage shear box tests.

## Slope stability analysis

Details and results of the original slope stability analysis carried out as part of the pre-2000 study are not available, however it can be assumed that the results gave Factors of Safety (FoS) that complied with acceptance criteria at the time of the study.

The design engineers stated their assumptions for material strength as given in Table 1, with a significantly higher cohesion than the values obtained in their testing program. The rationale for choosing the higher values was that they were based on 'experience within similar materials across Western Australia' and 'closer reflect the expected natural ground values following depressurisation'.

## Pit performance

The pit was commenced in early 2000 and ceased in the latter part of that year due to irrecoverable instabilities. Instability occurred initially on the western wall when the base of the alluvial domain was exposed. A cutback to regain access to the pit floor was executed but further slope failures on the eastern wall precluded further pit development. The extent of the pit void walls affected by slope failure is shown in Figure 6.

**FIG 6** – Panoramic view of the pit circa 2002 showing the extent of failure.

# BACK ANALYSIS

Several 2D and 3D inelastic finite element analyses were carried out by MineGeoTech in order to better understand the geomechanical aspects of design, in preparation for future potential cutbacks to recover the ore left at the base of the pit.

## Case 1

A two-dimensional analysis was carried out using the parameter assumptions from the original study (data set 3 in Table 1) and the expected groundwater conditions based on transient flow drawdown. The parameters selected for the analysis are given in Table 2. The inelastic finite-element code RS2 (Rocscience, 2021a) was used, and the Shear-Strength-Reduction (SSR) methodology applied to obtain a Factor-of-Safety (FoS) for the design. The model geometry is as presented in Figure 7.

**Table 2**

Material parameters used in back analysis Case 1.

| Domain | Density (t/m$^3$) | Modulus (MPa) | Cohesion (kPa) | Friction angle (°) |
|---|---|---|---|---|
| Laterite | 2.1 | 500 | 250 | 35 |
| Alluvial sediments | 1.9 | 100 | 150 | 18 |
| Transitional | 2.3 | 100 | elastic | elastic |
| Fresh | 2.7 | 100 | elastic | elastic |

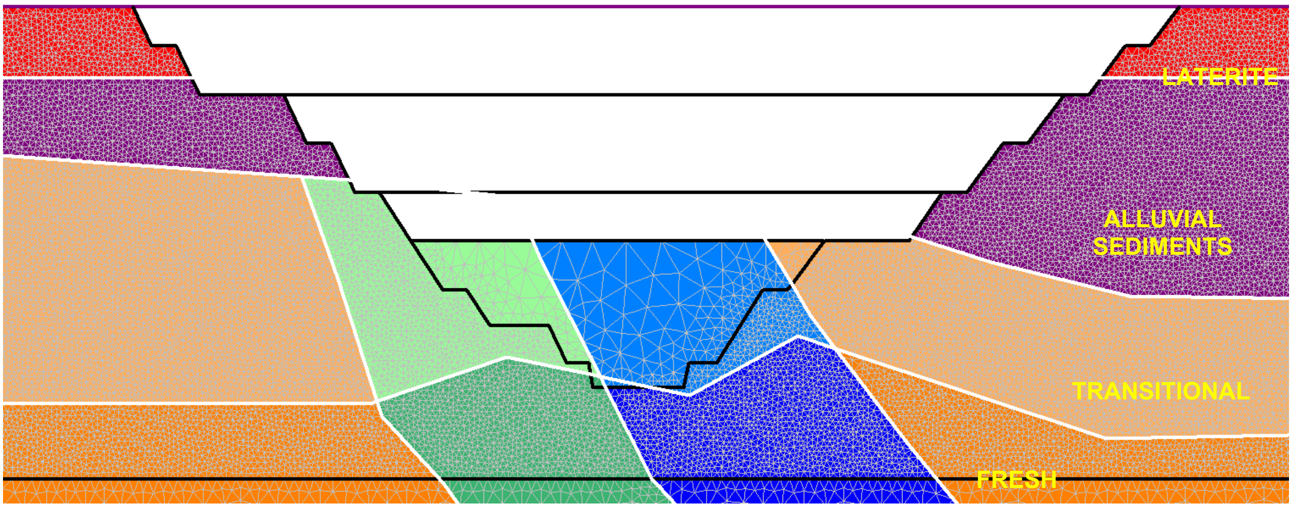



**FIG 7** – 2D finite element model used for back-analysis of pit stability (west–east section).

The results of this initial baseline analysis are summarised in Figure 8, which show plastic strain contours at model equilibrium. The strain contours do not form a continuous slip surface and the FoS against failure is 1.7. This result is consistent with the findings of the original design engineers.

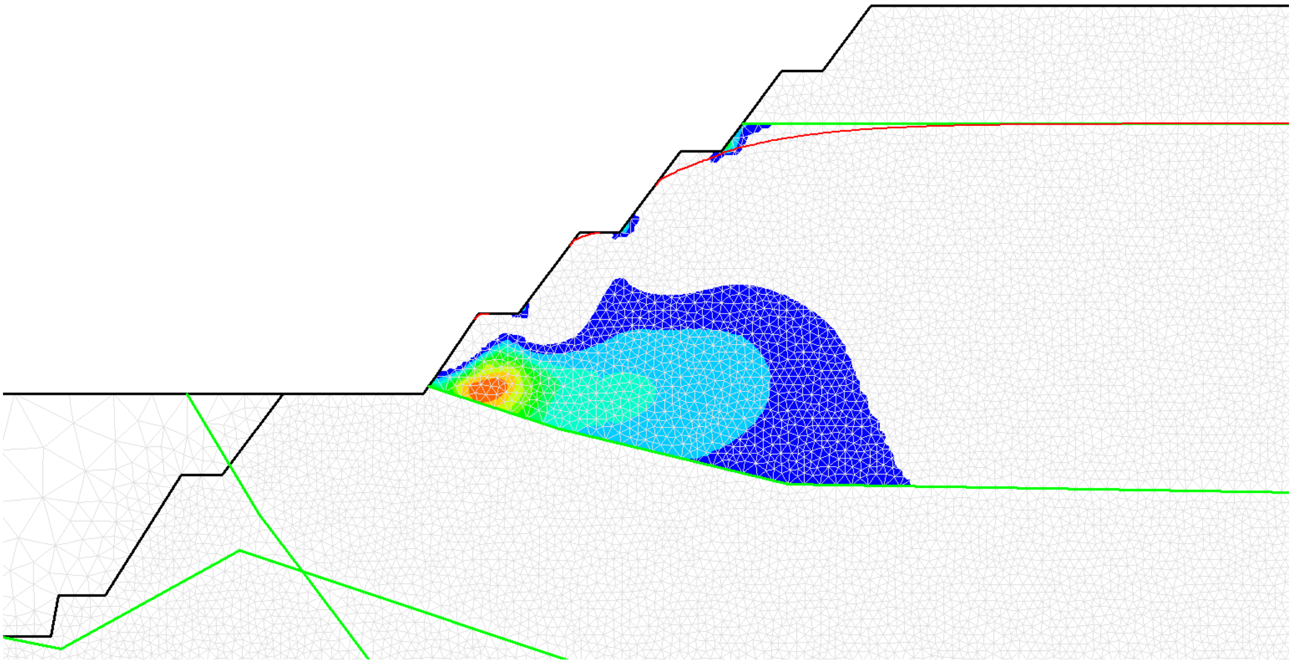


**FIG 8** – Strain contours at equilibrium. Critical FoS = 1.7.

## Case 2

An analysis using the parameters considered more appropriate for design (data set 4 in Table 1) was carried out to determine the actual FoS. Material parameters used in this analysis are documented in Table 3. Transient flow analysis was used to determine the groundwater drawdown concurrent with the mining stage. The results of this analysis are summarised in Figure 9 and indicate a FoS of 0.7, implying instability. The mechanism and extent of failure is indicated by the strain contours in Figure 9.

**TABLE 3**
Material parameters used in back analysis Case 2.

| Domain | Density (t/m$^3$) | Modulus (MPa) | Cohesion (kPa) | Friction angle (°) |
|---|---|---|---|---|
| Laterite | 2.1 | 500 | 250 | 35 |
| Alluvial sediments | 1.9 | 100 | **47** | **16** |
| Transitional | 2.3 | 100 | elastic | elastic |
| Fresh | 2.7 | 100 | elastic | elastic |

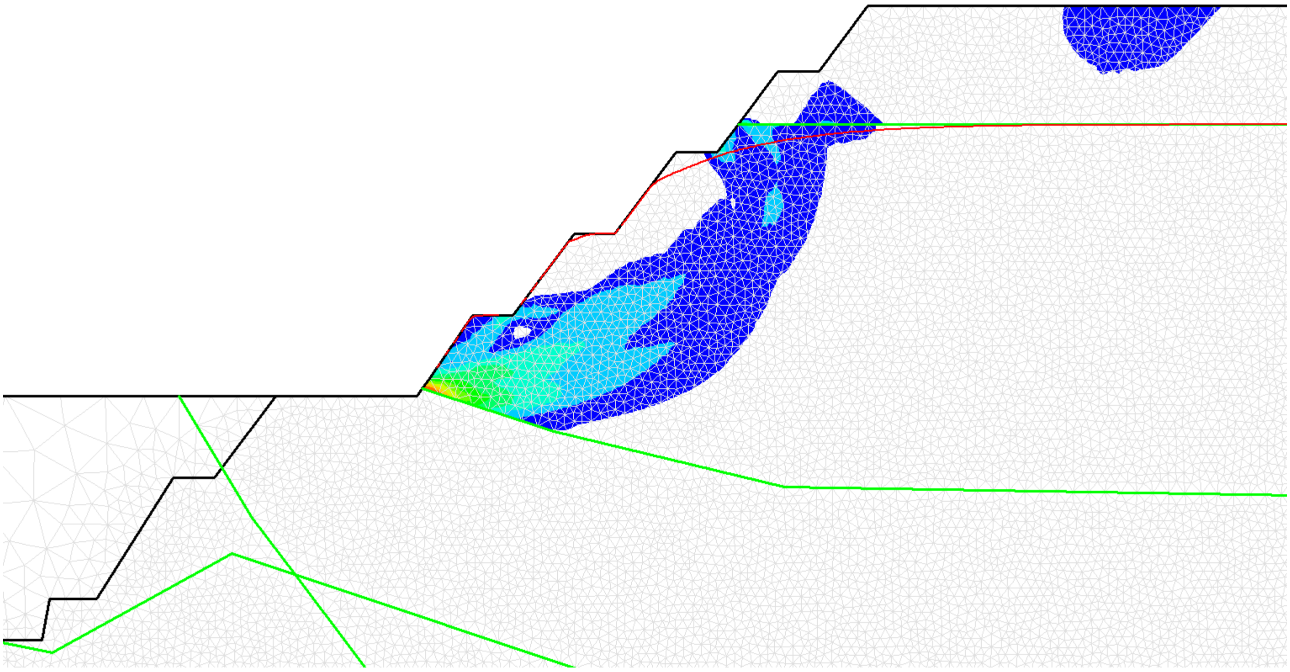


**FIG 9** – Strain contours at FoS of 0.7.

## Case 3

Case 2 was re-run assuming drained conditions and the results shown in Figure 10. The FoS using this assumption is 1.3. The critical requirement to correctly assess hydrogeological conditions of each geotechnical domain is apparent and shows that generic assumptions that pit walls can be drained can have significant economic consequences.

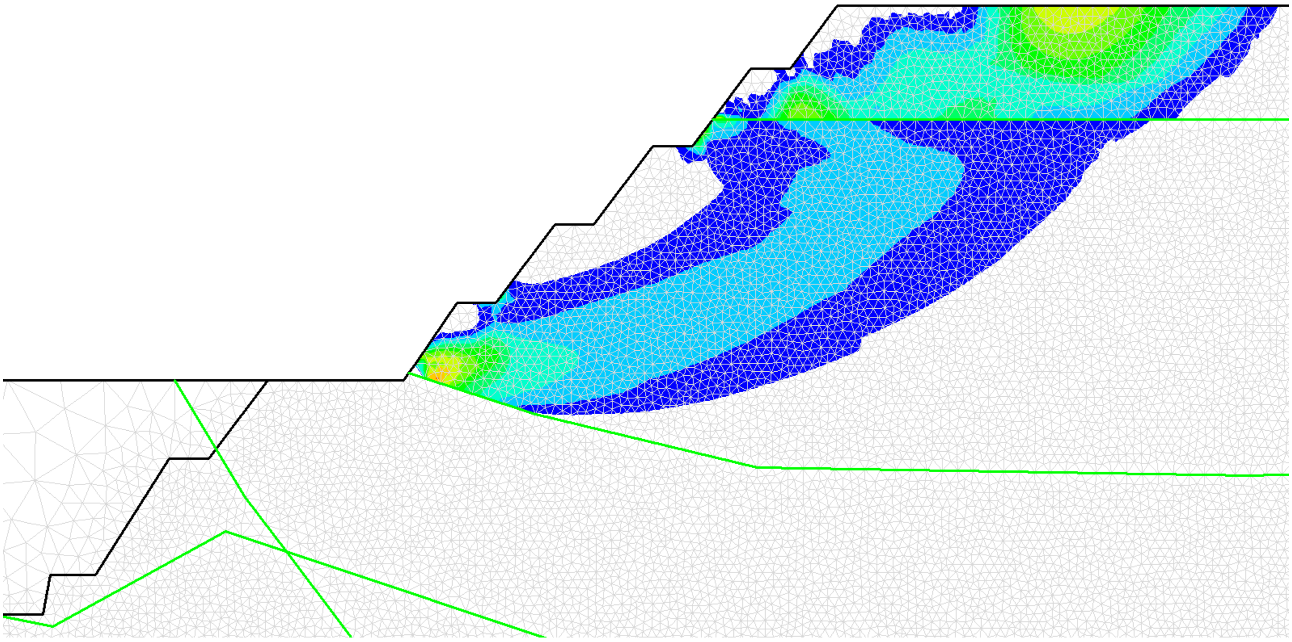
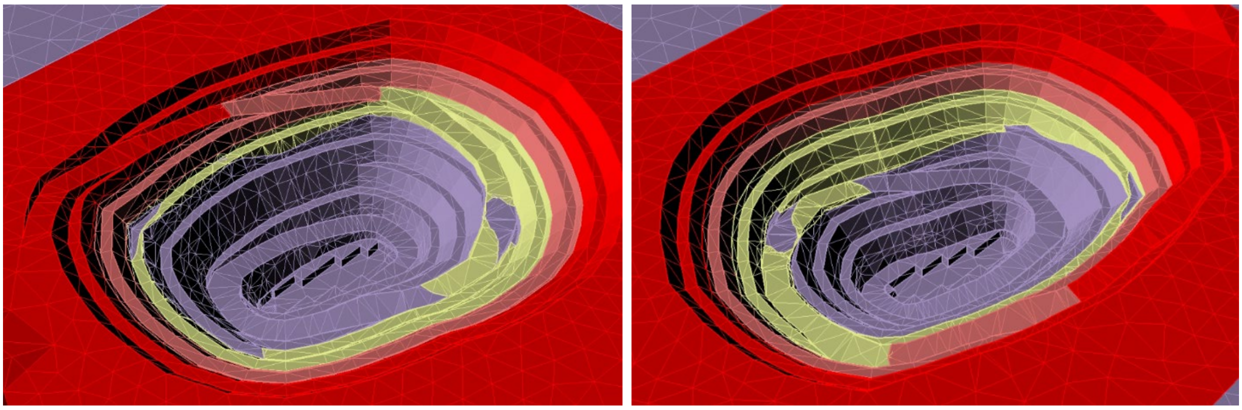


**FIG 10** – Strain contours at FoS of 1.3.

## Case 4

A 3D inelastic finite-element analysis was carried out using the RS3 software (Rocscience, 2021b), to investigate the impact of pit and geotechnical domain geometry on stability. The strength parameters assumed for this analysis are the same as for back analysis Case 2. The RS3 model is shown in Figure 11, with the modelled geotechnical domains indicated.



View from SE View from NW

**FIG 11** – RS3 finite-element model (red = laterite, pink = ferruginised zone, yellow = alluvium, purple = bedrock).

The results are as shown in Figures 12 and 13, as horizontal displacement contours. As can be seen, a good correlation between forecast and observed instability exists, lending credence to the modelling approach and parameters used. The model did not converge, indicating a FoS < 1.0.

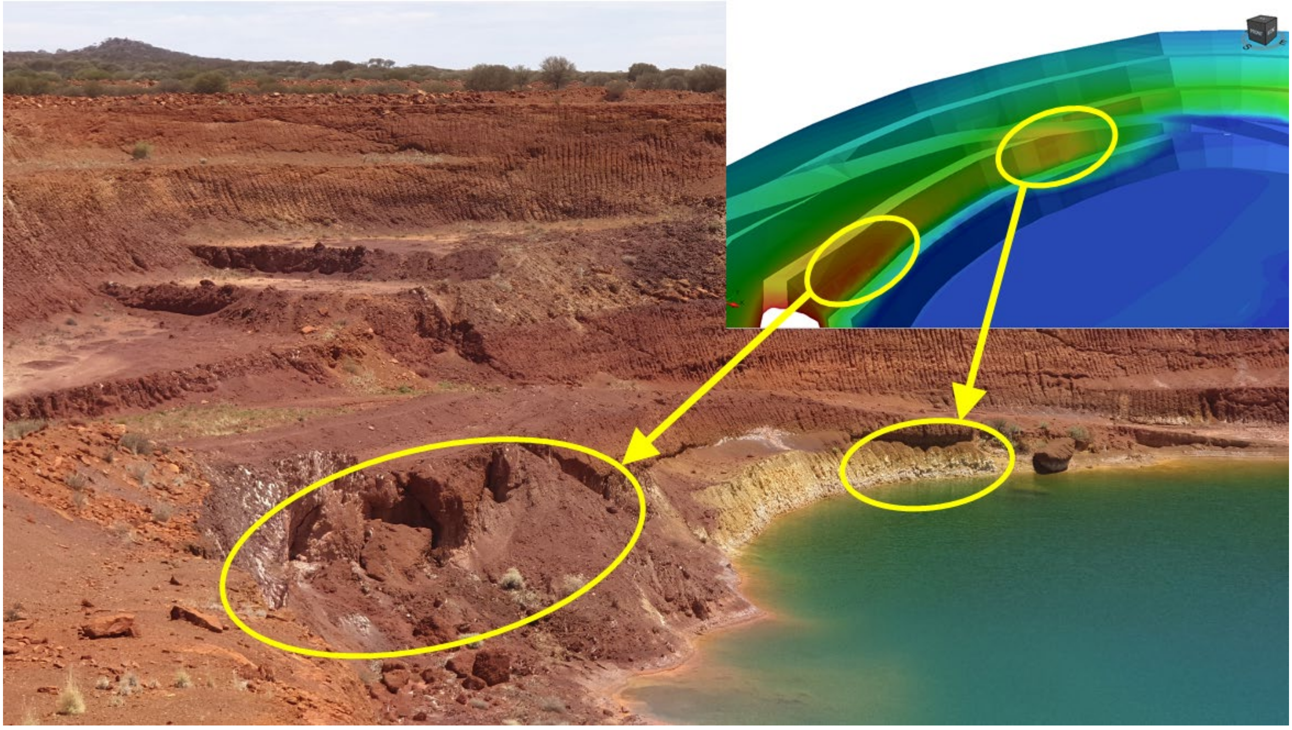


**FIG 12** – Correlation between modelled instability (based on displacement contours) and observed failures – west wall.

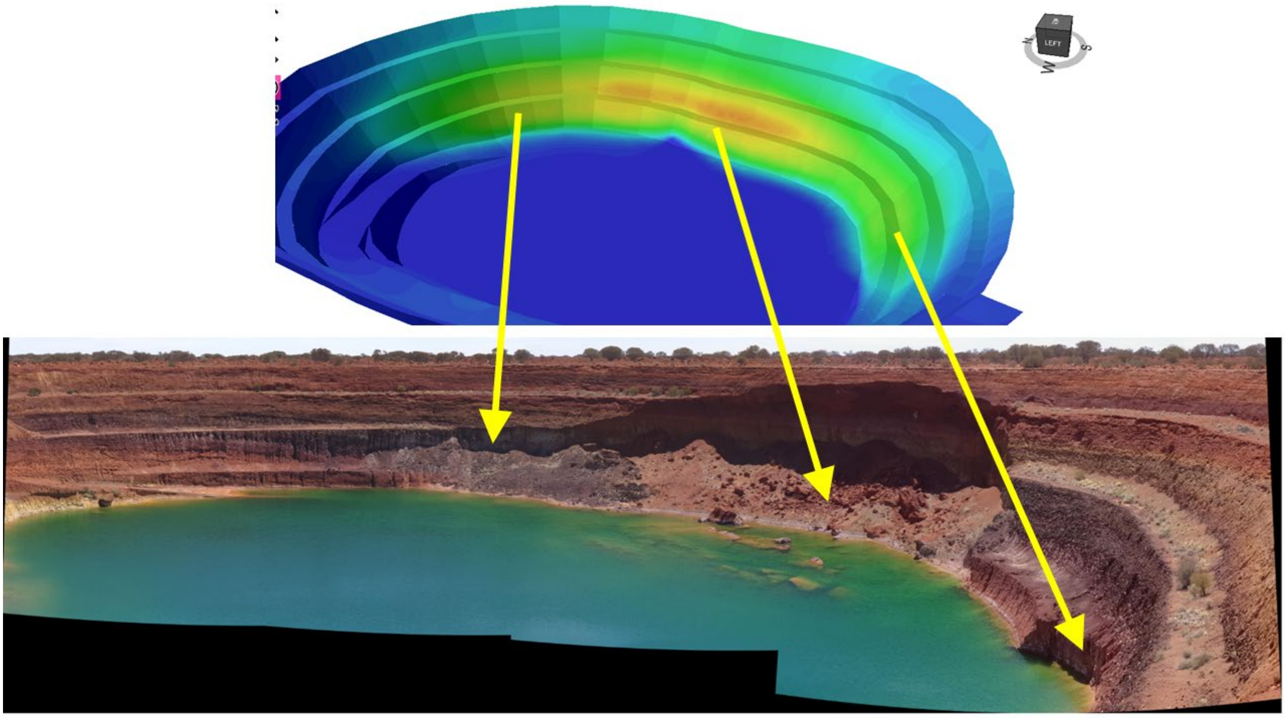


**FIG 13** – Correlation between modelled instability (based on displacement contours) and observed failures – east wall.

## DISCUSSION AND LEARNINGS

The two – and three-dimensional finite-element modelling, when using strength parameters determined from the laboratory test work and groundwater conditions based on transient flow analysis, give realistic results when compared to the actual instabilities recorded (Case 2 and Case 4). The FoS determined in these analyses is <1.0 and the areas of forecast instability match the corresponding failures in the pit.

Comparing the geometry of the modelled instability (Figure 9) to the east wall failure (Figure 13), it can be seen that the geometry of the predicted failure zone matches the observation of a significant overhang caused by the stronger laterite cap.

In the modelled cases, where unrealistically high strength parameters for the alluvials (Case 1) and unachievable groundwater conditions (Case 3) were assumed by the original designers, the analyses indicated stability to acceptable factors-of-safety. This failure in the design process resulted in a significant loss to the pit operators due to the inability to expose and recover the ore hosted in the bedrock.

The original design process did not consider the measured laboratory strength of the alluvials, nor the fact that they would take a significant passage of time to depressurise. Preliminary design work using the laboratory strengths and transient drawdown groundwater conditions, suggests overall slope angles of ~20° for the transported sediments above the weathered bedrock. This is in contrast to the ~40° used in the design.

A total of two multistage triaxial tests and two multistage direct shear tests form the basis for the current study, which is considered inadequate for a pre-feasibility level of study (only four actual samples were tested). The requirement for 'mine-scale' sampling of material strength (stated in Read and Stacey (2009), page 12) has not been met. It is possible that with careful undisturbed sampling of the core at the rig, with moisture preservation, and the use of single-stage direct shear testing, higher material strengths could be achieved. This in turn would indicate steeper wall angles than the 20° determined using the existing data. With more data, the variability in strength can also be used in probabilistic analysis to improve on the design by accepting higher risk in less critical areas of the pitwall.

The hydrogeological study conducted did not provide adequate field test work data for the hydraulic conductivity of the alluvials, focusing rather on the bedrock domains. However, this omission was not critical to the original design engineers because it was assumed that the slopes would be depressurised 20 m into the walls. This unrealistic assumption led to unconservative slope analysis results which justified the steep, unstable wall angles recommended for design.

The impact of minor confinement at the ends of pit by curving of the walls (Case 4) shows the significance of three-dimensional numerical modelling to correctly forecast the pit performance and the benefit of a minor amount of confinement from a curved geometry in very weak material. The same is understood with regards to the performance of bullnoses in stronger material.

## CONCLUSIONS

The case study back-analysed the pit failures to a high level of confidence, by using the laboratory determined shear strength of the alluvial sediments coupled with groundwater drawdown based on transient flow analysis.

The results matched the pit performance in terms of factor-of-safety (<1.0) and in geometry and location of the failures (on the straight walls not the concave walls in the alluvial sediments).

Back analysis using the assumptions made by the original pit designers resulted in factors-of-safety in excess of 1.0, which do not match pit performance.

More representative material strength parameters could have been achieved by improvements to the field data collection and testing procedures as follows:

- More undisturbed drill core samples for test work should have been acquired from a mine-scale drilling program, aimed at obtaining good representation of the material strengths and variability within the alluvials.
- Material samples should have been sealed and enclosed in PVC splits at the drill-rig after extraction from the core barrel splits as soon as practicable after exposure to the atmosphere, in order to preserve natural moisture and minimise disturbance to the internal structure of the material. Alternatively, the very weak material should have been collected with Shelby tubes.
- Laboratory tests should have been on undisturbed, inundated samples, consolidated at the confining pressure for the test.

- The consolidation process should be monitored over time to ensure the sample is not tested prematurely (this could take hours for an impermeable clay).
- Tests should be single-stage only, so that the peak strength for each confinement is obtained in addition to the residual.

Had the above process been followed, the material strengths obtained could potentially have been higher than those recorded in the original study, which in turn may have encouraged the original designers to accept them rather than make an unconservative assumption leading to ultimate failure of the design.

The hydrogeological investigation focused on dewatering aspects, rather than determining hydraulic parameters for groundwater drawdown analysis. No appreciation for the slow draining potential of the alluvial clays was considered in the design process. On the contrary, the unrealistic assumption was made that slopes would be depressurised for 20 m back from the walls, by the use of horizontal drain holes and pumping bores.

Improvements to the understanding of the groundwater pore pressure affecting slope design could have been achieved by:

- Hydrogeological field test work aimed at determining hydraulic conductivities for all the domains which are below the natural water table and exposed on the design pitwalls.
- Transient groundwater flow analysis to determine pore pressure distributions at each stage of mining. Steady-state analysis is not realistic in slow draining materials because the mining time frame is much shorter than the time take to reach steady state.
- No assumption that depressurisation of a saturated, low-permeability clay situated well below the water table, is possible within a practical time frame.

## ACKNOWLEDGEMENTS

MineGeoTech would like to acknowledge our client and current owner of the project for allowing us to present this work so that other projects better understand the impact from deep alluvial clay rich sediments and undertake the appropriate geotechnical and hydrogeological studies.

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# Managing remediation, and risk of failure, to complete extraction of a short life open pit

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## ABSTRACT

Northern Star's Thunderbox Open Pit C-Zone cutback was designed to maximise extraction in a short life excavation; In fresh rock on the western wall this included 20 m to 30 m bench heights with 6.5 m to 10 m berm widths and a 90° pre-split angle.

Rock mass quality in this area is generally fair to good, however, there are sections where the dominant foliation runs parallel with the designed pit wall, and more closely spaced, altered, foliation is present: these areas are difficult to predict and prone to failure.

Stringent controls and monitoring were required in order to safely access below these areas, in particular the use of slope stability monitoring radar, Trigger Action Response Plans (TARPs), re-entry procedures, visual inspections, drone inspections and day-to-day personnel awareness of the hazard.

Robust and comprehensively planned remediation was also required after two significant slope failure events where long reach equipment and rope access techniques were utilised to remediate the areas in question safely, but also in a timely manner, such that expected production targets were impacted as little as reasonably practicable.

This combination of rigorous risk management and well planned remediation resulted in a successful and complete extraction of the C-Zone cutback with no impact to operational safety.

## INTRODUCTION AND BACKGROUND

Thunderbox is situated at the southern end of the Yandal Greenstone Belt in an area where several major intra-greenstone shear zones converge and join with the Perseverance Fault, this shear system appears to be a major geological discontinuity, defining the boundary between two geological domains; The western domain is contiguous with the Wiluna — Mt Keith — Leinster — Mt Clifford sequence and is characterised by deformed and metamorphosed ultramafic and mafic dominated greenstone stratigraphy intruded by granitoid plutons (Figure 1); The eastern domain is dominated by sediments, felsic volcanics and felsic intrusive complexes, in addition to mafics.

At an operational scale the Thunderbox open pit complex is set in a steep-west-dipping sequence of concordant metasediments and metavolcanics. The hanging wall is a sequence of foliated basalt and mafic porphyry, the immediate footwall rock is a thin band of schistose ultramafic, to widths of 10 m, and the footwall rock is a sequence of bedded greywacke and epiclastics.

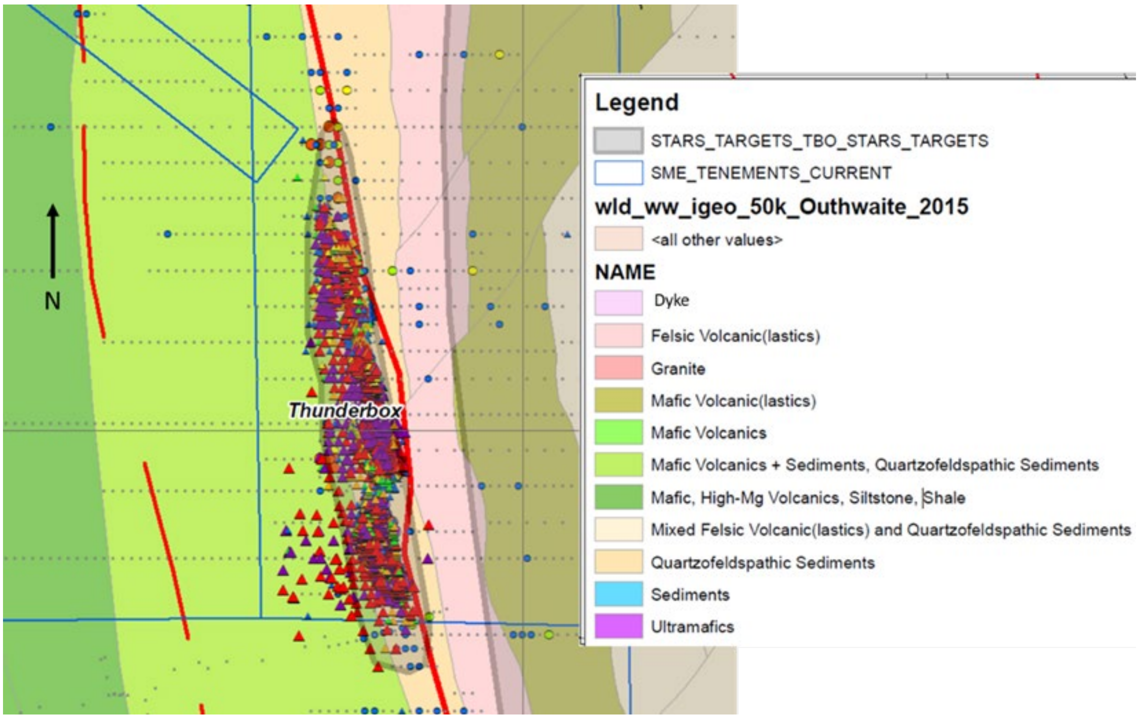


**FIG 1** – Regional geology.

Mineralisation is hosted in the hanging wall of the Thunderbox Shear Zone in an up to 80 m wide intermediate intrusive porphyry with pepperitic apron; The shear zone is over 200 m wide and mineralisation is predominantly confined to the intrusive porphyry. Mineralisation is cut and offset by a major fault known as the 'Gap Fault'.

Gold resources occur in three principal lenses of porphyry:

- A Zone, to the south, which is of narrow width with relatively lower grades.
- C Zone, in the centre, with widths up to 100 m and relatively higher grades.
- D Zone, in the north, which is laterally discontinuous and of narrow width, with relatively lower grades.

A and C Zones had been mined by previous owners prior to the acquisition of the resource by Saracen/Northern Star; cutbacks have been completed in A and C zones with D Zone cutback ongoing (Figure 2).

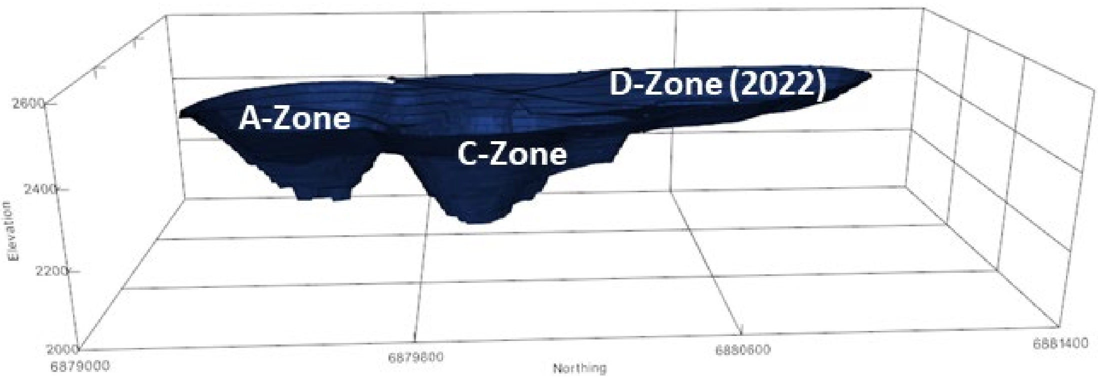


**FIG 2** – Thunderbox long section.

## West Wall – design

The west wall of Thunderbox pit is designed at 85° (90° pre-split) with bench face heights of 20 m to 30 m and berm widths of 6.5 m to 10 m; berm width is loosely based on the modified Richie Criteria (Equation 1).

$$\text{Berm width} = 0.17 \times \text{bench height} + 3 \qquad (1)$$

The inter-ramp angle in fresh rock failure area is 66° and the overall slope angle is 51° (Figure 3).

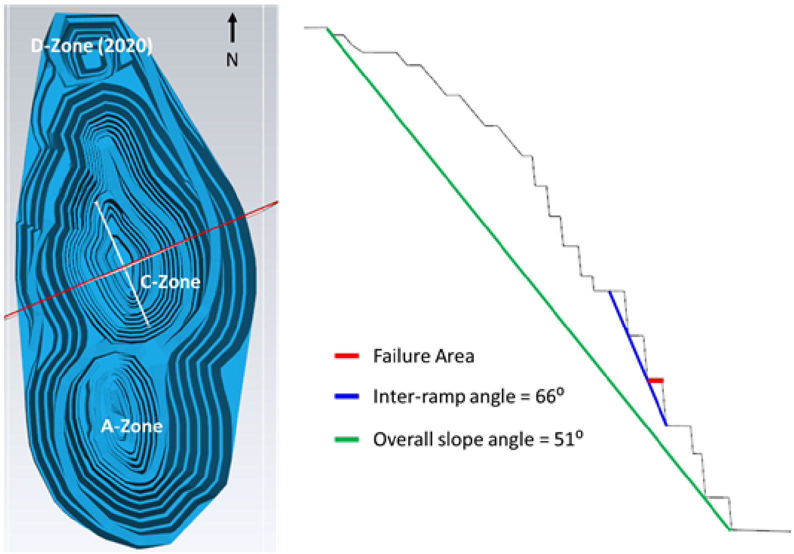


**FIG 3** – Inter-ramp and overall slope angles.

## West Wall – geology and geometry

The western wall (HW) of the Thunderbox deposit is grouped locally into a domain referred to as the HW Mafics; this domain is predominantly made up of basalt, mafic volcaniclastic sandstone tuff and mafic volcaniclastic siltstone. Two dominant joint sets have been identified in the HW Mafic domain, a 74° westerly dipping set and a south-easterly dipping sub horizontal set (Figure 4) with two further minor sets identified dipping steeply to the north-west and east respectively; for the purposes of numerical modelling/rock mass classification three joint sets are typically used unless otherwise indicated by *in situ* observations.

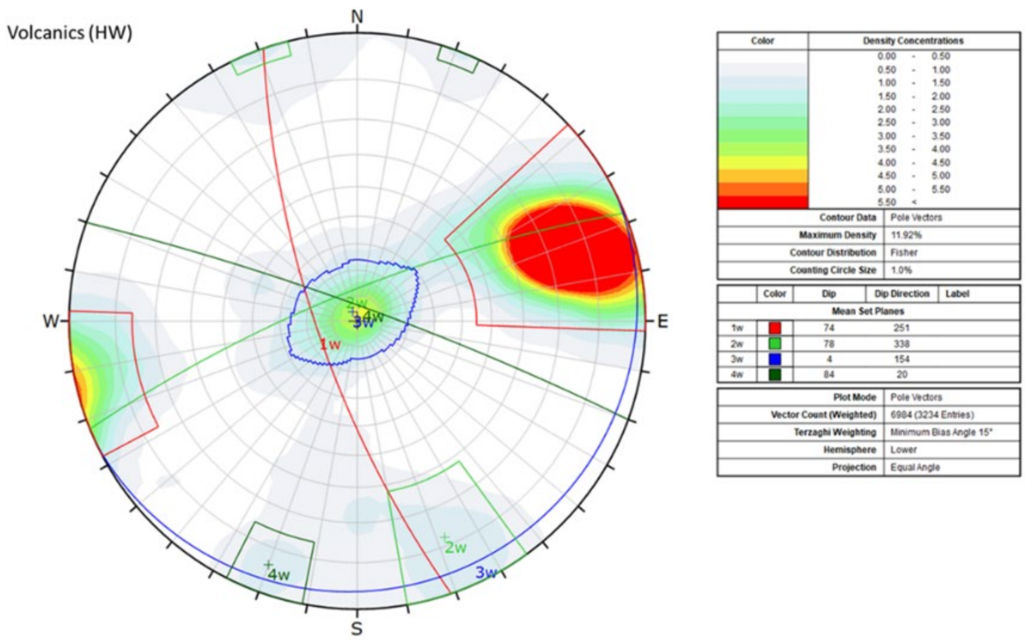


**FIG 4** – Dominant joint sets.

The majority of the rock mass of the western wall is considered to be strong rock of good to fair quality (Table 1); however, areas have been observed with significantly reduced RQD and chloritic alteration on foliation which reduces rock mass quality from fair to good, into the poor range.

**TABLE 1**

Rock mass classification after Barton, Lie and Lunde (1974) and Bar and Barton (2017).

| Area | RQD | Jn | Jr | Ja | O Factor | Jwice | SRF-Slope | Q-Slope | Category |
|---|---|---|---|---|---|---|---|---|---|
| West wall expected/design range | 90 | 9 | 1–2 | 1–4 | 2 | 1.5 | 2.5 | 24–3 | Poor to Good |
| West wall – close foliation | 50 | 9 | 1.5 | 4 | 0.75 | 1.5 | 2.5 | 0.9 | Very Poor |

This does not normally present an unmanageable issue where foliation is favourably orientated with respect to pit walls, however, where foliation is unfavourably orientated this presents a significant risk to the operation. Through mapping and analysis, it was found this relationship could be demonstrated via the Q-Slope method (Bar and Barton, 2017). For slope angles of 85° the Q-Slope expected range is 24 to 3 (Table 1 and Figure 5), which remains stable *in situ*. Stability issues begin to be seen where Q-Slope is less than 3 and become particularly noticeable where areas of closely spaced foliation (RQD = 50) are poorly orientated with respect to the pit wall. In these areas Q-Slope can be as low as 0.9 (Figure 5).

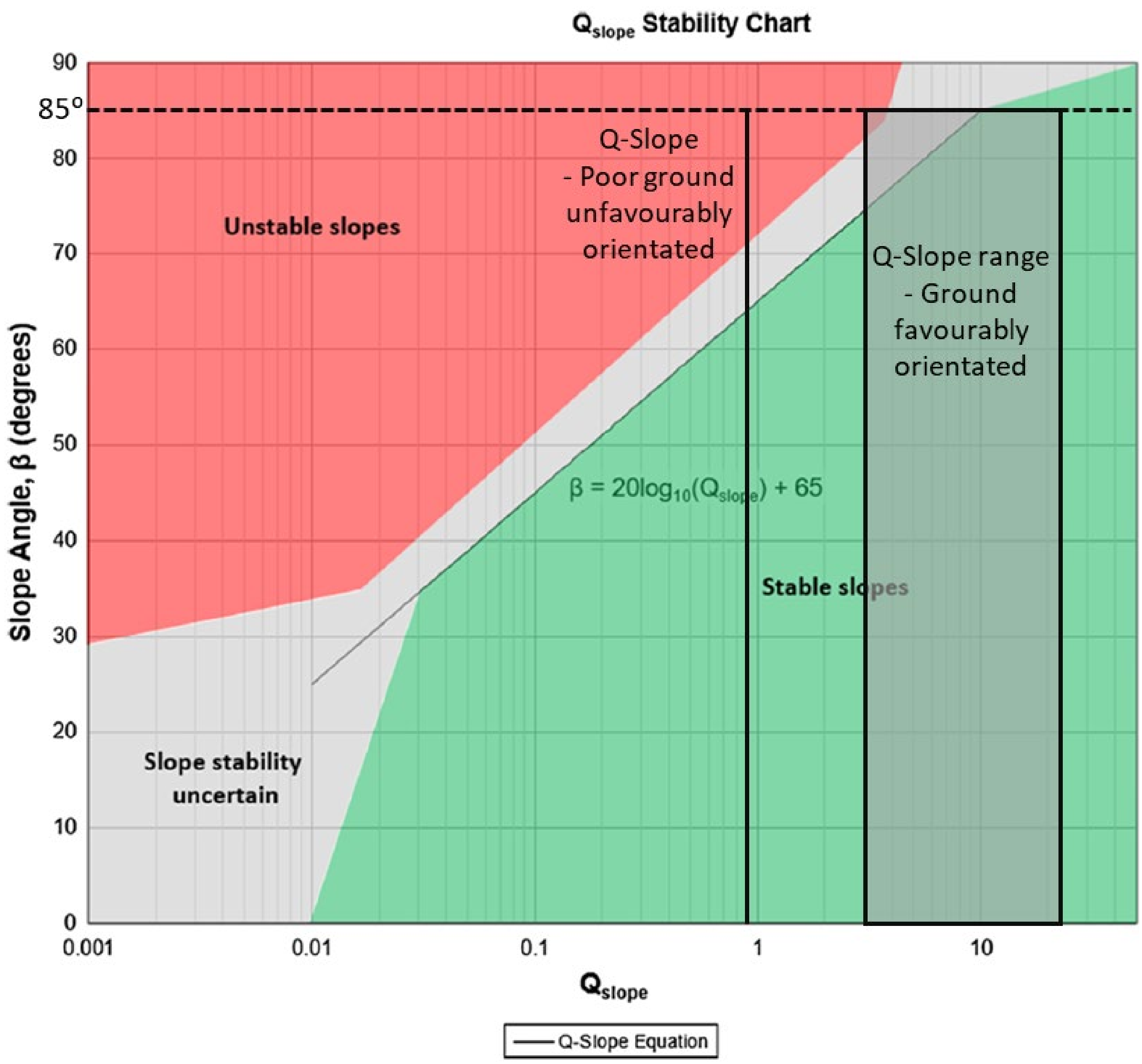


**FIG 5** – Q-Slope after Bar and Barton (2017).

## West Wall – history of failure

Over a period of eight months the west wall at 260RL failed five times (Table 2), with the final failure resulting in two months of remediation work to re-establish access to the pit floor (Figure 6). Most failures occurred in response to firing and were of a relatively small scale, meaning that significant remediation was not required. The final failure occurred in response to a heavy rainfall event and resulted in ramp closure while remediation was undertaken.

**TABLE 2**

History of failure.

| | Failure 1 | Failure 2 | Failure 3 | Failure 4 | Failure 5 |
|---|---|---|---|---|---|
| **Date** | 18/07/2019 | 29/07/2019 | 02/09/2019 | 24/09/2019 | 13/02/2020 |
| **Time of failure** | Within 3 hrs of firing | Within 3 hrs of firing | 9 hrs after firing | 3 hrs after firing | N/A – rainfall |
| **Tonnage of failure** | 70 t | 150 t | 15 t | 350 t | 9000 t |

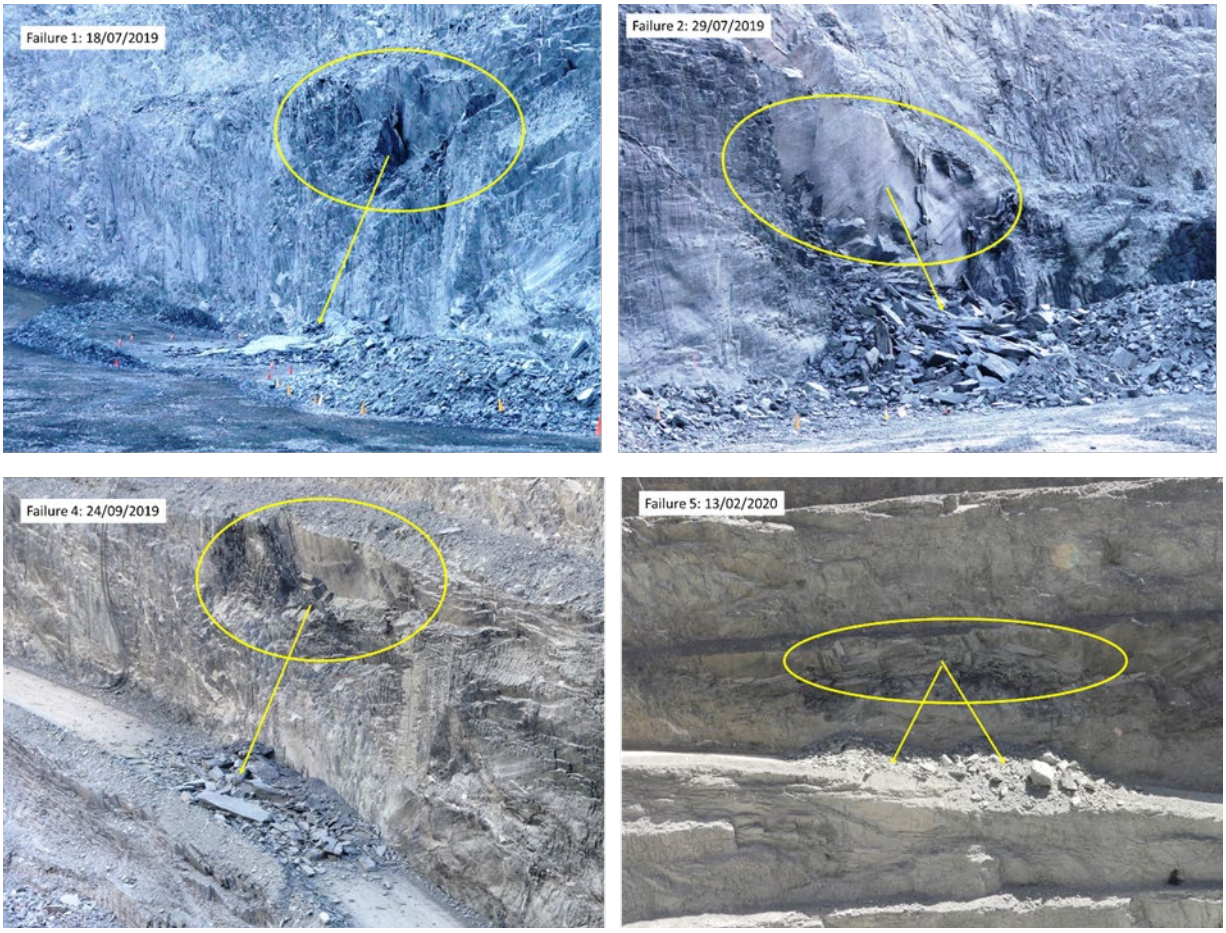


**FIG 6** – Historic failures.

## CONTROLS AND RISK MITIGATION

A number of controls and mitigation methodologies were used to manage the risk to safe production from the west wall area; these included controls to manage:

- Work group awareness and understanding (education).
- Restricted access at key risk times ie after potential trigger events such as blasting, slope stability radar monitoring.

- Trigger Actions Response Plans (TARPs).
- Upskilling of key personnel such as Shift Supervisors in the use and interpretation of monitoring tools, and remediation after failure.

## Slope monitoring and inspections

Slope stability monitoring was undertaken using a Reutech MSR300, and later MSR Modular, radar system; this system uses real aperture radar technology to individually resolve each modelled point in azimuth, range and elevation, allowing movement/failures to be tracked without relying on a Digital Terrain Model (DTM). Three radars in total were used to monitor Thunderbox C-Zone with two dedicated specifically to the western wall (Figure 7). Data transfer from radar units to office server was completed using a series of point to point wifi links.

Prior to the fifth (and final) failure 'Geotechnical' and 'Critical' radar alarms were set at 1.5 mm/hr and 3 mm/hr velocity respectively; after the final failure, and for the remainder of mining in C-Zone, alarm thresholds were reduced to 0.3 mm/hr and 1.5 mm/hr as shown in Table 3. The lower alarm thresholds did result in periods of pit closure with no associated instability; however, this was considered as lying within the 'as low as reasonably practicable' risk mitigation category. The decision was based on the identification of low background velocities (25 mmm to 50 mm in four months – Figure 8), the failure mechanism, and the potentially small window between alarm trigger and failure initiation at higher thresholds.

A geotechnical visual inspection was also required after known trigger events, such as rainfall or blasting, and after any closure due to radar alarms before re-opening to pit to general workgroups.

**TABLE 3**

Radar alarm thresholds.

| Alarm | Velocity towards | Time frame |
|---|---|---|
| Geotechnical (original) | -1.5 mm/hr | 2 hours |
| Geotechnical (after Failure 5) | -0.3 mm/hr | 2 hours |
| Critical (original) | -3 mm/hr | 2 hours |
| Critical (after Failure 5) | -1.5 mm/hr | 2 hours |

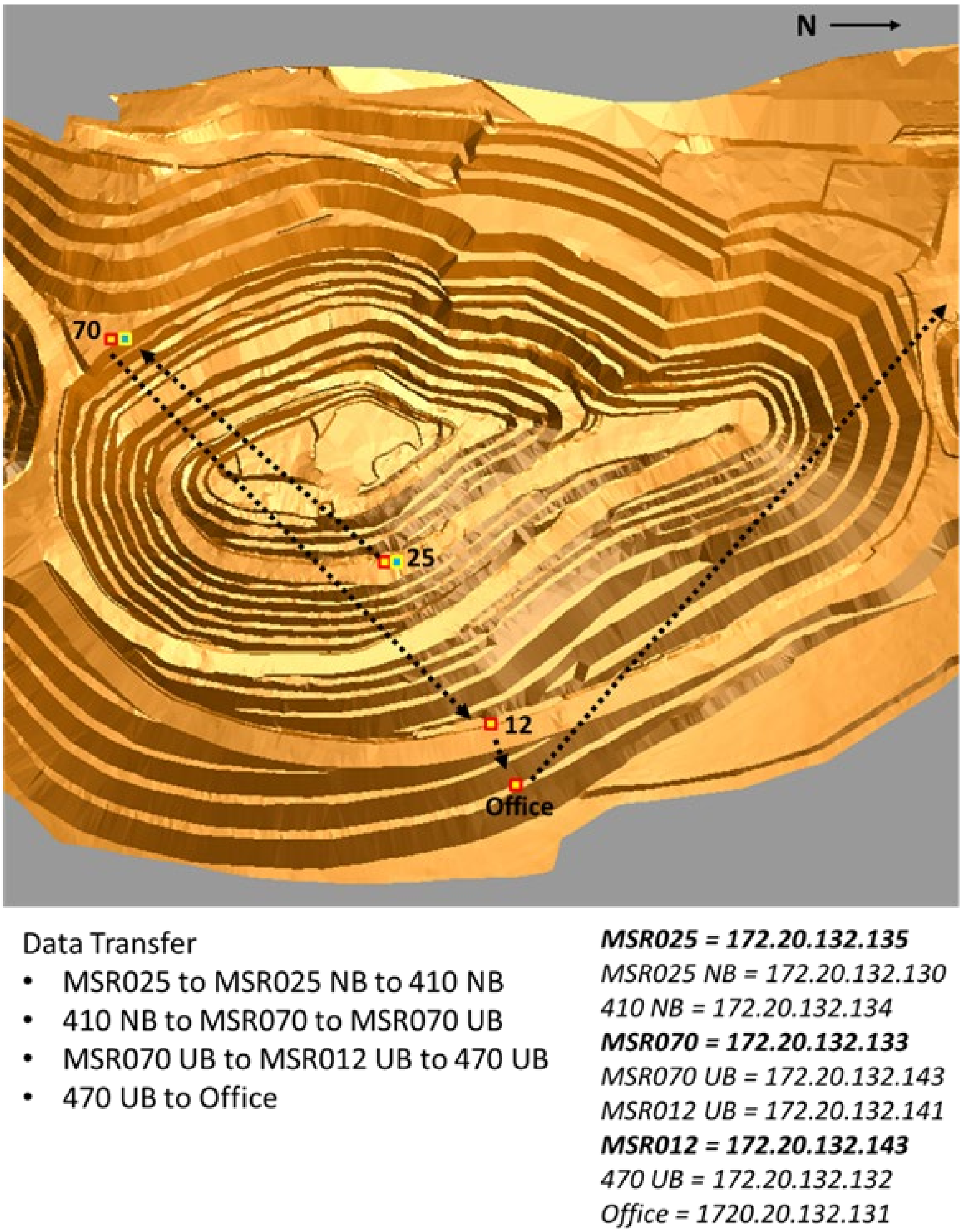


**FIG 7** – Radar locations and data transfer.

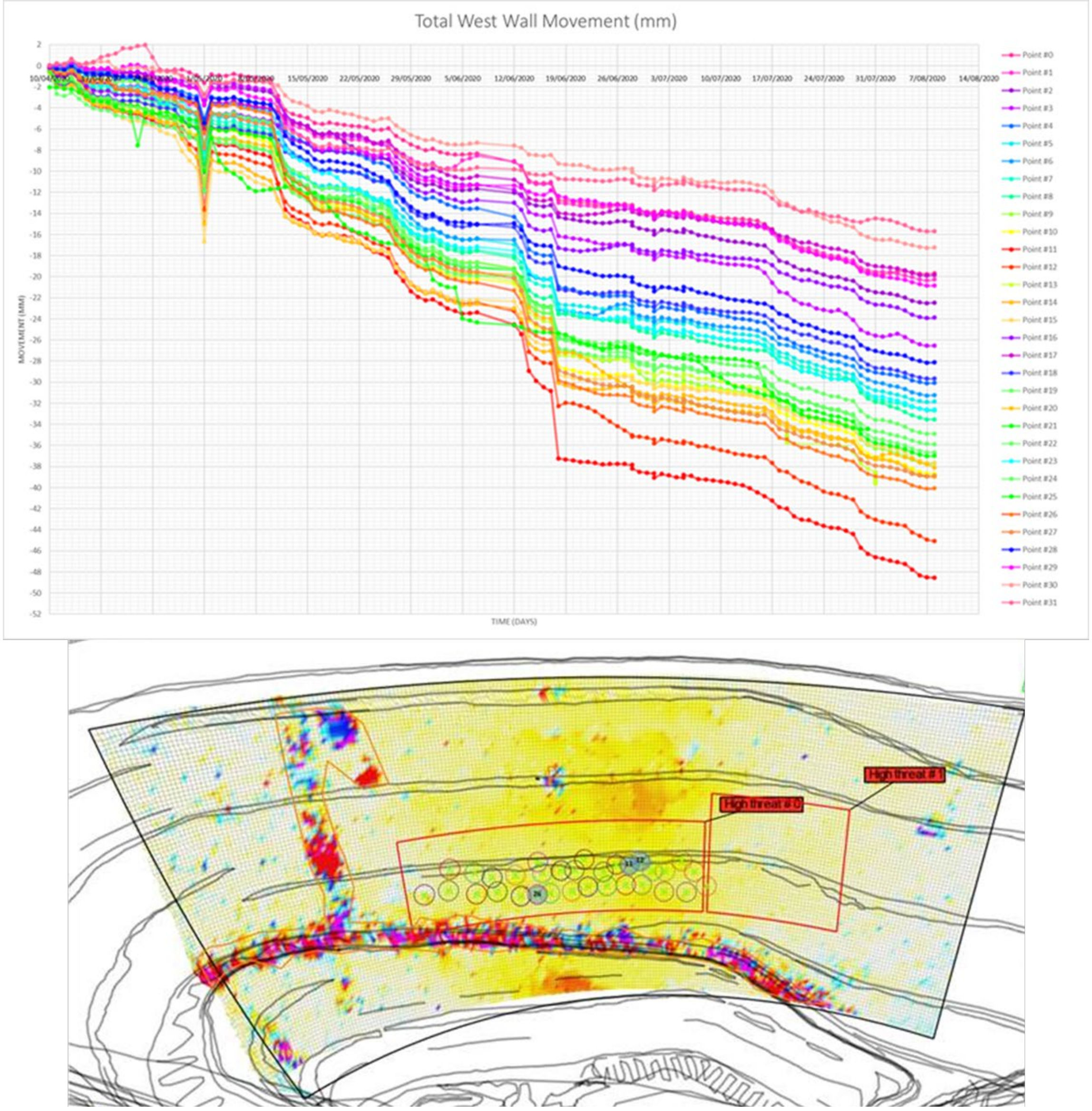


**FIG 8** – Total movement and constant monitoring point locations.

## Trigger Action Response Plans (TARP), communication and training

Communication and training of crews and key stakeholders are integral to ensuring safe production where a significant potential hazard has been identified. All crews were trained in radar monitoring systems and the monitoring TARP via refresher presentations and as part of site specific inductions. Crews were then updated on changes via Return to Work (RTW) and Pre-Shift (PSI) meetings. Additional training was given to Shift Supervisors and Leading Hands, empowering them to undertake an initial threat assessment. Whilst the response to the slope movement alarms remained relatively prescriptive, and largely based on an engineering assessment of threat level, the additional training for field leaders minimised operational downtime due to minor system faults/resets. The specific training also allowed rapid escalation in the event of a 'Geotechnical' or 'Critical' alarm being triggered without waiting for engineering assessment.

The radar system was, and continues to be, monitored 24/7 by Mill Control operators who received specific training on how to respond in the event of an alarm trigger in conjunction with the TARP shown in Figure 9. The TARP was developed to specifically correspond to the colours displayed on screen when different radar alarms were triggered, providing an immediate visual cue for which response line to follow. The TARP provides response guidance ranging from temporary communication outage, through minor system faults, and up to significant movement detected. Specific advice was provided for alarms in the vicinity of the pit ramp, which was for personnel to remain *in situ* in a designated safe area of the pit (rather than attempt to evacuate). This decision was based on the potentially rapid nature of failure in fresh rock, and there being only a single means egress from the open pit.

**THUNDERBOX OPERATIONS**

**Trigger Action Response Plan: RADAR ALARMS.**

Username = XXX Password = XXX Shift Supervisor: XXXX XXX XXX Mining Superintendent: XXXX XXX XXX

| | BLUE; LEVEL 1 | YELLOW; LEVEL 2 | ORANGE; LEVEL 3 | RED; LEVEL 4 - *Cease Work* |
|---|---|---|---|---|
| Conditions | Calibration Scan (system reset)<br>Comms link down<br>No movement data available until reset complete | Minor System Fault<br>Low fuel, doors open, low battery, scan time exceeded. | Geotechnical Stability Alarm<br>Movement detected | Critical Stability Alarm<br>Critical System Fault<br>Significant movement / serious system fault |
| MILL CONTROL | Notify Shift Supervisor (Radio Channel 1)<br>Specify which radar is alarming<br>(MSR012 OR MSR070 OR MSR025) | Notify Shift Supervisor (Radio Channel 1)<br>Specify which radar is alarming<br>(MSR012 OR MSR070 OR MSR025) | Notify Shift Supervisor (Radio Channel 1)<br>Contact Mining Superintendent if Shift Supervisor cannot be reached.<br>Specify which radar is alarming<br>(MSR012 OR MSR070 OR MSR025) | Notify Shift Supervisor (Radio Channel 1)<br>Contact Mining Superintendent if Shift Supervisor cannot be reached.<br>Specify which radar is alarming<br>(MSR012 OR MSR070 OR MSR025) |
| SHIFT SUPERVISOR | Day Shift: Contact Geotechnical Engineer, or delegate, and check status of movement prior to system reset.<br>Night Shift: Remove personnel and machinary from within 30m of highwall beneath relevent wall until data is restored. Machinary in pit should remain in situ (away from highwall) if alarm region is above / below ramp.<br>Contact Geotechnical Engineer, or delegate, if data is not restored within 2 hours. | Check radar equipment, rectify fault if possible, notify Geotechnical Engineer, or delegate, when appropriate i.e. in the morning if minor fault occurs during night shift. | Day Shift: Contact Geotechnical Engineer, or delegate, for advice.<br>Night Shift: Close ramp and remove personnel and machinery from within 30m of highwall beneath relevant wall until alarm clears. Machinery in pit should remain in situ (away from highwall) if alarm region is above / below ramp.<br>Contact Geotechnical Engineer, or delegate, if alarm state does not clear within two hours. | CLOSE PIT TO ALL TRAFFIC<br>Remove all personnel and machinery from within 30m of highwall beneath relevant wall. Machinery in pit should remain in situ (away from highwall) if alarm region is above/ below ramp.<br>Contact Geotechnical Engineer & Quarry Manager. |
| MINING SUPERINTENDENT | Normal Operations | Normal Operations | Day Shift: Liaise with Geotechnical Engineer / Shift Supervisor as required.<br>Night Shift: Normal Operations | Liaise with Geotechnical Engineer, or delegate, and Shift Supervisor / Quarry Manager as required. |
| QUARRY MANAGER | Normal Operations | Normal Operations | Normal Operations | Liaise with Geotechnical Engineer re: area closures, likely failure time / volume and potential re-entry times.<br>Inform relevant statutory and corporate stakeholders in the event of failure. |
| GEOTECHNICAL ENGINEER or Delegate | Check data for failure trends and advise Shift Supervisor of required actions. | Assist Shift Supervisor to rectify fault as required. | Day Shift: Check radar data and advise relevant stakeholders of required actions.<br>Night Shift: Check radar data and advise Shift Supervisor if contacted.<br>Contact Superintendent if area below alarm region is to remain closed. | Check radar data and advise Quarry Manager of required actions. |

**FIG 9** – Radar Trigger Action Response Plan (TARP).

## Slope remediation

Remediation was required after Failures 4 and 5 (Table 2).

Failure 4 was of a relatively small volume (≈350 t), however, a significant amount of closely foliated, potentially loose/unstable, material was observed to be left on the wall after the failure which needed to be removed to re-establish safe access to the pit floor (Figure 10). Remediation was completed by constructing a pad and using a long reach excavator to pull down loose material. The excavator model used was a CAT 340 Long Reach with 18 m max reach and 141 kN bucket digging force (Figure 11), the unit came with an additional 2 t counterweight installed to ensure stability – from building pad to removal, this remediation took approximately 10 days.

Use of the long reach excavator was found to be a safe and efficient mechanism for removing closely foliated areas of already loose material. However, where larger blocks and/or more locked in blocks were encountered the low digging force on the bucket meant that it was difficult to extract them; in one instance this resulted in damage to the bucket which required welding to repair before continuing work.


**FIG 10** – Long reach excavator remediation.

WORKING RANGES & FORCES

| | |
|---|---|
| Maximum Digging Depth | 13050 mm |
| Boom | 10.6 m (34'9") |
| Stick | LRE Stick 7.1 m (23'4") |
| Maximum Reach at Ground Level | 18080 mm |
| Maximum Loading Height | 12770 mm |
| Minimum Loading Height | 3210 mm |
| Maximum Depth Cut for 2440 mm (8 ft) Level Bottom | 12960 mm |
| Bucket Digging Force - ISO | 141 kN |
| Stick Digging Force - ISO | 92 kN |
| Bucket | GD 0.93 m3 (1.22 yd3) |
| Maximum Cutting Height | 15620 mm |

**FIG 11** – CAT 340 Long Reach working ranges and forces.

The fifth failure (Table 2), at 9000 t, was the largest failure to originate from the western wall, and a number of remediation options were considered for this area (Table 4). The selected option (highlighted green) was the use of a crane and man basket to remove loose material and large blocks from the failed area using the on-site rope access crew. This was assessed as presenting a risk as low as reasonably practicable (ALARP) to personnel undertaking remediation ie those who would be accessing the pit to complete extraction.

This involved prior removal of failed material using an underground remote loader (Figure 12) and the building of incremental pads for the crane, but nonetheless offered several advantages over other options, including:

- Smaller, incremental, pads required compared to other options.
- Manual (rope access) scaling of wall above failure to establish an area where permanent man anchors could be installed was not required.
- Rope access crew were on-site and could be promptly utilised to complete work.
- Monitoring had already shown over previous failures that large scale movement could be reliably picked up – by remediating incrementally over the failed area the crane was always located in a safe area, the man basket enabled scaling crew to be moved away from the wall rapidly should further movement be picked up.
- The man basket provided overhead protection which significantly reduced the risk to personnel from individual rockfall/rock dislodgement which may not be picked up by the radar.
- Smaller pads and reduced manual work decreased the estimated time to remediate from 3+ months to ≈2 months.

Remediation was completed on schedule, finishing on the 09/04/2020, a little less than two months after failure on 13/02/2020.

**TABLE 4**

Remediation options for Failure 5.

| Option | Risk classification | Main risk category | Likelihood | Hierarchy of control | Conclusion |
|---|---|---|---|---|---|
| Clear material from ramp – subsequently reduce/limit personnel exposure. | Moderate | Health and safety | Possible | Admin | Too high a risk of personnel injury |
| Remediation using rope access techniques | Catastrophic<br>Major | Operational<br>Health and safety | Almost certain<br>Rare | Elimination | Timing impractical due to remaining life of C-Zone<br>Anchor points for rope access personnel meant lack of manoeuvrability in case of further rockfall |
| Remediation using long reach excavator | Catastrophic | Operational | Almost certain | Elimination | Pad requirements excessive given remaining life of C-Zone and pit floor level with respect to ramp<br>Observed size of blocks left *in situ* too big for bucket digging force of machine |
| Remediation using rope access crew and crane with man basket attachment | Major<br>Minor | Operational<br>Health and safety | Almost certain<br>Rare | Elimination | An operational impact is expected, however, it is likely to be less than when using other options<br>Use of crane basket provided good manoeuvrability in case of further rockfall and overhead protection |
| Cutback | Catastrophic | Cost to business | Almost Certain | Elimination | Impractical given remaining life of C-Zone Pit and ROM access location |

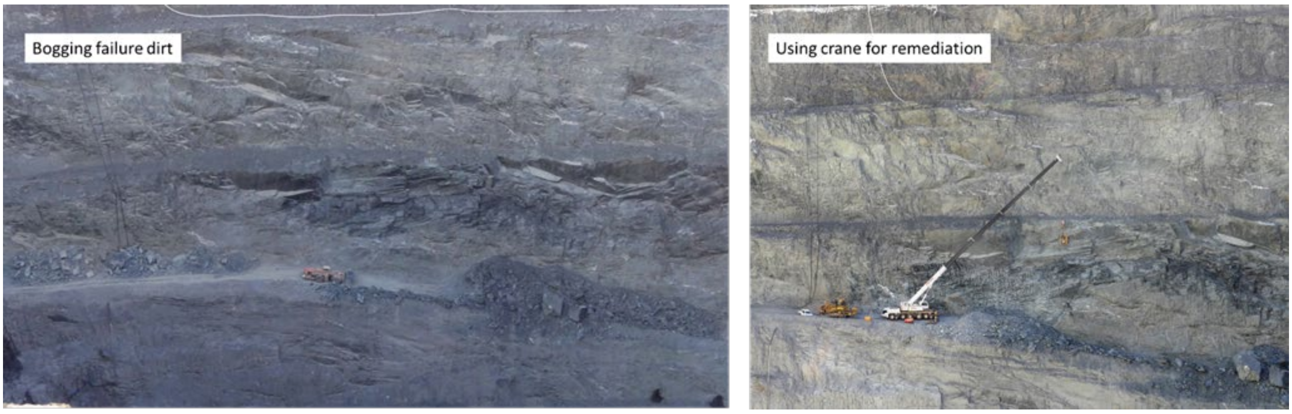


**FIG 12** – Underground remote loader removing failed material and remediation using crane with man basket attachment.

## CONCLUSIONS

### Monitoring, communication and training

The approach taken at Thunderbox towards a significant hazard was a three part strategy:

1. Good monitoring data, with regularly calibrated alarm thresholds, is available and used to inform decision-making.
2. Operating personnel are adequately and regularly trained on required responses for the changing environment. Responses are clear and are easily understood, with TARPs used as the primary communication aid.
3. Supervisory personnel are trained to a higher level of understanding and empowered to make decisions.

The very low alarm thresholds used did result in-pit closure, and associated production loss, as a result of 'false' alarms, however, given that the failure area was above the main egress from the pit and the likely failure mechanism could result in rapidly developing instability, minor periods of production loss while waiting for engineering assessment were preferred over potential personnel exposure to the hazard.

Additional training for Shift Supervisors/Leading Hands improved the understanding of monitoring tools. This reduced downtime due to minor system faults and allowed for rapid escalation in the event of significant movement being identified, rather than the delay caused in waiting for Geotechnical/Engineering assessment of threat level.

### Slope remediation

Remediation of areas outside the reach of mining equipment available on-site was completed using both long reach equipment and pseudo (from a crane) rope access techniques.

Use of long reach equipment was successful where height was within the reach without excessively large pad requirements and where material blocks to be removed were small and already loose. Where material blocks were large and/or locked in the low bucket digging force made extraction difficult.

Use of a crane and man basket to complete remediation was successful in all areas and provided a number of benefits in terms of both personnel safety and operational efficiency, including:

- Pads could be constructed incrementally – minimising required material.
- Crane was always situated in safe ground.
- Personnel in man basket could be rapidly removed from the wall should further movement be detected.

- The man basket provided protection from small rocks falling which would not be picked up by the radar.
- Remediation was sped up by reducing/eliminating the need for manual scaling/installation of man anchors above area.


## ACKNOWLEDGEMENTS

The authors would like to acknowledge Northern Star Resources Ltd, in particular Mr Simon Thomas, for providing technical review of this paper and permission to publish.

They would also like to acknowledge the hard work and dedication of site based personnel at all levels who enabled the Thunderbox C-Zone cutback to be extracted successfully, and the help and advice of numerous other Northern Star Resources Ltd/Saracen employees – in particular Mr Peter Ganza, Mr Simon Jessop and Mr Raleigh Finlayson.

The technical advice of Mr Ian de Bruyn and Ms Diane Walker of SRK Consulting was also invaluable at several stages of the C-Zone cutback, both before and after failure.

Finally, the experience and help provided by Geovert Ground Engineering Pty Ltd, both in the planning and execution of Failure 5 remediation was vital to its ultimate success.


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# Mining method, and controls, to cut back a failed slope at KCGMs Fimiston Pit (KCGM Superpit)

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## ABSTRACT

The East wall of the Kalgoorlie Consolidated Gold Mines Fimiston Open Pit (KCGM Superpit) has a history of wall instabilities impacting mining operations; the most significant of which occurred in May 2018, when more than 1 Mt of rock dislodged over six, 30 m high bench stacks and impacted the pit floor. Since 2011 approximately 1.250 Mt of material has fallen from the East wall at Fimiston Pit across three failure events (2011, 2018 and 2020). Each failure has been associated with major structures intersecting the pit walls: the Fiji Fault, the Australia East Secondary Splay Fault and the EWD3 Fault. In each instance the trigger for failure has been transient pore pressure in the pit walls.

Northern Star will mine a cut back behind the failure area over a vertical distance of 180 m. Cut back mining will include the extraction of a remaining *in situ* wedge, which is assessed as kinematically able to fail, and a subvertical failure backscarp in Stage 1. A substantial amount of rill material from the 2018 and 2020 failures will be extracted in Stage 2.

This paper presents the methodology in use to mine Stage 1 of the cut back, safely, through this potentially high-risk area, detailing the specific mining methodology employed, the controls put in place and the monitoring required.

Day-to-day monitoring will be undertaken using the already installed slope stability radar and will feed into a daily assessment of activities which may be undertaken in the cut back area. Initial activities will focus on eliminating the potentially unstable wedge and vertical backscarp with use of innovative drill and blast techniques/products and the extensive use of remote capable equipment to limit personnel exposure.

## INTRODUCTION

The KCGM gold mining and processing operations are located on the outskirts of the City of Kalgoorlie-Boulder in Western Australia, approximately 600 km east-north-east of Perth (Figure 1). The operations comprise the Fimiston open cut mine, the Mt Charlotte underground mine, a gold ore processing plant at the Fimiston site and an ultra-fine grinding processing plant at the site of the former Gidji Roaster smelter. This paper focuses on controls implemented in order to safely cut back around the 2018 east wall failure, location shown in Figure 2, at the Fimiston Open Pit (KCGM Superpit).


**FIG 1** – KCGM location map.

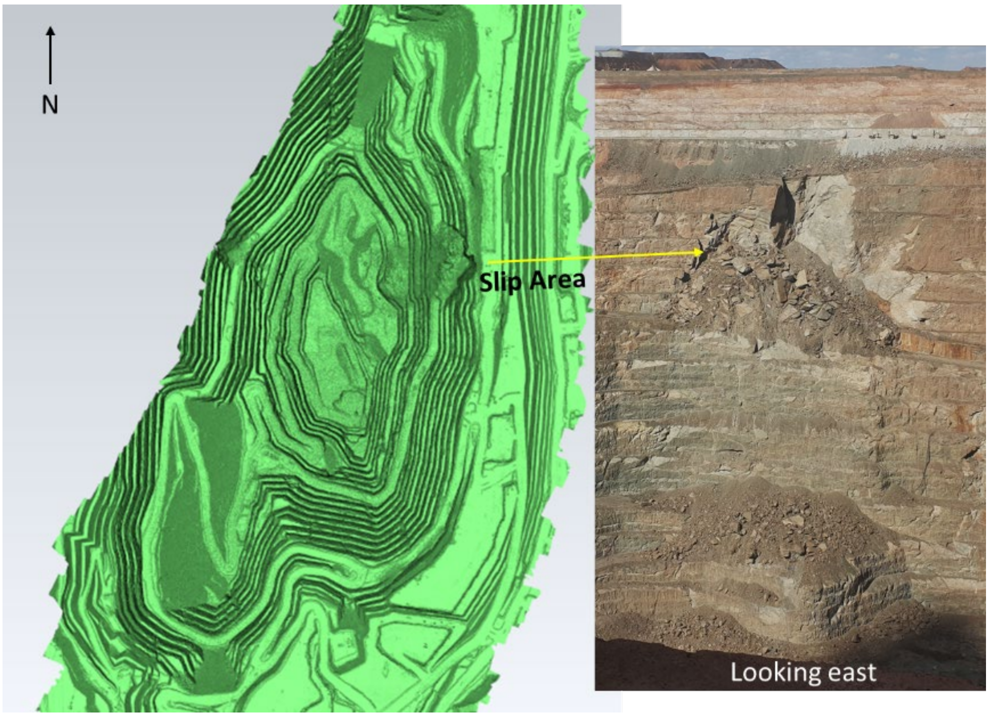


**FIG 2** – Location of East wall slip.

## Geology and major structures of the east wall

### *Geology and rock quality*

The East wall slip area (Figure 2) is located within the Paringa Basalt lithological unit (Figure 3a), which is comprised of two domains, 8 and 9, differentiated on the basis of rock mass quality (Table 1). Rock mass quality is reduced in the vicinity of the local faults, with a very slight reduction in in Domain 8 when compared to Domain 9. The Paringa Basalt (PB) is a 400–700 m accumulation of basaltic meta-lavas with minor interbedded metasediments (Hao, 2021); basalt is tholeiitic and characterised by pillow and flow breccia textures with interflow sediments of varying thicknesses present between lava flows.

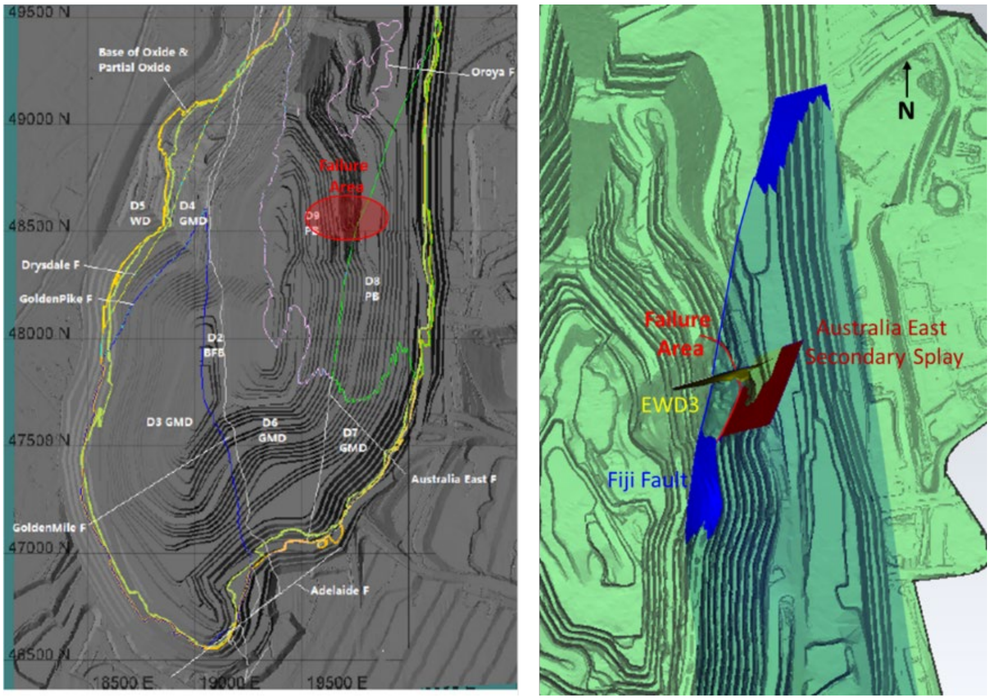


**FIG 3** – (a) Fimiston Pit geotechnical domains (Hao, 2021). (b) Major structures associated with East wall stability.

**TABLE 1**

RMR rating of Paringa Basalt domains (after Hao, 2021).

| RMR89 ranking | RMR89 quality class | PB – Domain 8 ALL | PB – Domain 8 FAULT | PB – Domain 9 ALL | PB – Domain 9 FAULT |
|---|---|---|---|---|---|
| 0–20 | Very poor rock | 0% | 0% | 0% | 0% |
| 20–40 | Poor rock | 1% | 3% | 0% | 2% |
| 40–60 | Fair rock | 8% | 31% | 5% | 20% |
| 60–80 | Good rock | 91% | 66% | 95% | 77% |
| 80–100 | Very good rock | 1% | 0% | 0% | 0% |

### *Major structures*

The main faults which contributed to the 2018 East wall failure are the Fiji Fault, the Australia East Secondary Splay and EWD3 Fault (Figure 3). The Fiji Fault is unhealed with 30–100 mm of graphitic clay gouge and persistence of over 1.5 km in a north and south direction (Hao, 2021), its typical dip/dip directions are 20–30°/240–260°. This correlates to an approximately 55° dip out of the pit wall (Figure 4). The Fiji Fault is extremely sensitive to pore pressure, which is considered a key driver for failure in the east wall (Wines, 2019).

The Australia East Secondary Splay splays off the Australia East ductile shear zone, which is 1–2 m thick and persistent over 1–2 kms (personal communication Y. Hao 2022). The EWD3 Fault is a subvertical structure with some gouge present and persistent over 100–200 m in the pit walls (personal communication Y. Hao 2022).

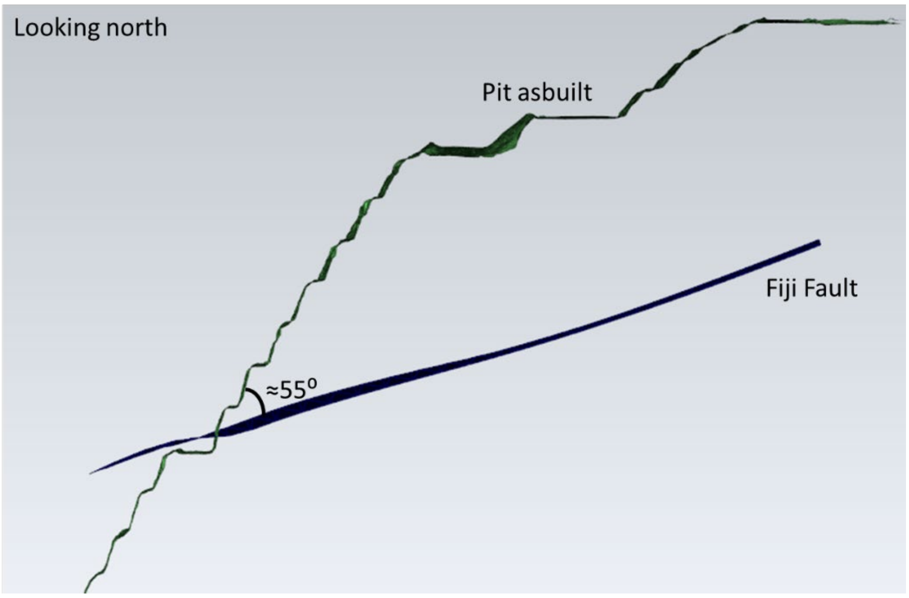


**FIG 4** – Cross-section showing dip of Fiji Fault out of pit wall.

## History of failure on the East wall

A series of failures have been observed around the east wall (Figure 5). Each of the observed failures have been associated with major structures exposed in the pit walls, and each have also been triggered by transient pore pressures on those major structures. Each failure was identified by slope stability monitoring radar and managed in an operationally safe manner. With regard to Figure 5, the key observations of each event were as follows:

### *2011*

- Failure in March 2011 occurred after heavy rainfall, with approximately 80 000 t of material movement. This resulted in the ramp below being blocked until the failed material was removed.

### *2018*

- Two failures occurred on 14/15 May, with approximately 1 Mt of material movement. The ramp above was rendered unusable.
- The ramp below was blocked, with material reporting to the pit floor. The failure was identified by the *in situ* slope stability monitoring radar.

### *2020*

- Approximately 70 000 t fell on 25 Feb after heavy rainfall. The pit was already closed due to rainfall ie there was no personnel exposure.
- Several rocks breached the 2018 exclusion zone.
- The majority of material was retained on rill from previous failures.

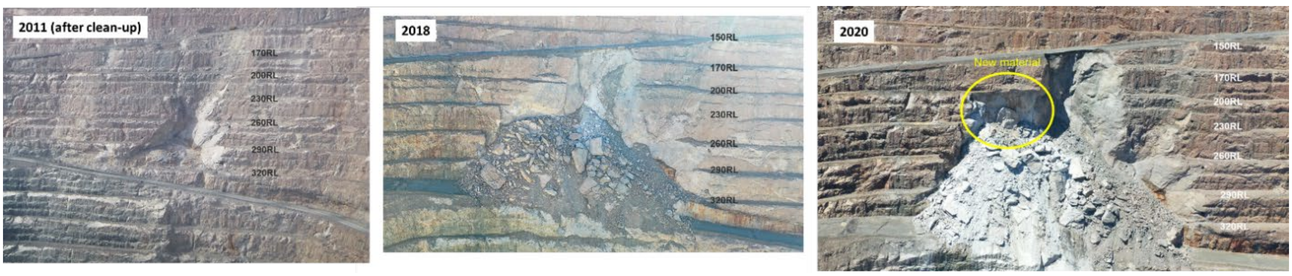


**FIG 5** – History of failure.

## REMEDIATION OF SLIP AREA

Remediation of the slip area will predominantly be undertaken by means of a planned cut back behind (to the east) of the failed area. The aim of this cut back is to eliminate intersection of a significant structure identified in the pit wall, the Australia East Secondary Splay, and improve the orientation of the pit wall where it intersects another, the Fiji fault (Figure 6).

The EWD3 Fault will remain in largely the same relative orientation (perpendicular) to the new cut back but is not expected to cause stability issues.

Given the role which transient pore pressure has played in previous failures a suite of 150 m to 250 m length depressurisation holes are planned to be drilled in the vicinity of key structures. A custom-built remote drill rig, capable of horizontal directional drilling, is currently under construction and will be supplied by SRG Global. Hole collars are currently planned approximately every 200 m along strike and 40 m vertically, with a splay of four to five holes drilled from each location.

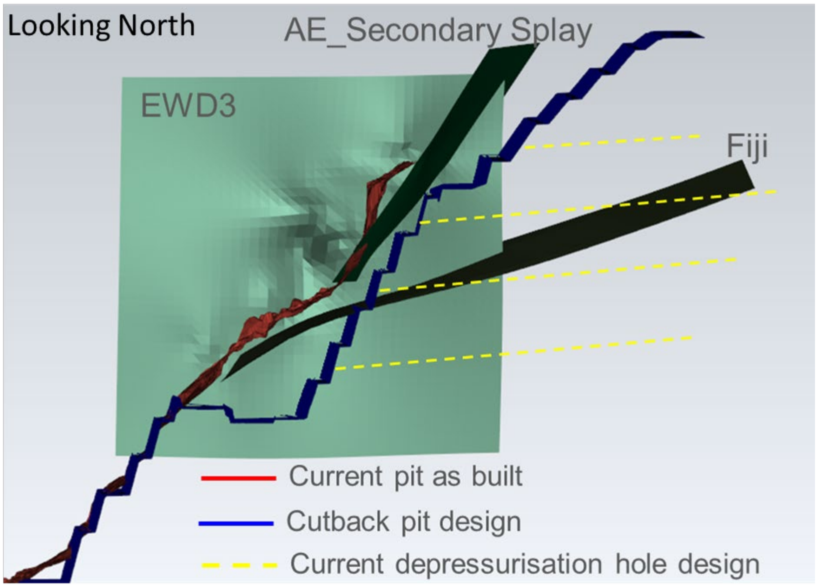


**FIG 6** – Planned cut back cross-section showing major structures and planned depressurisation holes.

## CONTROLS DURING REMEDIATION

The cut back and depressurisation were assessed as the critical controls against future possible failures, similar to that observed in 2011, 2018 and 2020. Execution of the planned cut back and depressurisation required detailed planning, with risks and controls identified at each step. The term ‘Control Zone’ is used to define the active working areas where specific controls have been put in place, with the term ‘exclusion zone’ used to refer to ‘No Entry’ areas. The ‘Area Controller’ is the designated area manager. All risks were assessed in terms of the safety hierarchy of controls (Figure 7) and are discussed on this basis below.

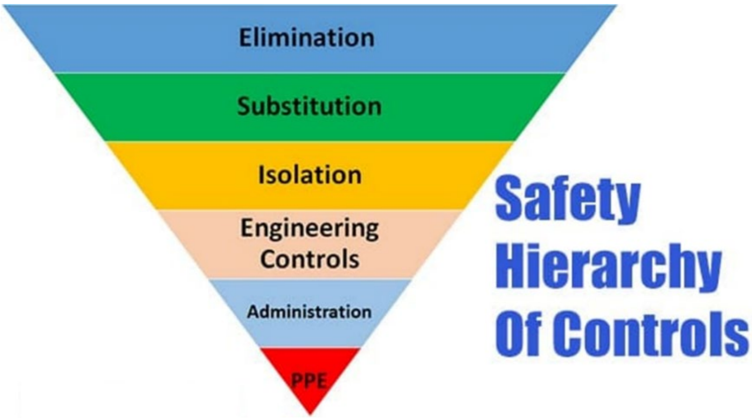


**FIG 7** – Safety hierarchy of controls.

## Engineering control – depressurisation drilling

Figure 8 shows the currently approved pit design, with each of the red points representing a splay of three to five 150 m to 250 m long depressurisation holes. Currently planned locations result in approximately 75 000 m of drilling over the life of the pit. Depressurisation holes will be complimented by Vibrating Wire Piezometers (VWPs) installed around the perimeter of the pit to verify the effectiveness of depressurisation, with design varied as required based on the gathered data.

The depressurisation holes will be drilled by a remote capable Vermeer D60–90 horizontal directional drill rig (Figure 8) supplied by SRG Global.


**FIG 8** – Depressurisation hole location and drill rig.

## Isolation – lower exclusion zone

An exclusion zone was delineated after the 2018 failure, and to date remains in place (Figure 9). This exclusion zone will remain in place during the mining of the East wall cut back; the exclusion zone is designated as a 'No Entry' area.

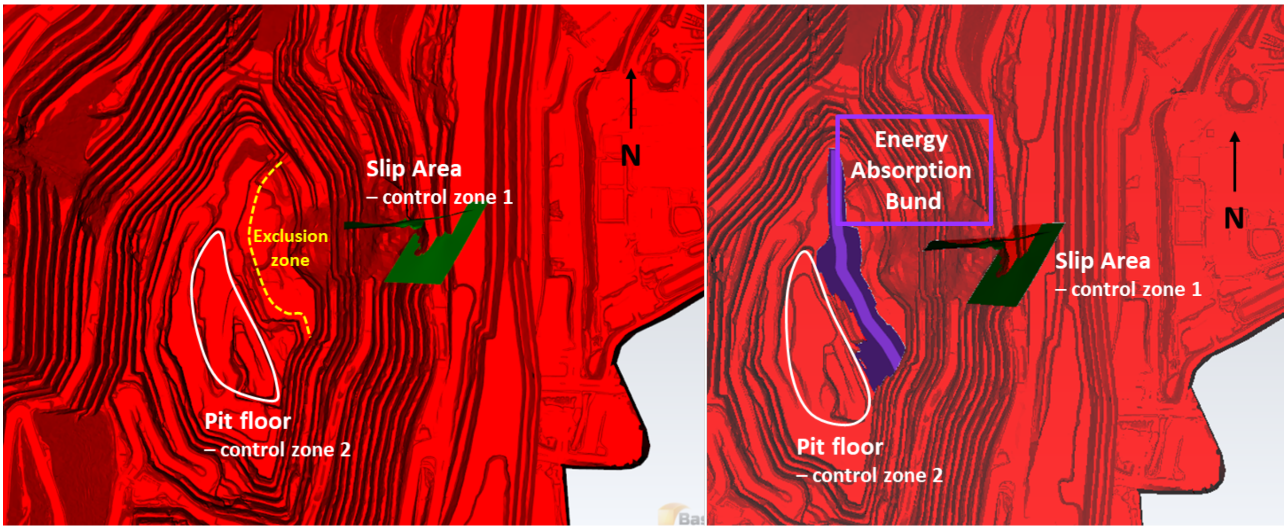


**FIG 9** – *In situ* exclusion zone, Energy Absorption Bund (EAB), and planned control areas.

## Engineering control – Energy Absorption Bund (EAB)

Figure 9 shows the planned energy absorption bund. Rockfall analysis was conducted and determined the risk of a rock falling from the vicinity of the slip and breaching the exclusion zone was ≈1 per cent (Figure 10). This was based on rockfall modelling using the program Trajec3D, version 1.7.2.7, from BasRock. Whilst a 1 per cent residual risk of rockfall breaching the exclusion zone, could, arguably be called as low as reasonably practicable (ALARP); Initial workshops between site personnel and Perth based technical specialists identified the potential to further reduce the risk by constructing a 15 m high bund along the perimeter of the exclusion zone (Figure 9).

Additional modelling undertaken to assess the efficacy of a 15 m high bund determined that residual rockfall risk could be practically reduced to ≈0.4 per cent (Figure 10).

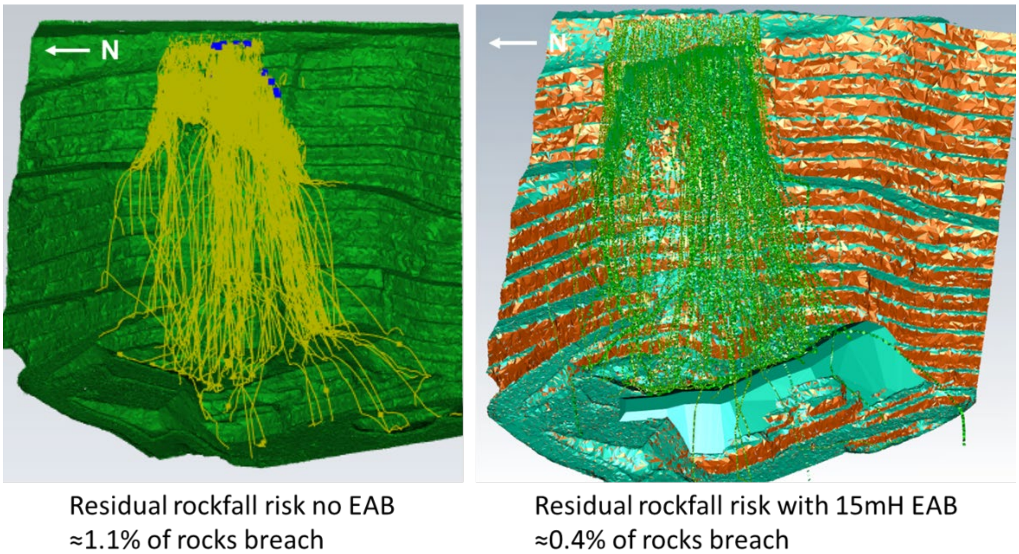


**FIG 10** – Residual rockfall risk.

## Isolation – upper and lower control zones

The exclusion zone will be complimented by the use of upper and lower control zones.

The perimeter of the upper control zone (Control Zone 1) will be based on the location of the Australia East Secondary Splay, EWD3 – which both remain exposed in the initial mining phase of the cut back (Figures 8 and 9) – and the geometry of the 2018 failure backscarp. The boundaries of Control Zone 1 will be adjusted as the location of the exposed faults shifts with changing relative level (RL) (Figures 8 and 9). Within the area, however, the controls remain consistent, including:

- No pedestrians (as much a reasonably practicable)
- Minimise use of water
- Utilisation of remote equipment.

The boundaries of the lower control zone (Control Zone 2) will also vary, depending on activity and the location of mining. The primary purpose for control zone 2 will be to restrict access to the pit floor during high-risk activities and during/after trigger events for rockfall.

For purposes of mining the East wall cut back high risk activities are classed as:

- Drilling within Control Zone 1 – where *in situ* wedge remains (Figures 7 and 8).
- Drill pattern preparation and pit edge clean up within Control Zone 1.
- Mining – removing material as part of production cycle – in Control Zone 1.

Potential trigger events for rockfall are:

- Firing of blasts, whilst initiating explosives and the period immediately thereafter.
- Rainfall, or any other use/application of water, which may result in a transient pore pressure on key structures.

## Isolation – use of remotely operated equipment

As outlined above, remote equipment will be used within Control Zone 1. The majority of the required equipment is currently available within the company/on-site and includes:

- **Remote Underground (UG) loader**: This will be used for pattern preparation, pit edge clean up and the removal of material within the control zone which cannot be reached with the shovel/excavator fleet.
- **Remote drill rigs**: Used on-site to drill area where historic workings are exposed/in close proximity to the pit floor. It will also be used within the control area.

Other equipment being developed/sourced specifically for the East wall cut back includes:

- **Personnel basket crane attachment to allow remote charging of drilled holes**: This is being specifically engineered to remove the need for personnel to enter Control Zone 1 at any point in the mining sequence,
- **Webgen remote detonation system (Figure 11)**: This system includes a wireless primer, encoder controller, initiation system transmitter, transmitter controller and antenna which eliminates the need for shots to be manually tied in once holes are loaded with explosives; It is currently being trialled on-site.

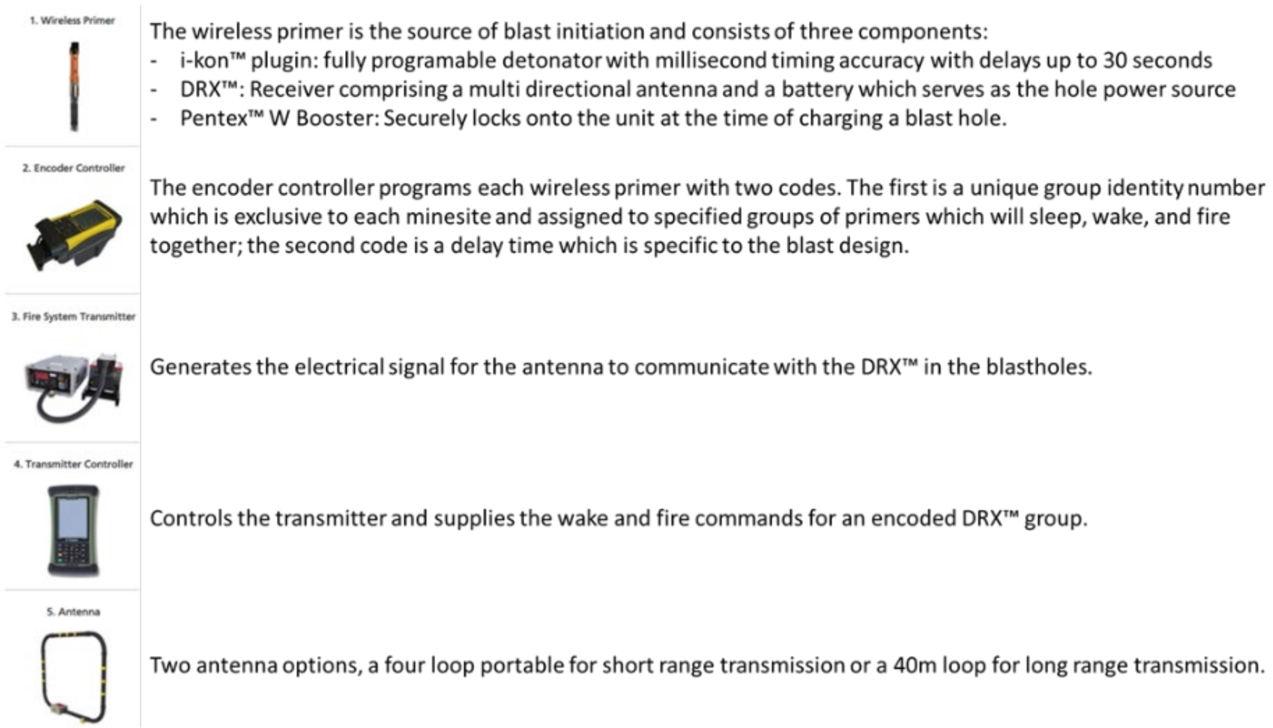


**FIG 11** – Webgen wireless detonation system.

## Administrative – Area Controller

It was recognised early in the planning process that a cut back of the complexity of the Fimiston Pit East wall, with many different control mechanisms and requirements, had high potential to result in both production losses and safety issues, if not managed correctly.

It was decided to second a senior member of the site engineering team into the role of Area Controller with the primary purpose of ensuring safe production during the entire period of mining the East wall cut back. Key duties and responsibilities will include:

- Liaise with key internal and external stakeholders to obtain and design the required remote equipment eg crane personnel basket.
- Liaise with key stakeholders re: manning and training of personnel for remote equipment.
- Maintain, update and drive compliance to risk assessments and work instructions.
- Maintain and update Control Zones 1 and 2 – including the 'Geotechnical Interaction Checklist' (Figure 12) which is to be completed daily.
- Communicate to key workgroups to ensure understanding of key objectives at each mining stage.
- Overarching supervision/checks for EAB design and execution.
- Maintain and update project plan/schedule and communicate key timelines and variances.
- Liaise with Drill and Blast team re: Pit edge drill patterns and blasting requirements.

**Geotechnical Interaction Checklist**

To be completed by the OBH Area Controller OR Production Geotechnical Engineer and Senior Shift Supervisor

Date: ____________

Upper shot number and planned activity ____________

Lower shot number and planned activity ____________

| | | |
|---|---|---|
| Has a visual assessment been undertaken? | Yes | No |

Comments: ____________

| | | |
|---|---|---|
| **Have any of the following been observed:** | | |
| Significant loose material build up on berms in area? | Yes | No |
| Water running from walls / major structures? | Yes | No |
| Dilation on foliation / major structures? | Yes | No |
| Visual signs of rockfall? | Yes | No |
| Have slope stability monitoring systems been checked? | Yes | No |

Comments: ____________

| | | |
|---|---|---|
| Has there been any rainfall recorded in the last 24hrs? | Yes | No |
| Has pit edge blasting been undertaken above area in last 24hrs | Yes | No |

**Sign-off**

OBH Area Controller / Geotech Eng ____________ Senior Shift Supervisor ____________

| | | |
|---|---|---|
| Approval for planned activities | Yes | No |

Geotechnical Superintendent OR Open Pit Development Superintendent OR QM

**FIG 12** – Geotechnical Interaction Checklist.

## CONCLUSIONS

The East wall of Fimiston Pit, has a history of significant structurally controlled wall failure, often triggered by the presence of water on/in key structures, notably the Fiji Fault and Australia East Secondary Splay

A cut back is proposed to mine behind the remnants of the 2018/2020 failures. Initial mining is already underway with the controls described in this paper to be in place/come into effect from the -150RL in July 2022. The design of the cut back will eliminate the intersection of the Australia East Secondary Splay with the pit wall, and result in an improved orientation of intersection between the slope and the Fiji Fault. As water pressure was found to be a key variable in East wall failures a suite of depressurisation holes have been planned to further reduce the risk of future instability.

To ensure that the planned cut back is mined safely and in a timely manner a number of controls have been put in place, or will remain in place after 2018 failure, including:

- An exclusion zone put in place, on the then pit floor, after the 2018 failure.
- An Energy Absorption Bund (EAB) along the perimeter of the exclusion zone to reduce residual rockfall risk from ≈1 per cent to ≈0.4 per cent.
- Control zones on the pit edge of the cut back and on the pit floor.
- Use of remote equipment such that personnel do not have to enter the upper control zone (Control Zone 1) during the mining process.

- A senior member of the site engineering team seconded into the position of area controller to manage safe production.
- Ongoing use of slope stability radar to monitor wall movement and the restriction of access during/after trigger events.
- Depressurisation drilling and monitoring of effectiveness using Vibrating Wire Piezometers (VWPs).

## ACKNOWLEDGEMENTS

The authors would like to acknowledge Northern Star Resources Ltd for providing technical review of this paper, in particular the thoughtful comments of Mr. Simon Thomas, and permission to publish.

They would also like to acknowledge the hard work and dedication of site-based personnel at all levels who have worked, and continue to work, towards safe extraction of the East wall cut back. The technical knowledge and work of Dr. Adam Beer and Mr. Yu Hao, of Northern Star Resources Ltd Thunderbox and KCGM Operations respectively, have been invaluable in this regard.

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# Smart mines

# Inferring the earth moving equipment–environment interaction in open pit mining

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## ABSTRACT

In mining, grade control generally focuses on blasthole sampling and the estimation of ore control block models with little or no attention given to how the materials are being excavated from the ground. In the process of loading trucks, the underlying variability of the individual bucket load will determine the variability of truck payload. Hence, accurate material movement demands a good knowledge of the excavation process and the buckets' interaction with the environment. However, equipment frequently goes into off-nominal states due to unexpected delays, disturbances or faults. The large amount of such disturbances causes information loss that reduces the statistical power and biases estimates, leading to increased uncertainty in the production. A reliable method that inferences the missing knowledge about the interaction between the machine and the environment from the available data sources, is vital to accurately model the material movement.

In this study, a twostep method was implemented that performed unsupervised clustering and then predicted the missing information. The first method is Density-based spatial clustering of applications with noise (DBSCAN) based spatial clustering which divides the diggers' and buckets' positional data into connected loading segments. Clear patterns of segmented bucket dig positions were observed. The second model utilised Gaussian process regression which was trained with the clustered data and the model was then used to infer the mean locations of the test clusters. Bucket dig locations were then simulated at the inferred mean locations for different durations and compared against the known bucket dig locations. This method was tested at an open pit mine in the Pilbara region of Western Australia. The experimental results demonstrate the advantage of the proposed method in inferencing the missing information of bucket-environment interactions and therefore enables miners to continuously track the material movement.

## INTRODUCTION

Mining operations look for new approaches to maximise the ore and minimise the waste recovery at the process plant. The mining process includes drilling, blasting, loading and hauling and primary crushing. Having problems in one stage may lead to inefficiencies in the downstream process and therefore impact the production outcome. Among the mentioned steps, loading efficiency plays a vital role in increasing production and reducing costs as the loading equipment is the source of ore supply or waste removal (Balamurali, 2022; Vasylchuk and Deutsch, 2015).

Hydraulic excavators are primarily used as loading equipment in large open pit mining operations around the world. In the process of loading trucks, the underlying variability of the individual bucket loads will determine the variability of truck payloads. Hence, accurate material movement demands a good knowledge of the excavation process and the buckets' interaction with the environment. The main hypothesis of this research is that these excavators can be used as a diagnostic tool to continuously and accurately estimate the moments (mean and variance) of the material loaded to each truck and therefore infer the propagation of material uncertainty down the material tracking pipeline. It can be achieved by knowing the time of engagement and disengagement of the bucket from the ground and its positional data coming from the sensor measurements. GPS sensors used on excavators facilitate collecting and analysing spatial and temporal information of the excavators and buckets during loading and dumping. Knowing the exact bucket dig locations helps the mining operators to infer the actual material taken from the ground by matching the grade control block models and the bucket dig locations.

However, equipment frequently goes into off-nominal states due to unexpected delays, disturbances or faults. A large amount of such disturbances causes loss of information about where the bucket interacts with the ground and thus reduces the accuracy in the estimation of the loaded material, leading to increased uncertainty in the production. In order to accurately model the material movement, it is crucial to have a reliable method that can infer the missing information regarding the bucket dig positions.

This paper proposes a two-step approach to infer the missing bucket dig locations and therefore help to estimate the moments of the material that goes on to trucks. Before carrying out further analysis on the data, splitting bucket dig positions into connected loading segments is necessary in data processing. This research proposes an unsupervised clustering method, DBSCAN, to group the connected bucket locations for each of the nearest excavator's GPS locations. The DBSCAN algorithm was originally developed by Ester *et al* (1996) for clustering spatial points based on density difference.

The next step uses the Gaussian process regressor to learn the mapping between the excavator's GPS positions and the corresponding clusters' moments. The trained model is then used to infer the moments of missing clusters. Bucket dig locations are then simulated at missing clusters using the inferred moments of clusters. This twostep method was tested in a region at the Pilbara iron ore deposit situated in the Hamersley Province, Western Australia.

# METHODS

## DBSCAN

A point is considered crowded if it has many other neighbouring points near it. The DBSCAN finds these dense points and places them and their neighbours in a cluster. The DBSCAN has two main parameters. The parameter ε (epsilon) defines the radius of a neighbourhood with respect to some point. The next parameter 'MinPts' is the density threshold. If a neighbourhood includes the MinPts points, it will be considered as a dense region.

Consider a set of points in some space to be clustered. For the purpose of DBSCAN clustering, the points are classified as core points, (density-) reachable points and outliers, as follows:

- A point p is a core point if at least MinPts points are within distance ε of it (including p).
- A point q is directly reachable from p if point q is within distance ε from core point p. Points are only said to be directly reachable from core points.
- A point q is reachable from p if there is a path $p_1$, ..., $p_n$ with $p_1$ = p and $p_n$ = q, where each $p_{i+1}$ is directly reachable from $p_i$. This implies that the initial point and all points on the path must be core points, with the possible exception of q.
- All points not reachable from any other point are outliers or noise points.

Now if p is a core point, then it forms a cluster together with all points (core or non-core) that are reachable from it. Each cluster contains at least one core point; non-core points can be part of a cluster, but they form its "edge", since they cannot be used to reach more points.

This clustering method does not need prior information of the number of clusters as input. The definitions and detailed parts of applying DBSCAN on GPS data can be found in Gong *et al* (2015).

In this study, the input variables to the DBCAN are the x, y and z coordinates of buckets' dig and dump positions and x and y coordinates of the excavator's GPS positions in local mins grid.

## Gaussian Process Regression (GPR)

GPR is a generic supervised learning method designed to solve regression and probabilistic classification problems. The spread of the bucket locations within each cluster is estimated by calculating the mean and standard deviation of each cluster in both the x and y directions. The estimated moments are then used as target values for training the GPR. GPR learns the mapping between the excavator's GPS positions and the target values. The trained model is then used to infer the mean and standard deviation (in both x and y directions) of each unknown cluster using the given GPS positions of the excavator. The scikit-learn implementation for Gaussian Process Regressor was used in this study. Scikit-learn is an opensource package that provides various machine learning tools written in python (Pedregosa *et al*, 2011).

## RESULTS AND DISCUSSION

### DBSCAN to find the representative clusters

DBSCAN is used to group bucket dig positions (x, y and z coordinates) into clusters (representative bucket dig locations) and noise due to their difference in spatial density (Figure 1b). Figures 1c and 1d show the plan and vertical view of bucket dig locations after removing the outliers from Figure 1b. Each representative cluster is associated with its closest excavator's GPS positions (Figure 1a). Figure 1 shows that well separated clusters are achieved in both horizontal and vertical planes using DBSCAN.

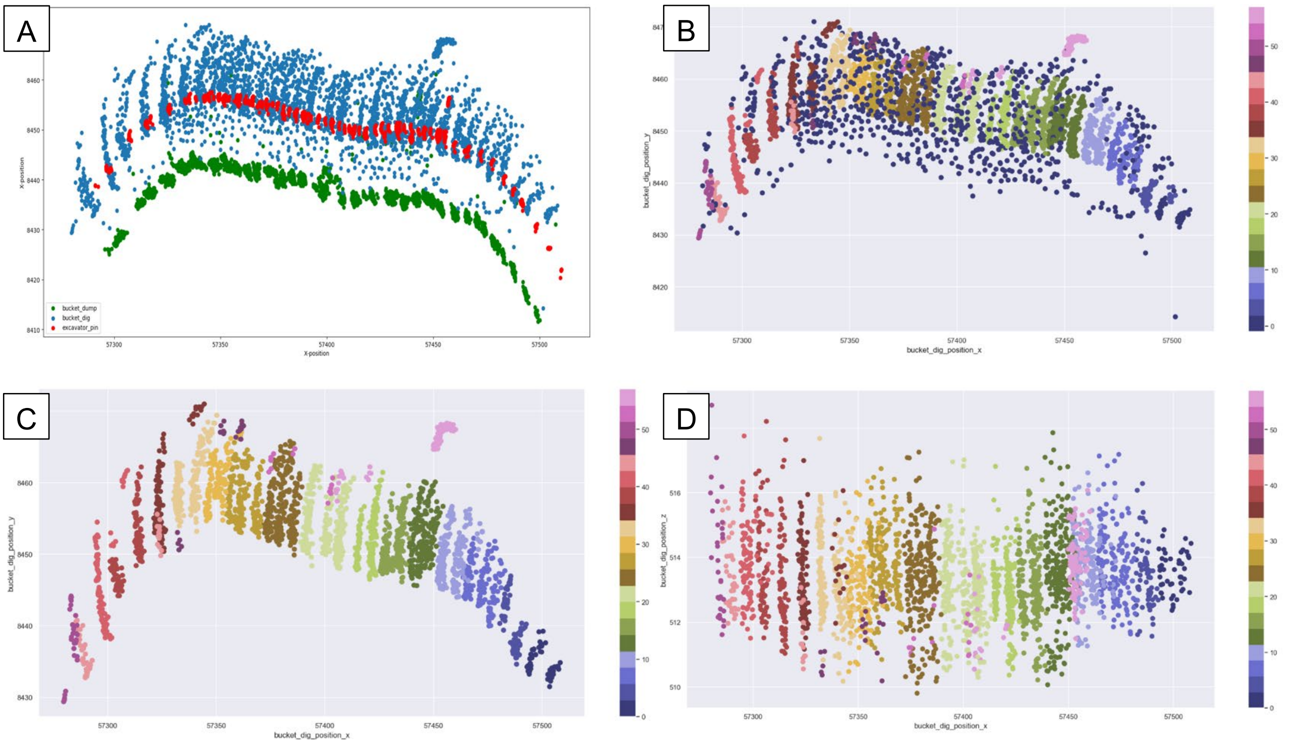


**FIG 1** – (a) Shows the plan view of bucket dig locations, excavator's GPS location and bucket dump locations in the test region. (b) Shows the segmented bucket locations and noise data. Each segment is coloured according to the cluster labels. (c) and (d) Show the clusters on plan and vertical view after removing the noise.

### Inferencing the representative bucket dig location

#### *Scenario 1 – prediction on random location*

In scenario 1, during the training process some of the moments of clusters were removed at random locations from the training samples and the model was trained with remaining samples. As can be seen in Figure 2a, the trained model was then used to infer the unknown moments at random locations in both the x and y directions using the excavator's GPS positions. The visual assessment in Figure 2a shows that the inferred values are reasonably closer to the ground truth values.

Using the predicted mean and variance, Gaussian samples were simulated at the missing locations in x and y directions and the bucket dig locations that fell in each removed cluster were recreated. Figure 3b shows the plan view of the Gaussian simulated bucket dig locations at the inferred mean clusters' locations and Figure 3c compares the simulated bucket dig locations with the ground truth bucket dig locations.

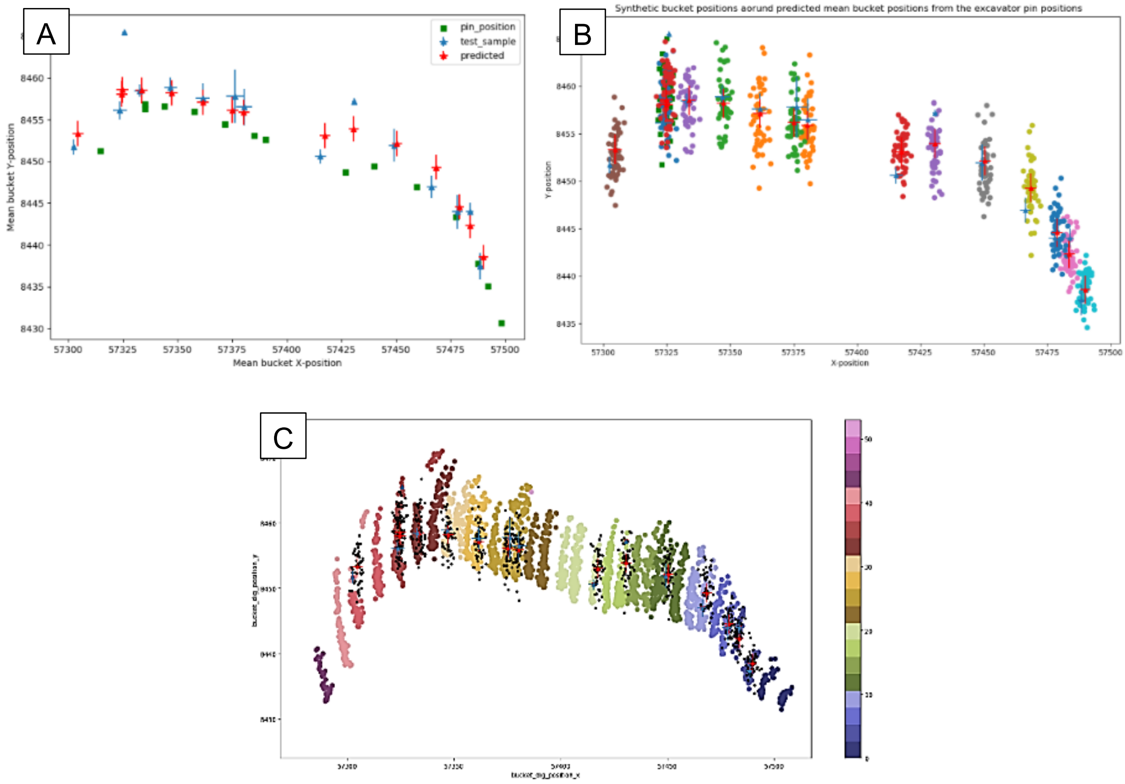


**FIG 2** – The plan view of (a) ground truth and predicted moments of clustered bucket dig locations and excavators' pin position. (b) The simulated bucket locations and (c) compares the simulated locations (black) with recorded bucket locations (locations are coloured according to cluster labels).

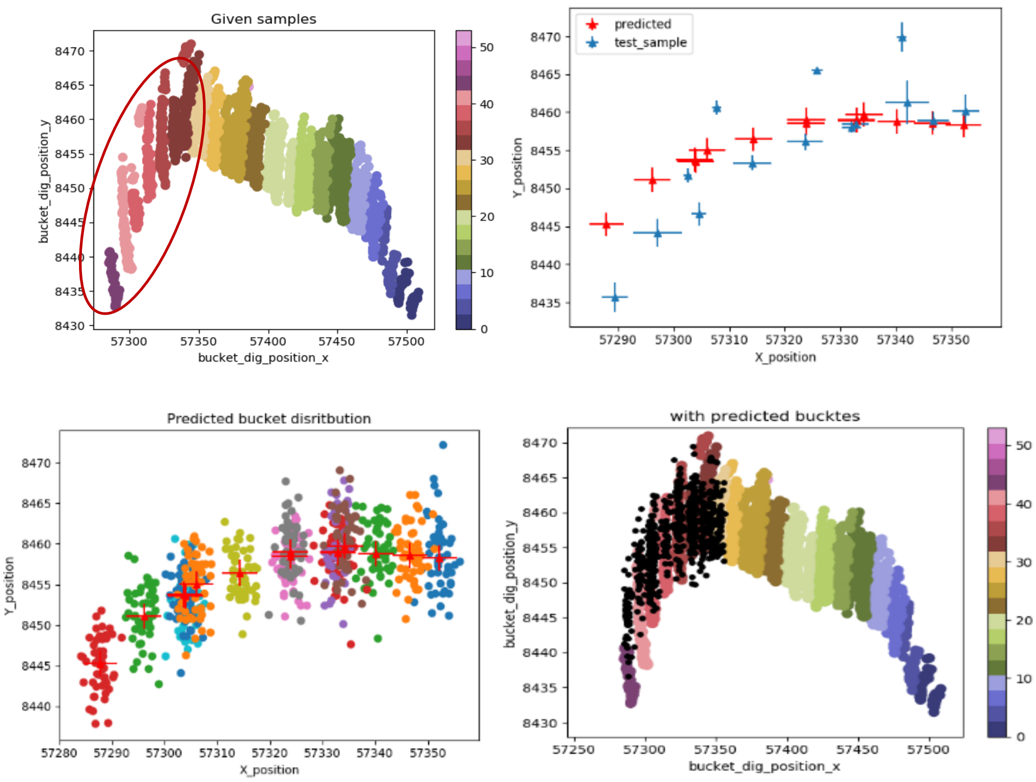


**FIG 3** – The plan view of (a) given bucket dig locations coloured by its cluster labels; (b) shows the inferred moments of the test cluster samples and the ground truth values in the circled area of (a). (c) shows the simulated bucket locations; and (d) compares the simulated locations (black) with given bucket locations (locations are coloured according to cluster labels).

### *Scenario 2 – prediction on continuous locations for a longer period*

Scenario 2 was implemented to evaluate the accuracy of the model prediction for a longer period. As can be seen in Figure 3a, a continuous bucket dig locations were removed for a longer period and the corresponding cluster moments were removed from the training samples. GPR was then trained on the remaining samples and the model predicted the moments on test locations.

Figure 3 compares the inferred clusters' moments on continuous timestamps and simulated bucket dig locations with the ground truth values. Figure 3b shows that the distance between the inferred means and the ground truth cluster means increases when the test locations are further away from the training locations. This might be due to the large time interval as the model has not been sufficiently trained to make predictions for longer periods or due to the steep change on the excavator's path where this kind of pattern hasn't been captured in the trained model. Further studies with longer sequence data at different regions and large excavators' GPS data during loading, can improve the model prediction accuracy. Even though the distance between the inferred and ground truth cluster means has increased with the steep curve of the excavator's travelling path (Figure 3b), the simulated bucket dig locations can still form clusters closer to the clusters previously obtained by DBSCAN.

This study shows that the accuracy of the model predictions is high on random locations when the model is trained on closer samples for shorter time periods compared to the predictions made for longer periods on continuous locations. However, in both circumstances, the simulated bucket locations form the clusters that are reasonably close to the clusters from the actual bucket dig locations.

Knowing the exact bucket dig locations helps the mining operators to infer the actual material taken from the ground by matching the grade control block models and the bucket dig locations. However, imperfect mining such as blasting, sheeting and grading of roads and benches will move ore and waste around the boundaries. Due to geometry, the operation of excavation equipment cannot be physically matched with the shape and size of the orebody, and therefore excavators blend the material further (Isaaks, Treloar and Elenbaas, 2014; Dimitrakopoulos and Godoy, 2014; Verly, 2005). Hence, the bucket dig locations are recorded in a way that makes it highly uncertain in knowing the exact material taken from the ground.

In this case, bucket dig locations simulated by the proposed model can still be used as bucket – material interaction points when a large number of bucket dig locations are missed and we need to know the distribution of the material loaded onto trucks from the corresponding locations. However, the uncertainty associated with the material taken from the inferred dig locations will be larger compared to the uncertainty of the loaded material when the dig locations are known.

The proposed model can be further improved by inferencing the representative load dig locations for each truck instead of inferencing large connected bucket dig locations (clusters) that correspond to the excavator's GPS positions. However, it would be more accurate if the model could infer each individual missing bucket dig location. This can be achieved by including more samples collected from larger study areas and other information coming from the sensors used in excavators and trucks. Using different spatial temporal clustering methods such as ST-DBSCAN and Hidden Semi Markov Model (HSMM), can improve the segmentation of bucket dig locations corresponding to each truck. Incorporating a time variable helps to identify sudden changes. Sudden increases can be observed when the excavator moves forward to the next load point or is waiting for the next truck. If such a 'sudden increase' is found, the cluster can be divided into two clusters at the point of sudden increase. This information helps to further refine the cluster results. These are subject to further investigation.

## CONCLUSION

A model approach is presented which can be used to infer the missing bucket dig locations when the bucket sensors frequently go into off-nominal states due to unexpected disturbances or faults. Therefore, the proposed study facilitates to continuously tracking the material at subsequent locations in the material tracking pipeline.

Several limitations and future research directions in the methodology have also been noted. To address this, the proposed model should be improved by evaluating other clustering methods by incorporating both temporal and spatial variables coming from different load and haul equipment at the time of mining. The GPR method used to regress the location information in this study should also be compared with other regressors to confirm its superiority; meanwhile, other possible sensor information can also be included in the regressor to improve the prediction accuracy.


## ACKNOWLEDGEMENTS

This work has been supported by the Australian Centre for Field Robotics and the Rio Tinto Centre for Mine Automation, the University of Sydney.


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# Applying control frameworks to enhance vehicle interaction technology outcomes

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## ABSTRACT

Qualitative risk assessment methods are not well suited to considering complex systems and potential interactions of new technology.

As part of supporting tier one mining organisations and international industry groups, the author and colleagues have innovated processes to help provide better solutions to sites/companies considering a move to implementing proximity detection and collision avoidance systems. The process identifies required operating states for the system. By applying a combination of research and team-based analyses, a rigorous identification of credible failure modes which could cause these states to be lost, is identified.

Finally, required business inputs, expressed in practical terms are distilled from regulator, industry, organisation and site in a way which clearly addresses the failure modes and so retain the required operating states.

These business inputs identify improvements for site arrangements (EMESRT levels 1 to 7) **before** identifying functional requirements for technological collision avoidance solutions (levels 8 for advisory and 9 for autonomous).

The paper presents a case study synthesised from multiple sites analyses and will provide guidance on where to find additional resources to assist in making the best possible transition to better equipment and road network solutions.

## DEVELOPING CONTROL FRAMEWORK METHODOLOGY

### Identified need

Technology for mobile equipment is improving. Organisations that Risk Mentor works with are committed to implementing leading practice technologies for their mobile equipment fleets, based on operational, moral and legal drivers.

Choosing which technology to deploy is not straightforward and comes with a significant cost implication when solutions worth many thousands of dollars are installed on thousands of items of mobile plant. The industry is littered with tales of woe around great collision avoidance technology (as marketed) becoming just an annoying alarm and ultimately being defeated by operators or removed by maintainers.

#### *The challenge*

All Australian jurisdictions have legislation that requires companies to take a risk-based approach to their business. This requires going beyond any available standards and involving all levels of the company in making and refining decisions on the equipment, process and environments that work will take place in.

Currently popular risk assessment tools, applied in other sites and settings, do not typically provide any useful output for inclusion in purchasing or design decisions for mining equipment. Other authors explain this more eloquently defining the problems around ranking and issue identification detailed in texts by Hubbard (2009), Gigerenzer (2002) and Hogarth and Pooley (2015).

Given the size and scale of the purchasing decision to make on Collision Avoidance or Automation technologies for a large fleet it is important to treat this as a project. Good project management requires a systematic approach that:

- defines the objective

- analyses the current state in detail
- breaks out the requirements of the project elements
- plans and schedules activities to move to the objective
- continually monitors and improves the project processes
- transitions from project to operations.

The remainder of this paper addresses the first three of these points. For reference, a sample of a Work Breakdown Structure which can deliver on these requirements is presented in Figure 1. This structure shows that the 2nd and 3rd key project requirements and addressed through the preparation and validation of a control framework for the site.

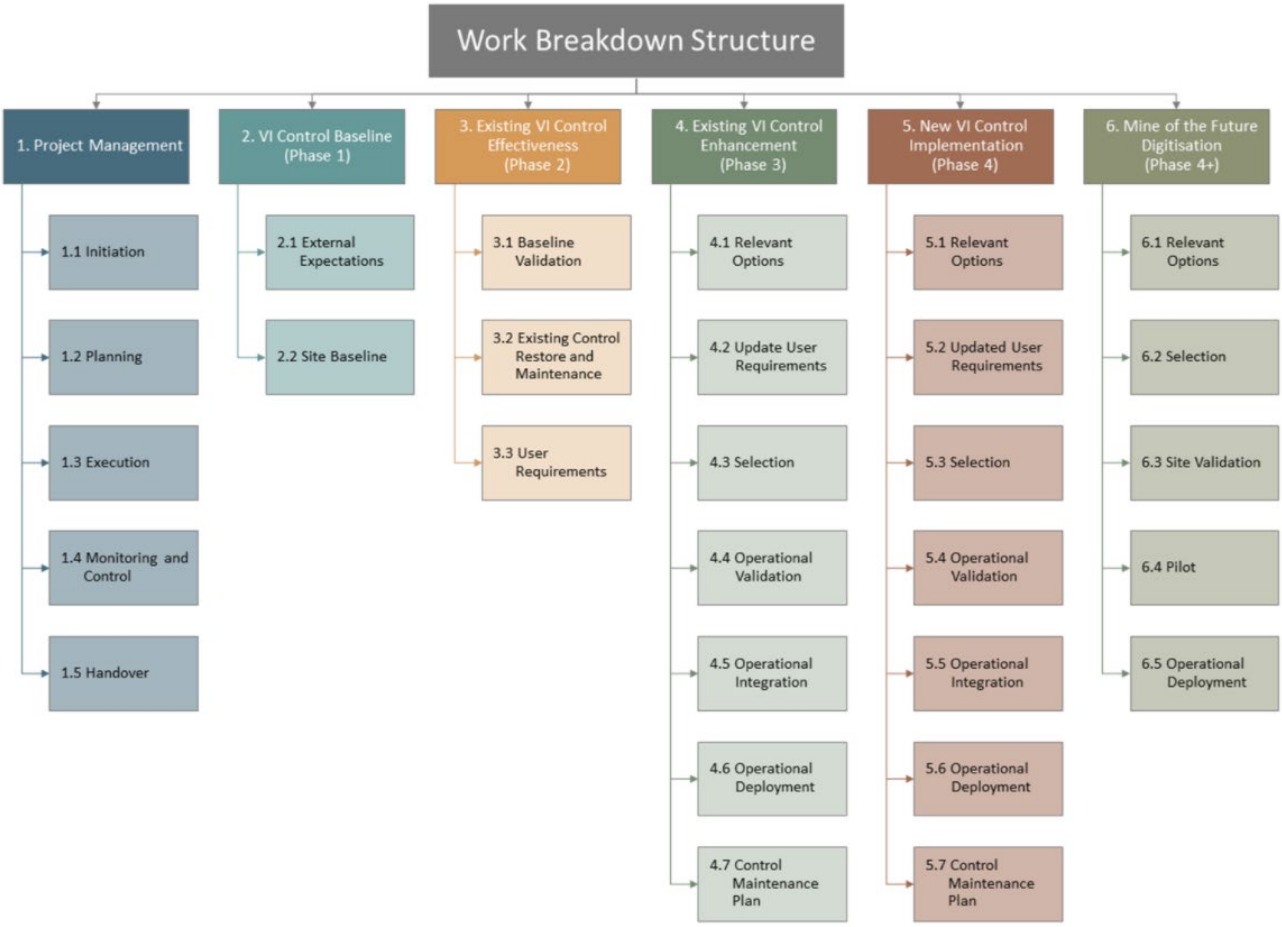


**FIG 1** – VI Project WBS (courtesy of EMESRT 2020a).

### *Techniques that have issues*

WRAC and Bow Ties had been applied by organisations to Vehicle Interactions and MISHC with input from the Australian mining industry (dominantly coal) had worked up their Risk Gate bow tie for the potential unwanted events (PUE) of Vehicle to Vehicle/Environment and Vehicle to Pedestrian incidents. This work generated a list of many hundreds of similar controls that a site should implement.

The controls, as documented both in Risk Gate and in risk assessment/bow ties, reviewed by the author do not meet the ICMM's guidance around being a control (Critical Control Management) (ICMM, 2015). Further, most controls as documented in Risk Assessments are terse to the point of being cryptic and it is hard to match the risk analyses with the site's requirements as documented in management plans, equipment specifications and procedures.

Working with our clients and EMESRT Risk Mentor reviewed other engineering standards to identify a possible way forwards for the risk analysis phase of the project. Taking input from AS IEC 60812–

2008 Failure Modes and Effects Analysis and MDG 2007 Guideline for the selection and implementation of collision management systems for mining the Control Framework approach was developed. This approach has now been validated at over 20 mining and industrial operations worldwide. EMESRT is the Earth Moving Equipment Safety Round Table, a group of industry representatives executing a charter to improve the safety and health of deployed machinery at mine sites.

### *Control framework explained*

Figure 2 gives an overview of the key elements in a control framework (CFw). Working in this way allows many requirements and issues to be included in a set of business inputs that cover: incident causes; statutory and company requirements; relevant standards, and; good practice guidance. A further description is provided in the EMESRT 2019 Annual Report (EMESRT, 2020a) at page 15.

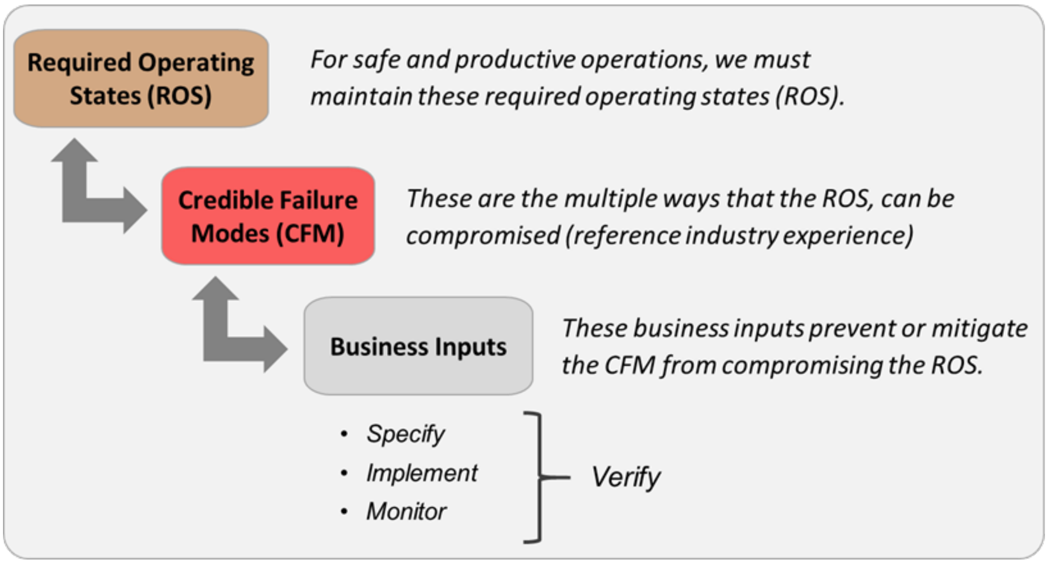


**FIG 2** – Outline of the control framework model.

A control framework for Vehicle Interactions is available at an industry level from EMESRT – and this forms a sound basis for developing a site-specific framework. The framework is anchored on a self-assessment of current performance, is refined to suit site requirements and informs the functional and performance requirements for any technology being considered.

### *Supporting specifications, designs and decisions*

To support the decisions which will extend to a decision on technology, each business input is considered in the context of what failure of required operating state is being addressed.

This is probably most easily understood through a specific example.

The required operating state is: *Vehicle operators give way appropriately to mobile plant and pedestrians.*

This state can be lost if the failure mode of: *Vehicle controller unaware of right of way requirements occurs.*

To address this requirement, the site should have a business input such as: *Give way requirements information prepared for mobile equipment operators and pedestrians* which covers the expectation of providing give way requirements being documented and communicated in a style appropriate for the audience.

When considered in this way – the complete arc of the decision at an intersection by a vehicle controller (ie an operator or an autonomous processing unit) has a lot of clarity.

## Terminology

There are some terms used in the CFw model that are explained in Table 1.

**TABLE 1**

Control framework terminology.

| Term | Description |
|---|---|
| Baseline | Required operating state before any additional technology is implemented. |
| Business Input | Planned intervention by the organisation which has a concise Expectation, actionable Specification, required method of Implementation and Monitoring requirements appropriately documented to suit site requirements. |
| CAS | Acronym for Collision Avoidance System |
| CFw | Abbreviation for Control Framework |
| Control | Can be considered as a business input – but for consistency should be a clearly defined human act, plant related feature (object) or a technological system (action of a human based on a signal from an item of plant). |
| Credible Failure Mode | Input from any source which compromises the required operating state |
| HSMS | Abbreviation for Health and Safety Management System |
| Nuisance Alarm(ing) | An alarm state that has limited or no context for understanding by the controllers/operators receiving the alarm. This includes excessive sounds, unclear messages and ergonomically poor control layouts. |
| Required Operating State | A state of the system which will prevent unwanted events in all circumstances. |
| Site Requirements | This is the combination of jurisdictional imperatives, company standards and unique features of the location and work activities on the site which constrain what should and should not occur. Site requirements extend to the people, equipment, wok environment and system elements in place. |
| V2V | Acronym for Vehicle to Vehicle data communication system deployed on many CAS systems so that vehicle trajectories are communicated between proximate vehicles. |

## Data analysed

A key part of the method is to draw on existing site and industry material. This 'front end loads' the analysis by:

- Reviewing a representative sample of site incident data – confirming and extending the CFw library of Credible Failure Modes for the analysis.
- Analysis of system documentation – identifying the active requirements of management plans, engineering specifications, procedures, training modules and other elements – and then updating the library of business inputs in the CFw to match this material.
- Mapping and linking the mandated elements of legislation and corporate imperatives to elements (dominantly business inputs) in the framework.

Typical output from a data analysis helps to guide towards which failure modes are most dominant in recorded site incidents. Figure 3 presents a typical output of such an analysis.

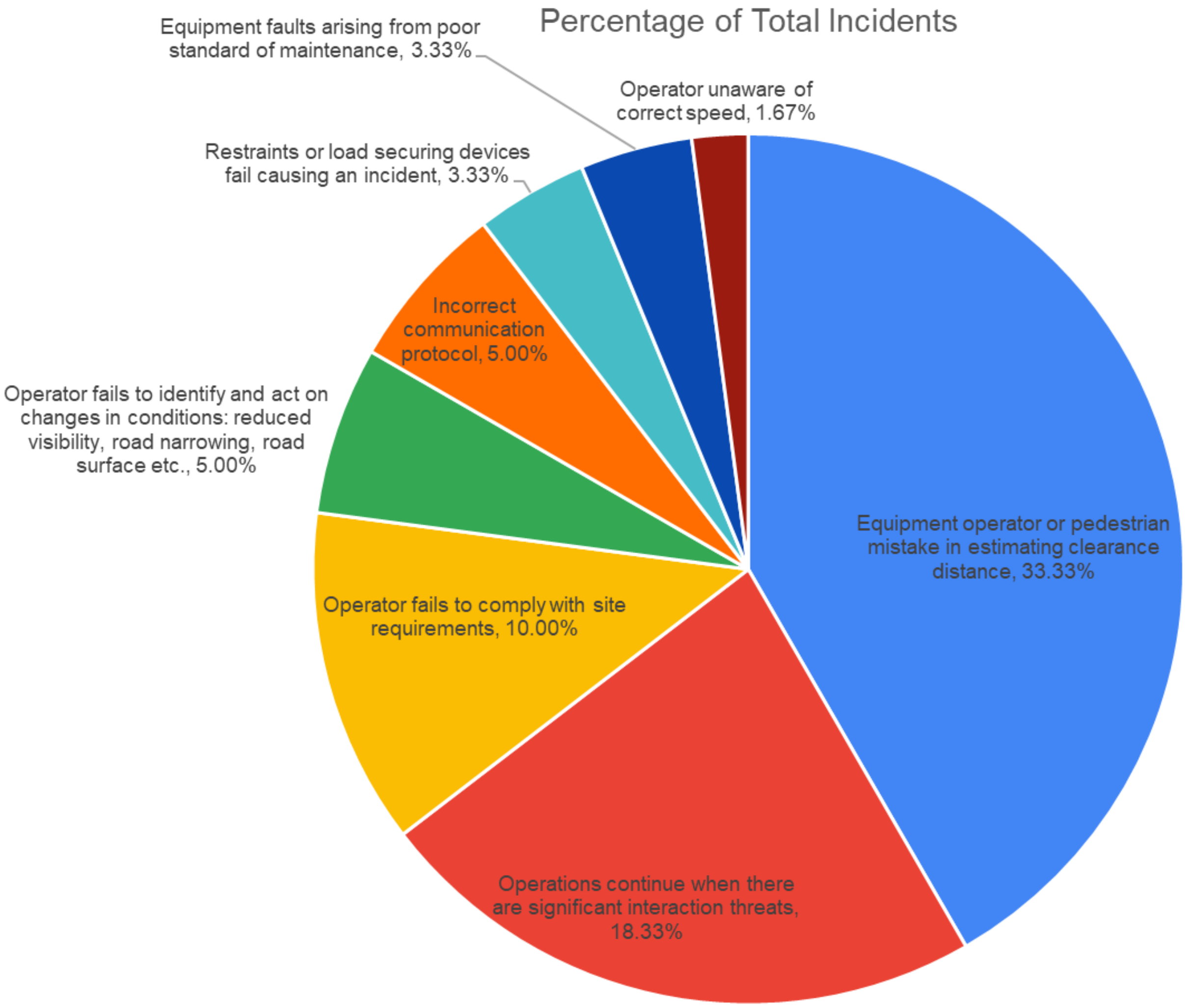


**FIG 3** – Typical Breakdown of Incidents by Failure Mode Type.

## Problems need solutions, not scores

The rigour of a control framework is that every failure mode has:

- A clear, logical basis for existing. Failure modes arise from incident reviews and science/engineering analyses of the operating state. They are not an artefact arising from a group discussion.
- One or more well described business inputs which completely constrain the failure mode.
- A known frequency of occurrence or calculated reliability/failure frequency.

### *Solutions/business inputs are resilient*

The business inputs described in the CFw have common elements:

- An expectation – which concisely states the purpose of the input and provides guidance on the why, where and when the input should occur. Business inputs should also have nominated accountable role holders – with carriage of confirming the input is in place across the business and responsible role holders who apply the input as and when required to constrain a failure mode.
- A clear specification. The specification describes what is required to meet the intent of the business input. Specifications vary according to the nature of the input. So:
    - Equipment – has processes for developing specifications for selection; ordering and pre-work before equipment arrives on-site; introduction to site to confirm delivered items meet requirements; maintenance strategies to match the OEM and site operating requirements;

associated system elements (training, physical conditions etc) to support effective deployment.

- Processes – match constraining the failure mode(s) they address, have requirements for documentation, training/assessment modules, and practical demonstration of understanding.
- Worksites/environments – are engineered, commissioned, maintained and monitored.
- Business systems – are designed to continually improve, amplify strategy, anchor site requirements, and enhance/maintain the complete set of business inputs.

- Implementation processes, which:
  - for equipment involve executing introduction to site and maintenance functions
  - embed processes through training and matching to skills
  - commission, maintain and monitor work environment conditions
  - implement system elements in line with site requirements.
- Improve decision-making regarding resources and interventions by monitoring (the least well-done aspect of all controls across industry), by:
  - reporting and analysing maintenance and field data from equipment
  - supervising and reviewing activity records of workers undertaking tasks
  - identifying trends and emerging requirements of conditions at worksites
  - auditing performance and analysing incidents/non-conformances to improve the business more broadly.

It is this combination of Purpose/Specification/Implementation/Monitoring and Verifying that makes the Business Inputs in a Control Framework resilient – so they can remain effective at preventing failure modes from upsetting the required operating state. Continually operating in these states makes for productive and safe businesses.

## Validation

Monitoring of business inputs occurs close in time and location. A validation process helps to confirm that the systems which support the timely and consistent performance of required inputs are healthy.

To achieve this there should be a requirement for a once removed (or greater) confirmation that equipment, supervisors and senior responses are occurring in line with site requirements. Eaton, Eldin and Song (2013) provides an overview of validation for safety measurement, and this is mirrored in guidance from McChrystal *et al* (2015) in establishing a level of oversight for operational activities.

## Informed technology decisions

With a level of clarity about what the business input is intended to be, then a statement of the requirements for the input's function can be developed. Informed in part by software developers in February 2021, a useful format for a Functional Requirement is a statement which provides a rationale and a brief summary (Wikipedia, 2022).

To this end, some sample functional statements are available for vehicle interaction at https://emesrt.org/ and they have the structure as shown in the following examples:

- As a Light Vehicle operator, I want to be alerted when my direction and speed of operation will encroach on the zone of influence of a heavy vehicle so I can change my path, request permission to encroach or slow or stop my movement.
- As a Heavy Vehicle operator, I want to be alerted when a light vehicle is within my zone of influence so that I can cease moving and place my vehicle into a safe state.

- As a Supervisor of vehicle operations I want to be alerted when a light vehicle has breached a heavy vehicle's zone of influence without permission so that I can prevent a recurrence of the event through a better understanding by workers involved or layout/organisation of travel routes.

With this level of clarity it is then possible to develop performance requirements for any technology being employed – which allows for higher quality equipment selection processes. These processes reduce the potential for nuisance alarms.

## CONTROL FRAMEWORKS TO UNDERPIN TECHNOLOGY – A TYPICAL CASE STUDY

A large surface mine is intending to move towards a Proximity Detection and Collision Avoidance System (PD/CAS). An initial review of the numbers of vehicles indicates that over 100 devices will be required to be fitted to all mobile equipment deployed on the site.

The size of the purchase triggers a requirement for a project to be initiated. (More guidance on some key elements of a project is provided by EMESRT (2020b) in their VI Control Improvement Project Guide 2020.

The key elements of the project are described in the following sections.

### Data analysis

There are two key areas of analysis required:

1. Analysis of site incident records – which confirms that failure modes identified in the generic CFw are reasonable for the site. This phase can also identify additional causes of a failure mode specific to the site and/or a completely new failure mode (which can arise if there are unique items of plant or arrangement of traffic flows).
2. System documentation – which provides guidance on the 'work-as-imagined' (see Holnagel (2014), p 120) elements of the site's Health and Safety Management System (HSMS). The author has experienced many analyses where large (by page count) HSMS documents related to vehicles and driving do not fully cover the 102 business inputs of the CFw – even though the sites are operating largely in accordance with them.

### Strategy and tactics distillation

It is important for the CFw to have a confirmed alignment with the general intent of the site's policies. These elements are checked in the HSMS and confirmed as relevant/included in the specification section of the CFw business inputs. Samples of the system level business inputs which are confirmed in this phase of the analysis are:

- clear accountabilities for key role holders (managers, supervisors, planners etc)
- fatigue management protocols (informing the set-up of rosters and manning levels)
- change Management Process, etc.

At the tactical level, the business inputs are more about specific requirements that should be in place to address failure modes. Some examples of these business inputs are:

- Site clearance requirements and processes for accurately estimating distances – information prepared for Operators – covering the sight lines from vehicles and any technologies deployed to improve vision (and responses to their failure/absence).
- All safety and operational systems on mobile equipment are maintained – which is about the execution of well-considered maintenance strategies for the deployed plant.
- Road surface changes are identified and managed by vehicle operators – highlighting tasks for water cart and road maintenance operators, as well as noting conditions which may need to be compared with a relevant Trigger Action Response Plan (TARP).

An important consideration when breaking down the business inputs in the CFw is to balance the requirements against the EMESRT design philosophy elements (see Figure 4). At the start of the project it is likely that there will be few populated business inputs for the react type controls – but the presence and resilience of the business inputs in the levels 1 to 7 greatly determine the load case for any level 8 and 9 controls when deployed.

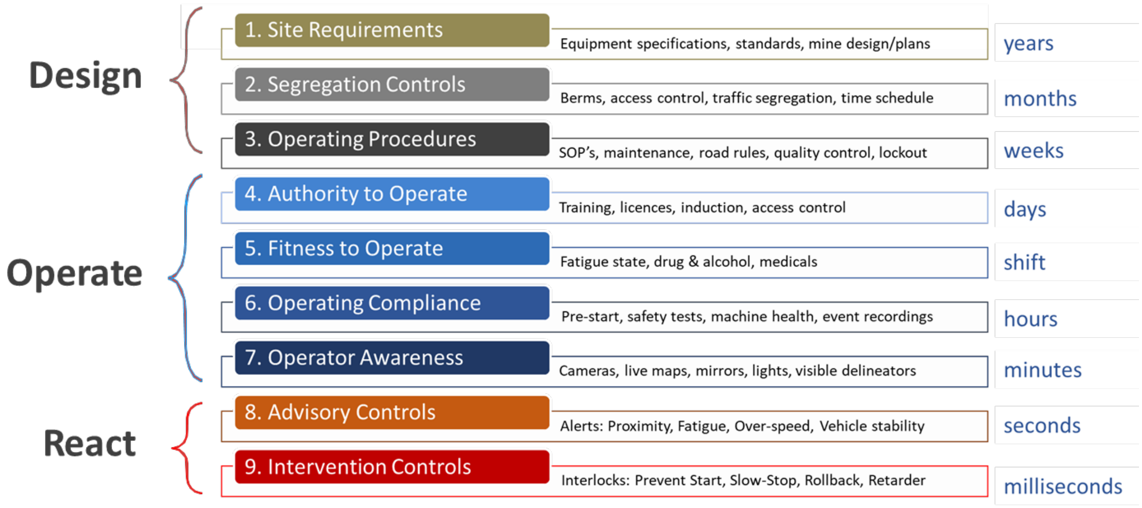


**FIG 4** – EMESRT Design Philosophy Elements – the 9 Level Model.

## Framework refinement

Continuing the build of the CFw involves engagement with key role holders through one-on-one interviews. Key role holders are dominantly in the functional/support areas. They include: planners; trainers; superintendents; engineers; stores/intro to site role holders, and; senior managers (for confirmation of resource allocation decisions).

The intent of these interviews is to confirm that the HSMS review has not missed any key items of information on which the vehicle interaction controls for the site rely.

## Obligations mapping

An important element (which can occur out of sequence in this set of subsections) is the mapping of statutory and corporate obligations to the business inputs in the CFw. There can also be some mapping to credible failure modes where clauses of statute/corporate standards require certain types of hazard to be considered.

The process of mapping can also help to update the business inputs to suit the relevant jurisdiction.

Having the obligations mapped also provides guidance on which business inputs may require monitoring signals to be most available. For example, a business input which responds to many clauses of statute should be more resilient than one which is a peripheral requirement for the site. To demonstrate the level of resilience then (ideally) an engineered confirmation of deployment (ie monitoring system) should be consolidated into information for senior management on the site – to help inform their resourcing decisions and guide an early response to trends towards/occurrences of non-conformance. Where an engineered solution is not plausible, then the best possible procedural approach (with documentation) should be required and in-place.

## Site inspections and team based validation

This phase of the project is where work-as-done is balanced against the documented work-as-imagined.

Site inspections and review of operational video footage helps to confirm how vehicle movements occur in practice. The inspections also cover engineering elements and generate questions to put to operational workers during the team-based session which follows.

The team-based workshop involves a cross-section of role holders on the site – but gives preference to involving line workers. The operators of vehicles (and the maintainers of autonomous controllers) can provide the most useful reflections on whether the framework elements as documented represent a practical and robust set of controls (business inputs) for implementation on-site.

A typical output from this type of session is a large number (typically greater than 100) of improvement opportunities for bringing the work-as-done back into alignment with the work-as-imagined documentation for the site.

Note – this is a vital step – as the work-as-imagined provides a large input to the functional and performance requirements that will be embedded in any technological solution. Technology typically 'turbo-charges' the consistency with which tasks are performed – and if this is outside what is required, then a failure of the system is almost guaranteed.

## Return to baseline

Taking the output from the previous step helps bring the site backup to a baseline level of performance.

The Control Management Sheets, used in the team based validation session are marked-up to highlight where there are identified gaps/suggested improvements.

The easiest (and sometimes quickest) 'wins' are those present in the levels 1 to 6 of the EMESRT 9 layer model. Making improvements to planning processes, worker training, supervision arrangements and road maintenance regimes are typical items identified by the teams.

An advantage of making this return to baseline is that it reduces the demand on any Level 8 or 9 technology deployed. For example if there is an improvement in intersection layouts and adoption of a vehicle hierarchy identified – then, when deployed – this can reduce the number of unwanted conflicts from one per month to one per annum.

Collision avoidance technology which would react to the conflicting travel trajectories of two vehicles would then become 12 times more effective – regardless of its level of base reliability. This is because, as a system, the vehicle movements on-site produce 1/12th the number of events which would require automatic braking. Achieving a 12-fold improvement in technology is vastly more difficult than achieving a 12-fold reduction in demand on that technology.

## Identifying where technology applies

The complete set of industry wide business inputs in the EMESRT CFw numbers over 100. Of these some are not amendable to the application of technology. Business inputs that are at levels 1 to 5 in the EMESRT framework – such as planning, developing specifications for equipment, designing road networks etc do not lend themselves to a simple application of hardware or software. They do, however, benefit from considering potential for technology in support – but this is typically out of scope for a vehicle interaction improvement project.

Other business inputs do benefit from technology being applied. Table 2 presents a sample of a typical extended analysis (post the team session) which can be used to guide equipment selection and deployment.

**TABLE 2**

Sample Output of a CFw Analysis.

| | |
|---|---|
| **Failure mode** | Controller unaware of give way requirements |
| **Business input** | Give way requirements training modules and applied by vehicle controllers |
| **Demand** | >1000 times per day |
| **Functional requirement** | As a vehicle operator, I want to be alerted when I am approaching an intersection that will conflict with a vehicle that has right of way so that I can slow or stop my vehicle to prevent interaction.<br>As a supervisor of vehicle operators I want to be informed of give way conflicts so that I can mentor the operators and consider changing vehicle routes to prevent conflicts occurring. |
| **Performance requirement comment** | Vehicles provide guidance to operators of conflicting trajectories at intersections. A smart collision avoidance system – that has geo-fenced intersections and V2V comms is guided towards.<br>Monitoring of vehicle movements at intersections and alerting of supervisors or other off-vehicle personnel. The use of forward/backward cameras on vehicles and fixed cameras at high traffic intersections PLUS a data transfer from the on-board CAS units to a reporting system are suggested from this requirement |

### *Demand frequency*

The business inputs which are relied upon to prevent loss of the required operating states will be utilised with differing frequencies. Typically those that act at level 6 and 7 of the EMESRT 9 layer model – that is Operating Compliance and Operator Awareness are the most frequently required.

To model the likely demand – it is of merit to simulate the vehicle movements – either by sketching out the full suite of vehicle interactions, or, more usefully, animating the interactions (eg using MS PowerPoint). An animated simulation of all vehicle movements is the 'gold standard' approach – but may be prohibitive for smaller projects.

From this sort of analysis it is possible to identify how often each business input that supports the key operating states of:

- operator maintains adequate clearances/distances
- vehicle operators give way appropriately to mobile plant and pedestrians
- operators drive vehicles at speeds which meet site rules and local conditions
- operators do not drive vehicles when impaired.

### *Functional requirements*

Functional requirements are developed in consideration of the 'arc' of requirements. That is, what does the consumer of the technology require to implement the business input which will address the failure mode to maintain the required operating state. To this end, the functional requirement statements take the form:

- As a *position* I want *function* so that *controller action or information flow* occurs.

### *Performance requirements*

A well-crafted functional requirement statement will help to concisely inform the requirements in the specification for a system that is put to tender. It is important that the requirements do not specify a particular technology – as this can limit the ability of an OEM to submit a conforming offer forward. If the offer meets the performance requirement under all identified scenarios – then it is worthy of consideration.

## Bridge to the digital future

The author and colleagues are working with industry to deliver these analyses to the resources industry.

An added benefit from the CFw approach is that it can identify where signal is required from business inputs. This signal, when gathered through technology, can form a powerful suite of information for decision makers.

Applying a CFw approach can help an organisation avoid the trap of drowning in a 'data lake' (https://en.wikipedia.org/wiki/Data_lake) amidst a multitude of correlations. A well-crafted framework helps provide the sense-making logic which can distil the data into meaningful information to support better decisions. These decisions maintain the operating states that make a site/business a productive and sustainable concern.

# CONCLUDING REMARKS

Control Frameworks are a useful strategy for critically evaluating a subject. The vehicle interaction challenges being faced by many open pit operators can lead to a 'brochure shopping' exercise of equipment selection. Having the functional and performance requirements clearly documented for a site can reduce the potential for deploying technologies which don't work because they annoy, rather than inform, vehicle operators/controllers.

The approach draws on work by EMESRT which has produced a validated industry model for improving heavy vehicle deployment on sites. This makes a good starting point for any site or organisation considering implementing automation or collision avoidance systems at their operation(s).

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# The application of digital twin machine learning models for Mine to Mill and Pit to Plant optimisation

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## ABSTRACT

Whilst the value of Pit to Plant and Mine to Mill optimisation has been proven since the 1990s, sustaining the productivity and efficiency gains typically requires ongoing test work, blast trials, plant surveys, and model calibration services. In contrast, the input to digital twin Pit to Plant or Mine to Mill optimisation is tens to hundreds of millions of tonnes of standard mine operating data. This paper presents a series of digital twin case studies from across the mine value chain, including machine learning testing methodology, model accuracy statistics and the application of Shapley Additive Explanations (SHAP) for model interpretation. These case studies demonstrate how machine learning and data fusion ore tracking connects hundreds of millions of tonnes of downstream performance data to produce digital twin models (operating models that are continuously updated and integrated into the wider Mine to Mill/Pit to Plant ore tracking system). Digital twin modelling software has been used to add block model variables for dig-rate, crusher and mill throughput, recovery (tailings grade), lump yield, product grade and product quality, fines yield and fines in lump. Block-by-block predictions are added to the geological resource and grade control block models as new block model variables. These new block model variables are used across all mine planning time horizons from life-of-mine planning to daily schedules. In addition to the materials handling and geometallurgical prediction case studies, the paper also presents digital twin case studies for drill and blast simulation and plant set point optimisation. The case studies show how digital twin simulation enables drill and blast engineers to simulate the effect of different blast designs in different blast domains using the mines operating data. Plant set point optimisation applies mathematical optimisation algorithms to the machine learning model to provide live operational decision support to metallurgists and plant operators. The digital twin machine learning case studies demonstrate a digital and sustainable paradigm for Mine to Mill and Pit to Plant optimisation.

## INTRODUCTION

### Mine-to-Mill and Pit to Plant

The concepts of Mine to Mill, Pit to Plant, and resource to market are terms which are interchangeable in this paper and relate to improving the overall productivity and performance of the whole mining value chain from orebody to final product by tracking the ore and slurry. Optimising each stage in isolation can result in suboptimal performance of the overall operation. It is a narrow view of the operation and often leads to operational silos where information and requirements are not exchanged for the benefit of the overall operation and improved productivity.

Mine to mill is a concept that has been around for many years and numerous publications have outlined the approach and the benefits. McKee (2013) states there was a large uptake of the term Mine to Mill from the mid-1990s, in which controlled blasting was the primary focus. From 2001–2010, orebody and metallurgical parameters were incorporated into the mine block model, providing the opportunity for improved operational short and long-term mine planning and scheduling (Bye, 2011) and there was a focus on downstream productivity improvement. Productivity gains from the

Mine to Mill methods are reported in the range of 10–20 per cent (Lam *et al,* 2001; Karageorgos *et al,* 2001; Paley and Kojovic, 2001; Kojovic, Michaux and McKenzie*,* 1995; Adel *et al,* 2006; Pease *et al,* 1998; Scott and McKee, 1994; Kojovic *et al,* 1998; Brunton *et al,* 2003).

## Conventional Mine to Mill

Conventional Mine to Mill programs are typically implemented through consulting and testing services including; collecting samples from the orebody, characterisation of these samples via laboratory tests, generation of process models using modelling software such as JKSimBlast and JKSimMet (by JKTech), distribution of predicted results into the geological block model, geostatistical interpretation, likely repeats of sample collection and laboratory tests for infill programs, assessment of mine plans with new variables populated and then roll out of the new operating strategies. It is an extensive and expensive undertaking involving many disciplines and needs to be repeated every few years for optimum and sustained benefit.

McKee (2013) states that even though productivity gains of 10–20 per cent should be sufficient for Mine to Mill to be taken seriously, widespread and sustained application of Mine to Mill methods has not been the case. The challenges are mostly around the requirement to change operating practice so that the operation runs holistically, ie it must be accepted that an increase in cost in one area is offset by more significant cost reductions in another (McKee, 2013).

## Non-additive geometallurgy

One of the biggest challenges in geometallurgy and mine planning is the issue of non-additive geological properties. Letton (2020) describes the ability of machine learning models to predict non-additive behaviour as ‘one of the most transformational aspects of all of this work for mining companies’. Geostatistics, aimed at modelling spatial to estimate properties at unsampled locations, can only be successful if the parameters being combined have a linear mixing characteristic. A parameter is deemed ‘additive’ if you can combine two samples of a known value, and the blended test results is the arithmetic average of the two.

Depending upon the scheduling time horizon, mine scheduling blocks/polygons are typically either tonnes or volume weighted to estimate the properties. Tonnes weighting assumes properties are additive, and yet it is well known that common parameters of interest such as grindability and target grind size (Work index, Axb) are non-additive properties.

Because the data used to train digital twin models is typically based on blended material, the models are able to predict the performance of blended material without having to rely on inappropriate averaging if the future blended material remains consistent or in line with the historic/trained blend.

The gold recovery, AG mill throughput prediction, crusher downtime blast models and copper flotation recovery case studies presented in this paper are all examples of digital twin machine learning being applied to non-additive downstream processes.

## Digital Mine to Mill using orebody learning software (digital twin modelling)

Orebody learning enables mining operations to systematically learn about their orebody through experience. Orebody learning combines data fusion ore tracking and machine learning to increase orebody knowledge (Figure 1). Data fusion ore tracking converts spatial geological data into a time series geological data set (fused geological data) suitable for machine learning modelling of processes such as load and haul, crushing, milling, and flotation (Figure 2). The case studies presented in this paper demonstrate how orebody knowledge is translated into operational decisions across the value chain, including:

- adding new ‘operational experience’ variables to the block model
- simulating effect of blast designs on downstream processes for each unique geology
- recommending optimal set points for different ore types on an hourly basis.

In contrast to 3D geological modelling from limited drill holes and laboratory tests, orebody knowledge acquired through data fusion ore tracking and machine learning fuses tens to hundreds of millions of tonnes of ore data to characterise the orebody as it is mined. Whilst conventional Mine to Mill programs are based upon generalised empirical models such as those built into JKSimblast™ and JKSimfloat™ (by JKTech), MAXTA™ version 2.2 (by Petra Data Science) orebody learning software (referred to as *the software* in this paper) continuously learns from actual operational performance. Typically, tens to hundreds of millions of tonnes of geological data is fused with operational performance data to predict, simulate and optimise across mine value chain. In effect, orebody learning models enable mining operations to systematically learn about their orebody through experience.

The orebody learning models produced by the software are sometimes referred to as digital twin models because the virtual representation is linked(twinned) and continuously updated from the physical operational data. Though the concept originated earlier (attributed to Michael Grieves, then of the University of Michigan, in 2002) the first practical definition of digital twin originated from NASA in an attempt to improve physical model simulation of spacecraft in 2010 (Negri, 2017).

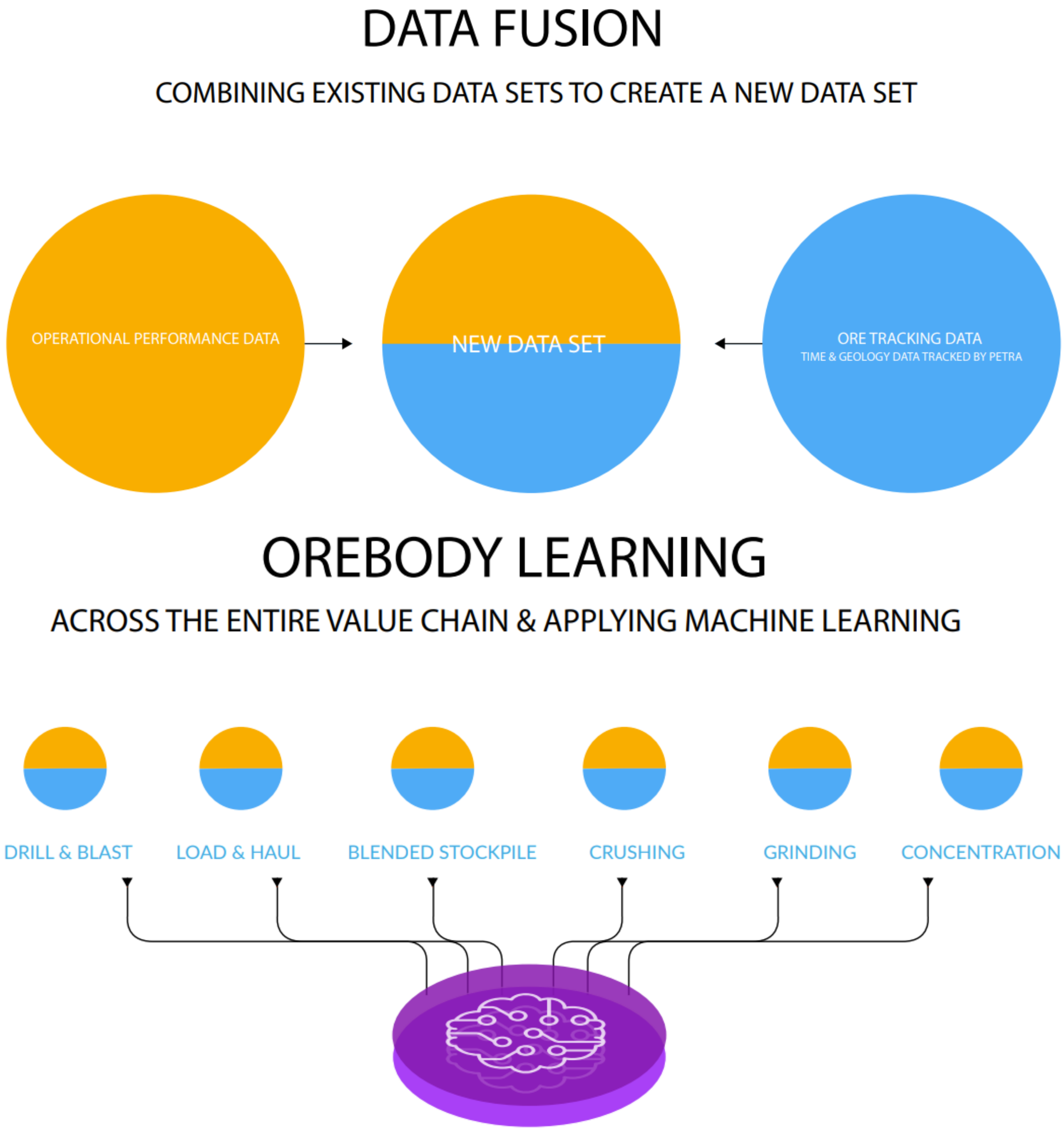



**FIG 1** – Data fusion and orebody learning.

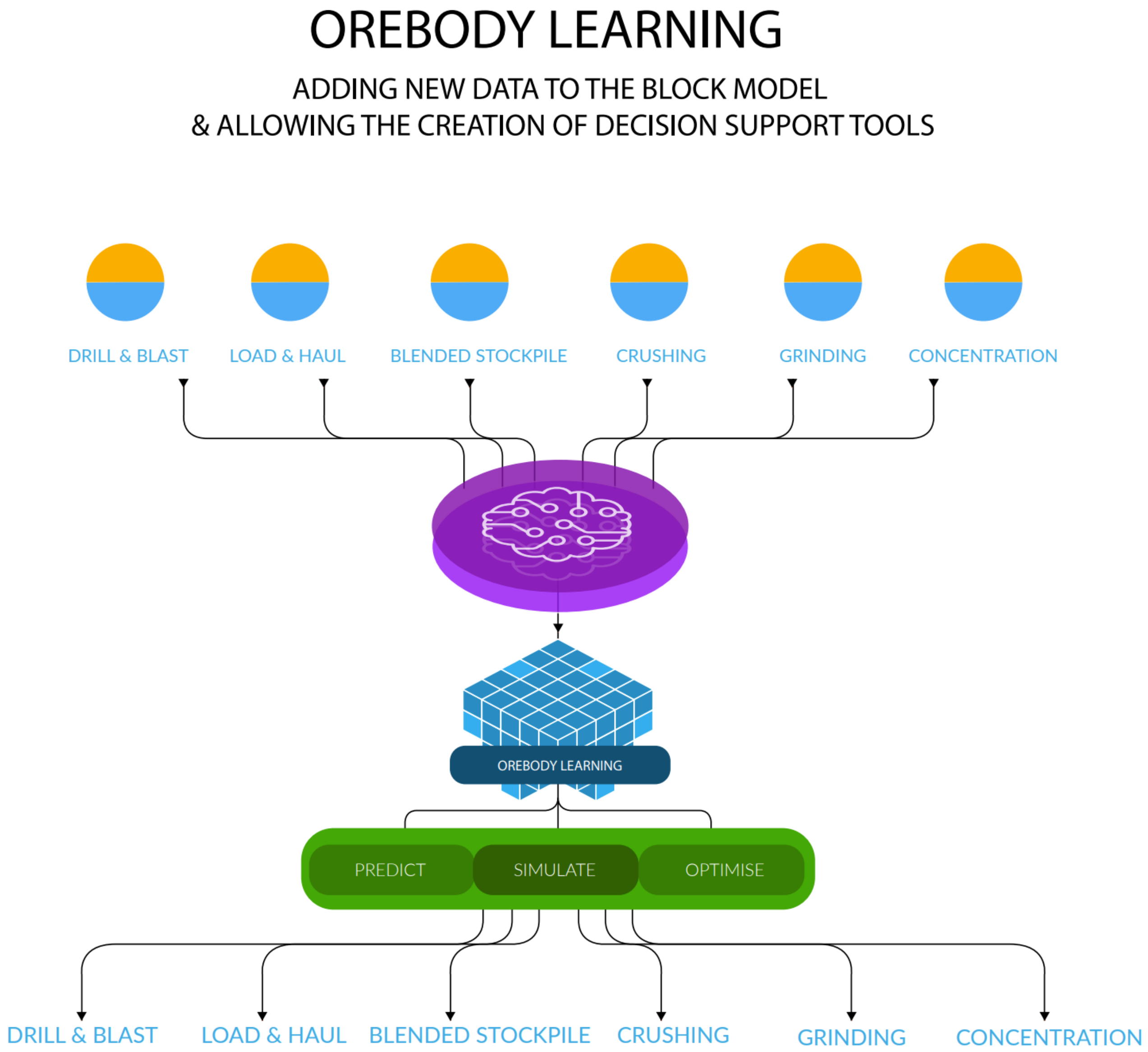



**FIG 2** – Orebody learning variables added to the block model for accurate mine planning and orebody learning (machine learning and optimisation) models for simulation and optimisation.

## Historical development of orebody learning

Stewart and Trueman (2001), Stewart (2005) and Maher, Stewart and Robotham (2016) site specific stope dilution models demonstrated the potential for site specific models to be more accurate than generalised empirical models and led to the 'big data' approach to Mine to Mill (Stewart, 2016) and later digital twin Mine to Mill orebody learning to predict, simulate and optimise processes across the mine value chain.

## Data fusion ore tracking

Multisensor data fusion is defined by Durrant-Whyte and Henderson (2016) as 'the process of combining observations from a number of different sensors to provide a robust and complete description of an environment or process of interest'. Multisensor data fusion methods are drawn from a diverse set of more traditional disciplines including: digital signal processing, statistical estimation, control theory, artificial intelligence, and classic numerical methods (Khaleghi *et al,* 2013).

Data fusion ore tracking involves the integration of all the data generated at the mining operation, from the geological model, blasting and fragmentation, mining, hauling, comminution and separation processes. It can be viewed as the digital version of traditional Mine to Mill systems. Geological and blasting data attributes are tracked and merged with operational performance (eg dig rates, throughput, recovery) data as it moves through the individual stages of the mining value chain. The new fused data set enables machine learning models to continuously learn about the orebody. The software's inbuilt data fusion ore tracking uses a range of multisensor data fusion methods depending upon the type of sensors, frequency of data, and frequency, accuracy and speed of processing of results.

## Digital twin models for orebody learning

Orebody learning software uses machine learning to 'twin' the fused data sets shown Figure 1 to predict, simulate and optimise orebody extraction and processing across the mine value chain (Figure 2). In this paper these digital twin machine learning models are referred to 'digital twin ML' models. This application of digital twin models for simulation is documented by Moore (2018). Machine learning is a branch of AI (artificial intelligence) that consists of deep learning, supervised and unsupervised learning. It is the convergence of computer science and mathematical branches of statistics and calculus and can be described at the automatic application of mathematical techniques on a massive scale using volumes of data enabled by extremely fast computing speeds.

There is often a large volume of data recorded about the mining operation and often after compilation there are thousands to hundreds of thousands of usable observations for training Machine Learning models.

These observations are segregated into a train set and a test set. The train set is used to generate the models and the test set is used to generate performance metrics. Because the test set is unseen (not used) in the modelling stage, it gives a more realistic representation of how the model will perform once deployed. This is different to some statistical approaches where modelling is done with the full data set and performance metrics are based on data that the model was built with.

During the modelling phase, the train set is often split again to provide guidance for model parameter tuning and analysis of time-based effects.

# CASE STUDY 1 – BAN HOUAYXAI GOLD MINE: PREDICTION OF GOLD TAIL GRADE

Ban Houayxai is a gold operation located in northern Laos.

During its operation, the CIL (carbon-in-leach) gold plant experienced extended periods of very low gold recovery, less than 50 per cent gold recovery. The losses (high gold content in tails) were thought to be a result of gold locked in silicates and sulfides and to limited to fresh, unweathered rocks (Carpenter *et al,* 2018). Internal petrographic and geochemical studies failed to clearly identify problematic zones in the orebody and the reason for high gold losses.

Orebody learning and data fused ore tracking was implemented using two years of operating data. This is equivalent to the tracking of ten million tonnes of ore from the pit to waste dumps and to the plant. The data was merged and collated to produce a digital representation of the operation from Pit to Plant. This digital representation included periods of low gold recovery pertaining to the processing of fresh unweathered ore.

The application of machine learning techniques to CIL predict gold tail grade (Au g/t) using only variables in the geological block model enabled gold tail grade to be spatially linked to information in the geological block model, referred to as orebody learning, see Figure 3. Block-by-block predictions of gold tail grade were added to the geological block models as new block model variables.

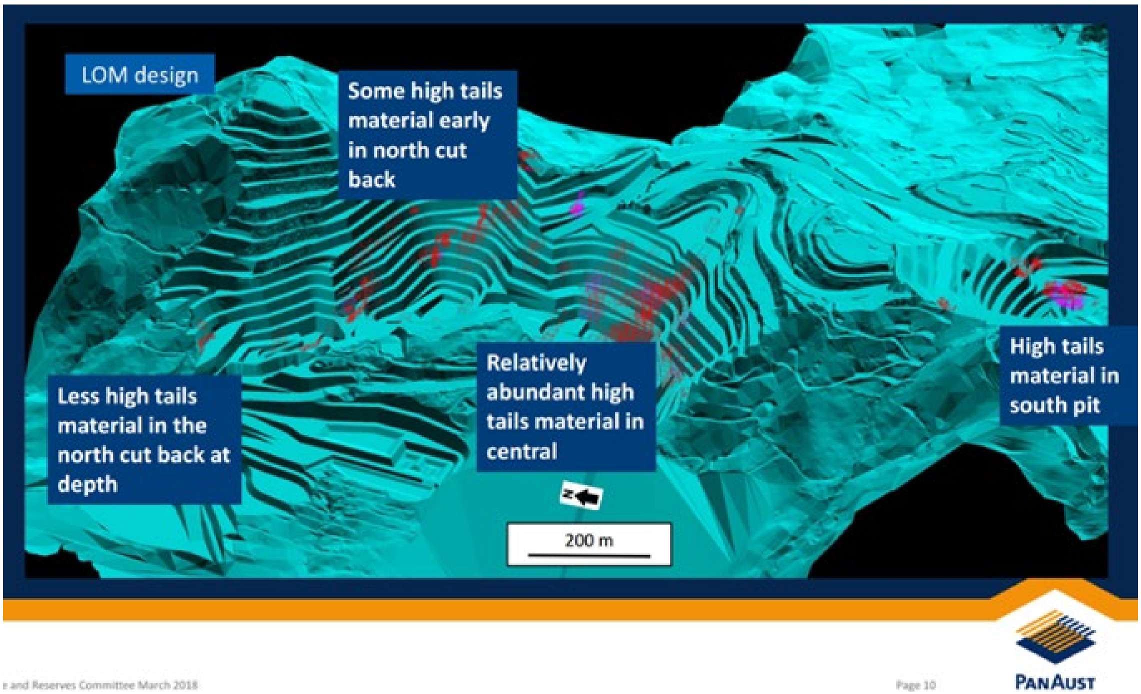



**FIG 3** – Identification of problematic (high tail grade) blocks in the LOM block model using orebody learning (a combination of data fused ore tracking and machine learning) (after Carpenter *et al,* 2018).

This identified potentially problematic blocks with high tails in the life-of-mine (LOM) and allowed planning and scheduling engineers to proactively avoid previous high tail grade excursions, thereby increasing the LOM Net Present Value (NPV) of the operation.

## CASE STUDY 2 – CITIC PACIFIC MINING (CPM): PREDICTION OF AUTOGENOUS GRINDING (AG) MILL THROUGHPUT

CPM operates in Western Australia and is a large magnetite processing facility consisting of multiple process trains, at the head of which are AG mills. The throughput of these mills dictates the capacity and production of the operation.

It is well known that multiple physical, chemical, geological and geometallurgical properties of the ore (not all of which are known) affect and determine the throughput of ore in AG mills. The aim was to develop a calculation to predict AG mill throughput using only variables from within the resource model. Mining, loading, primary crusher as well as coarse ore stockpile (COS) information was used to create the digital twin ML model. In this instance the COS proved the most challenging part of the system to model and track. This was overcome by the generation of virtual stockpiles and the use of iterative modelling to determine characteristics of the material movement through the stockpiles. This amounted to tracking 17.2 Mt of ore from the pit, through the primary crushers and the AG mills. The software was used to generate the digital twin ML AG mill throughput prediction model. In total 24 variables were used for the final calculation used to predict AG mill throughput.

This was deemed to be important from a processing perspective because process engineers could proactively plan for variations in material types rather than retrospectively react to ore material fed of unknown variability. It was also deemed valuable from a mine planning point of view as more accurate schedules could be created and planned (and compared against known variables such as stratigraphy) accordingly.

In house validation of the AG mill throughput prediction indicates less than five per cent deviation on a month-on-month basis to actual throughput rates. The same principle can be applied to higher resolution data such as a grade control model for more accurate day-to-day operation guidance.

The overall performance of the prediction model using test data (data not used in the model building) is shown Figure 4. Model development within the software is discussed in later sections of this paper.

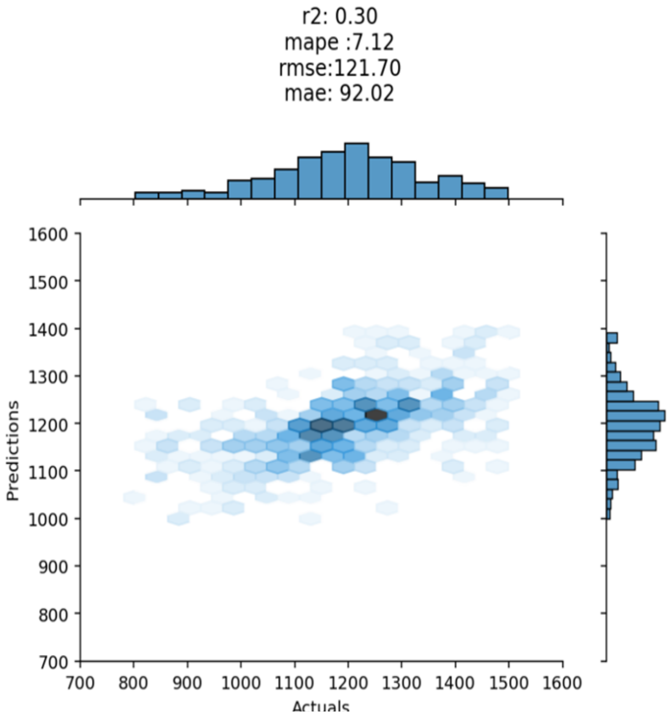



**FIG 4** – Digital twin ML model performance of AG mill throughput on unseen test data (data not used in training the model).

Machine learning predictions for AG mill throughput were populated into the geological block model on a block-by-block basis. A visualisation of the AG mill throughput prediction for each block using Vulcan software is shown in Figure 5. Please note throughput tonnes per hour have been adjusted but are proportionally correct.

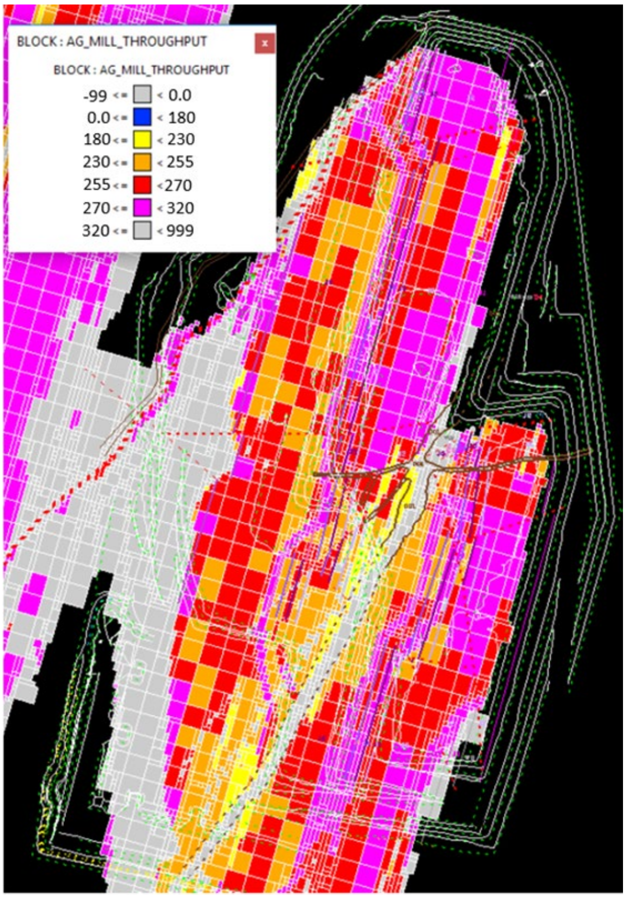



**FIG 5** – Machine learning model AG mill throughput by block visualisation.

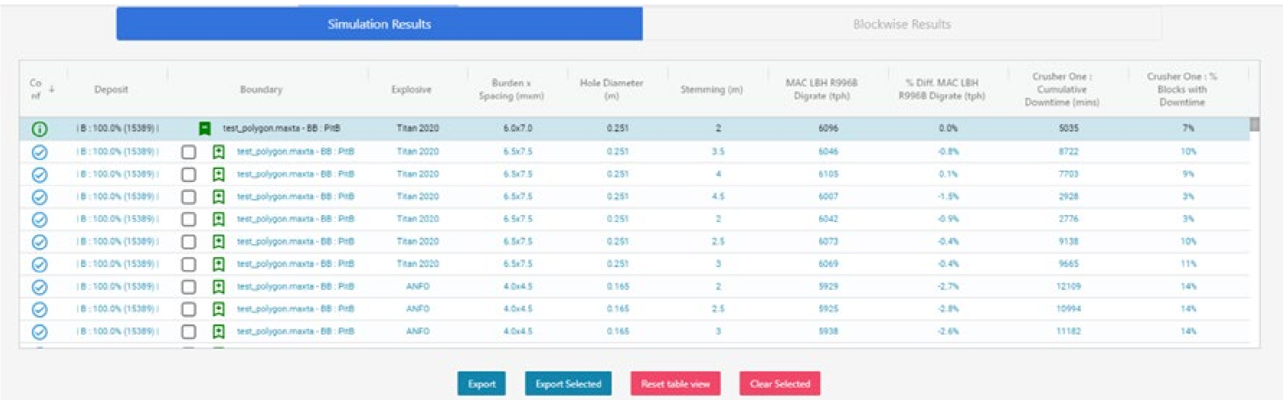
## CASE STUDY 3 – LARGE IRON ORE AUSTRALIAN OPERATION: PRIMARY CRUSHER DOWNTIME

Oversize material is a significant problem in primary crusher operation leading to unplanned downtime caused by oversize rocks blocking and bridging across the crusher feed opening. Unscheduled downtime is compounded by the safety issues around manual breakage or dislodging of jammed rocks.

At this operation oversize downtime accounts for up to 35 per cent of total unplanned downtime events and a typical oversize event equates to about 13 minutes lost operating time.

Using the software the operation can now identify with considerable certainty which zones within the blasted boundary will likely contain oversize material. Allowing the operation to proactively manage the treatment of this material by re-directing it to an assigned stockpile for selective breakage of large oversize using a rock-breaker. This proactive approach of stockpiling likely oversize material will increase annual plant capacity and significantly increase revenue.

As shown in Figure 6 the software was able to identify problematic oversize zones in the orebody, and the application of the software was extended at this operation to facilitate the investigation of drill and blast design parameters on primary crusher downtime due to oversize, meaning a simulation tool was developed where drill and blast design parameters such as burden and spacing, hole diameter, explosive type and stemming could be varied and the impact on crusher downtime predicted. Downtime is shown as a percentage of total within blast boundary likely to cause crusher downtime according to the blast design.




| Conf | Deposit | Boundary | Explosive | Burden x Spacing (mxm) | Hole Diameter (m) | Stemming (m) | MAC LBH R9968 Digrate (tph) | % Diff. MAC LBH R9968 Digrate (tph) | Crusher One : Cumulative Downtime (mins) | Crusher One : % Blocks with Downtime |
|---|---|---|---|---|---|---|---|---|---|---|
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | Titan 2020 | 6.0x7.0 | 0.251 | 2 | 6096 | 0.0% | 5035 | 7% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | Titan 2020 | 6.5x7.5 | 0.251 | 3.5 | 6046 | -0.8% | 8722 | 10% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | Titan 2020 | 6.5x7.5 | 0.251 | 4 | 6105 | 0.1% | 7703 | 9% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | Titan 2020 | 6.5x7.5 | 0.251 | 4.5 | 6007 | -1.5% | 2928 | 3% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | Titan 2020 | 6.5x7.5 | 0.251 | 2 | 6042 | -0.9% | 2776 | 3% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | Titan 2020 | 6.5x7.5 | 0.251 | 2.5 | 6073 | -0.4% | 9138 | 10% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | Titan 2020 | 6.5x7.5 | 0.251 | 3 | 6069 | -0.4% | 9665 | 11% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | ANFO | 4.0x4.5 | 0.165 | 2 | 5929 | -2.7% | 12109 | 14% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | ANFO | 4.0x4.5 | 0.165 | 2.5 | 5925 | -2.8% | 10994 | 14% |
| | \| B : 100.0% (15389) \| | test_polygon.maxta - BB : PitB | ANFO | 4.0x4.5 | 0.165 | 3 | 5938 | -2.6% | 11182 | 14% |

**FIG 6** – Example of drill and blast user interface showing blocks in the geological block model identified to contain oversize according to blast design.

As in the previous case studies the software uses operating data from each stage in the mining value chain, which is fused with geological data to produce a digital twin ML model of the operation. Millions of tonnes of ore was tracked, including geological variables from the block model, drill and blast design parameters are attached to drill hole locations, load and haul to the primary crusher. Downtime data and records were then merged with this digital twin ML to identify periods of primary crusher downtime due to oversize. It is from this data set that downtime prediction models including those used to show impact of blast design on primary crusher downtime were generated.

Figure 7 shows the performance metrics, details discussed in later sections, of the machine learning model. It also shows the model inputs and how these variables interact, also discussed in later sections of the paper.

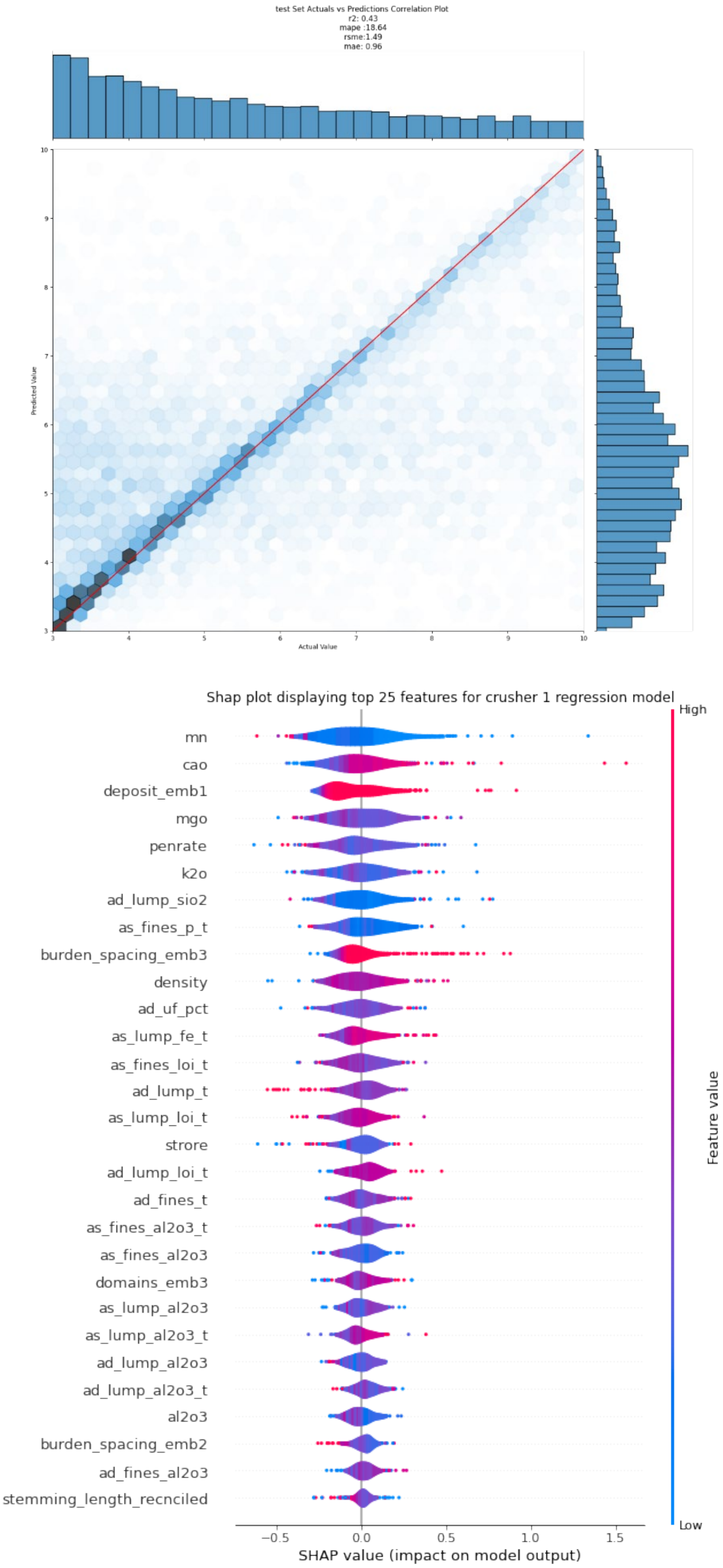



**FIG 7** – Machine learning model primary crusher downtime (i) performance on unseen test data and (ii) model inputs and interactions as shown in a SHAP chart (details discussed later).

## CASE STUDY 4 – LARGE COPPER GOLD OPERATION: PROCESS OPTIMISATION BY SET POINT RECOMMENDATION

The orebody learning software was implemented in the flotation circuit of a large open pit copper-gold porphyry operation to predict copper recovery and recommend optimal reagent addition rates and mass pull rates to maintain low copper tail grade and thereby maximise recovery.

The software ran live every hour via the DCS (distributed control system) with inputs from in-house built database containing fleet management system tracked geological properties and PI System (data historian by AVEVA).

The first part of the software implementation was to implement a digital twin ML prediction model to predict flotation copper tail grade six hours in advance, provided the plant was running in steady state and excluding any process excursions such as manual reduction in feed rate because of external/outside issues.

The flotation copper tail grade was predicted six hours in advance even though the flotation circuit residence time is less than 30 minutes. The digital twin ML model was heavily dependent on the geological properties of the ore upstream of the flotation circuit and operational set points were dynamically updated every hour depending upon these upstream geological ore properties.

In order to recommend flotation set points for best flotation performance, mathematical optimisation was overlayed on the prediction model to recommend the optimal process plant set points to minimise copper tail grade when treating similar ore properties in the past. In this way the optimal set points dynamically updated as the upstream geological ore properties vary. Automated data quality management algorithms are built into the software and are discussed later in this paper.

Whilst the software internally optimises the set points every hour, and the optimised set points are only published as operational decision support tools every six hours to limit frequent and large changes in process set points which commonly lead to circuit instability. The process control engineers have the option to vary the publishing frequency rate in the future if required.

Orebody learning software continues to learn from new operational experience as captured in the operational performance data (Figures 1 and 2). In practice, orebody learning occurs when the site process control engineers in consultation with the metallurgists decide to run an update on the digital twin ML models inside the software. Data scientists refer to this type of model update as model retraining. The software's automated model retraining updates and rebuilds the models using existing model inputs. The new model is automatically evaluated for accuracy before being released. As part of the automated retraining the software automatically rebuilds the digital twin ML models using the latest data available. In effect this update is learning the best set points for any changes in geology or plant operating conditions. The engineers at this operation decided to update the digital twin ML models approximately every six months.

Figure 8 shows the PI System trends with decision support optimised set point recommendations.

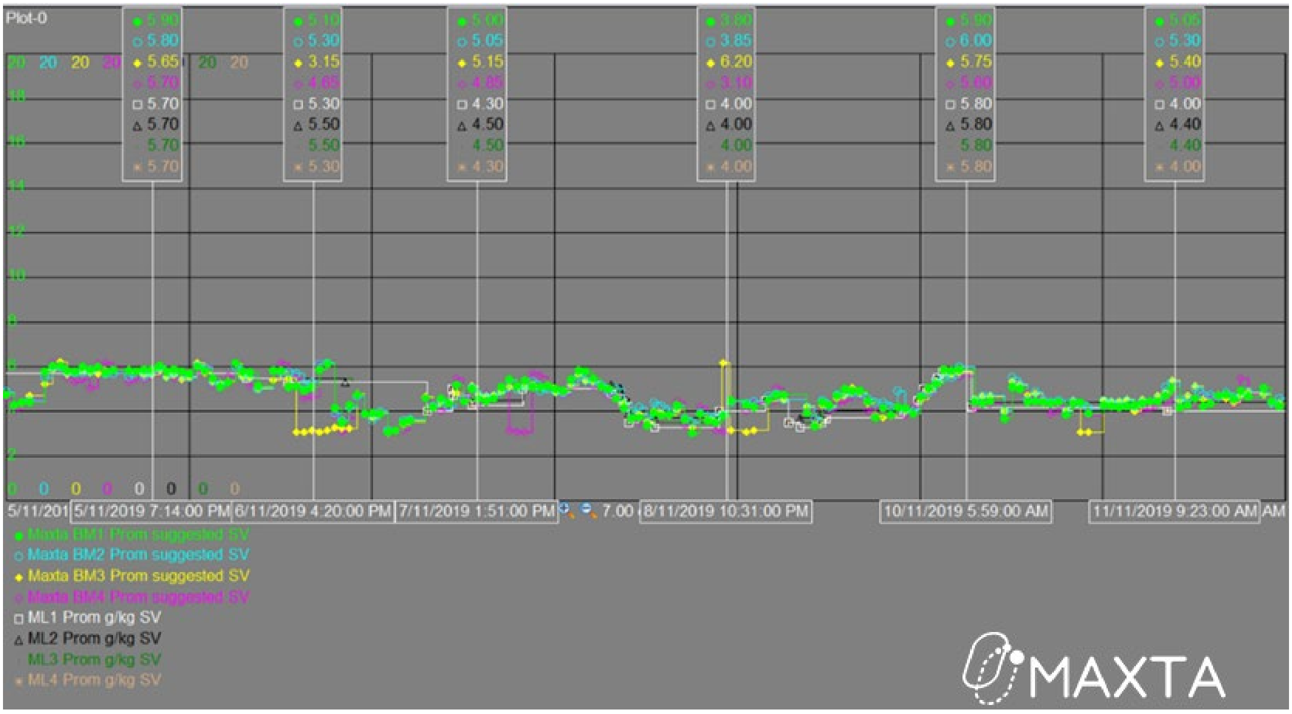



**FIG 8** – Digital twin operational decision support predictions and dynamic set point recommendations provided at six hour intervals – integrated into PI System (by AVEVA) and control room via the DCS (distributed control system).

# DISCUSSION

## Data quality management

Many mining operations have issues with poor and inconsistent data quality. Poor data quality can lead to low accuracy models. Purpose built automated data fusion and data quality management algorithms contained within the software automatically manage data quality:

- Calibration and autocorrection of key online sensors such as those coming from on stream analyses (OSAs) and particle size indicators (PSIs) with truth data, typically from control laboratory samples.
- Monitoring online sensor for anomalies to trigger real-time alerts and sensor health for model inputs.
- Automated real-time model switching contingent upon sensor health for model inputs.
- Model confidence including real-time model output compared to actuals via the DCS for continuous visibility of model accuracy.
- Data valorisation using multi-sensor cross validation.

Most importantly, it is pointless changing input data or fixing it for the purpose of building a model if, when the model is deployed to operations, it is exposed to poor quality data. Data cleaning and correction functions, such as those outlined here, need to be built into the software and must be applied to raw data feeds in real-time.

Raw mining data can be of poor quality but there are proven methods currently operational where these data issues are corrected and addressed in near-real-time to ensure AI and machine learning integrity.

## Model accuracy

As mentioned, machine learning model performance metrics pertain to model accuracy when tested on unseen data, meaning data not used in model building. Additionally, they errors are reported at the time granularity that the fused data sets (Figure 1). Typically, the fused data sets are aggregated on one-hour intervals.

The following metrics are used to quantify the accuracy of the digital twin ML models:

- Mean absolute error (MAE) is the average of the absolute differences between prediction and actual observation where all individual differences have equal weight.
- Mean absolute percentage error (MAPE) is the percentage average of the absolute differences between prediction and actual observation where all individual differences have equal weight.
- R-squared ($R^2$) is the square of correlation and it shows the proportion of variation in the dependent variable that can be attributed to the independent variable, for instance if $R^2 = 0.5$, then 50 per cent of the observed variation can be explained by the model.
- Root mean square error (RMSE) is the square root of the average of squared differences between prediction and actual observation. Since the errors are squared before they are averaged, the RMSE gives a relatively high weight to large errors.
- Symmetric mean absolute percentage error (SMAPE) divides the absolute errors by the mean of the absolute actual and the absolute predicted values. This is useful when the actual values are near zero which is the case when predicting tail grade concentration. Actual values near zero cause the MAPE value to become infinitely high.

Figure 9 shows the digital twin ML model predicted instantaneous dig rate (test data not used in training the model) compared to actual instantaneous dig rates.

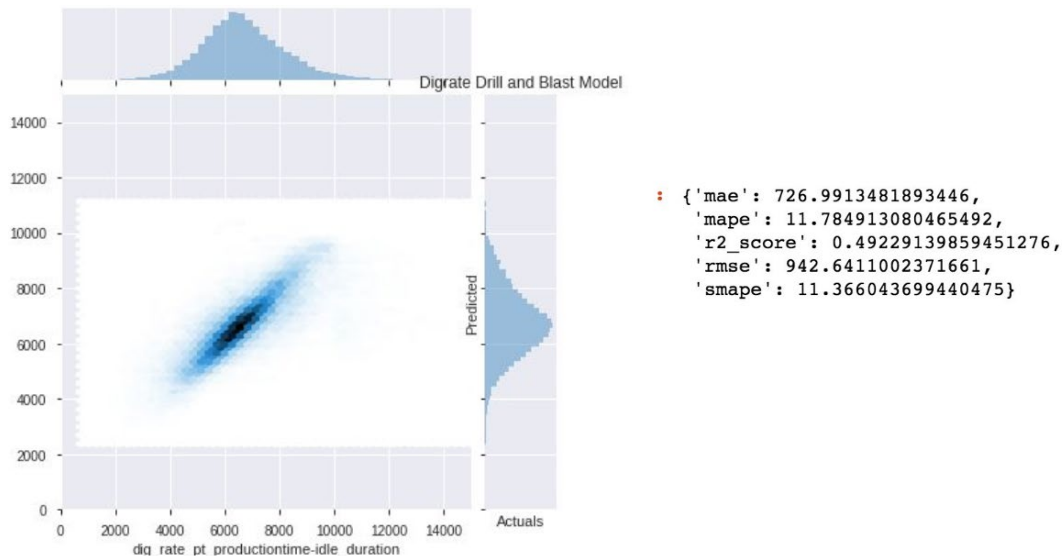



**FIG 9** – Example of predicted versus actuals using digital twin ML models to predict instantaneous dig rates, weighted according to actual observations, model accuracy metrics shown.

The accuracy of digital twin ML models typically range from 2 to 12 per cent MAPE (mean absolute percentage error) on test data not used in the development of the model. Typically, month on month accuracy is less than 2 per cent MAPE, whilst hourly prediction error for models used in optimisation is typically less than 12 per cent. Note that time series prediction models whilst more accurate for process variable prediction in real-time cannot be used for set point optimisation because they are not a model of the process. This accuracy is significantly higher than typically achieved using more conventional regression methods. Direct comparisons between statistical non-linear regression models and machine learning models typically show a 30–40 per cent reduction in MAPE on test data. There are several reasons for the difference in accuracy.

An important difference between machine learning and statistical models is the ability of machine learning models to leverage interactions between variables and the impact this has on the model output. For example, machine learning model input variables sometimes have a positive relationship and sometimes have a negative relationship to the prediction depending upon the value of other block model variables. Typically, machine learning models will include millions of complex interactions between block model variables (more on this in the next section).

Additionally, machine learning models are not biased by ratios of data groupings within a data set. For example, if a training data set only contains 8 per cent of a particular geological domain and the same geological domain makes up 35 per cent of the global orebody, the machine learning model performance will be unaffected by the unequal representation of the particular geological domain in training and reality. In contrast a statistical regression model would be biased towards the properties of the training data and misrepresent the properties of the actual ore if the geological domain ratios change. This negatively affects the longevity of statistical models.

## Interpreting machine learning models

Contrary to common perception around machine learning, the inputs and internal interactions of models are visible and can be investigated and interpreted. This is particularly true for supervised machine learning where predictions are derived using actual observations, that is, the target variable exists.

SHAP (Shapley Additive explanations) values are diagrams used to explain individual predictions. SHAP is based on the game theory where each variable is a 'player' in a game where the prediction is the 'payout'. Shapley values tells us how to fairly distribute the payout among the variables or players.

SHAP are available as free open-source libraries that produce interpretable plots and charts, below are a couple used extensively to interpret MAXTA models.

### *(i) Variable importance plot*

This chart lists the most significant variables in descending order. The top variables contribute more to the model than the bottom ones and thus have high predictive power (Towards Data Science, 2019).

### *(ii) SHAP value plot*

Figure 10 shows the positive and negative relationships of input variables with the target variable. It is made of all the datum in the train data. The variables are ranked in descending order. The horizontal location shows whether the effect of that value is associated with a higher or lower prediction. Colour shows whether that variable is high (in red) or low (in blue) for that observation (Towards Data Science, 2019).

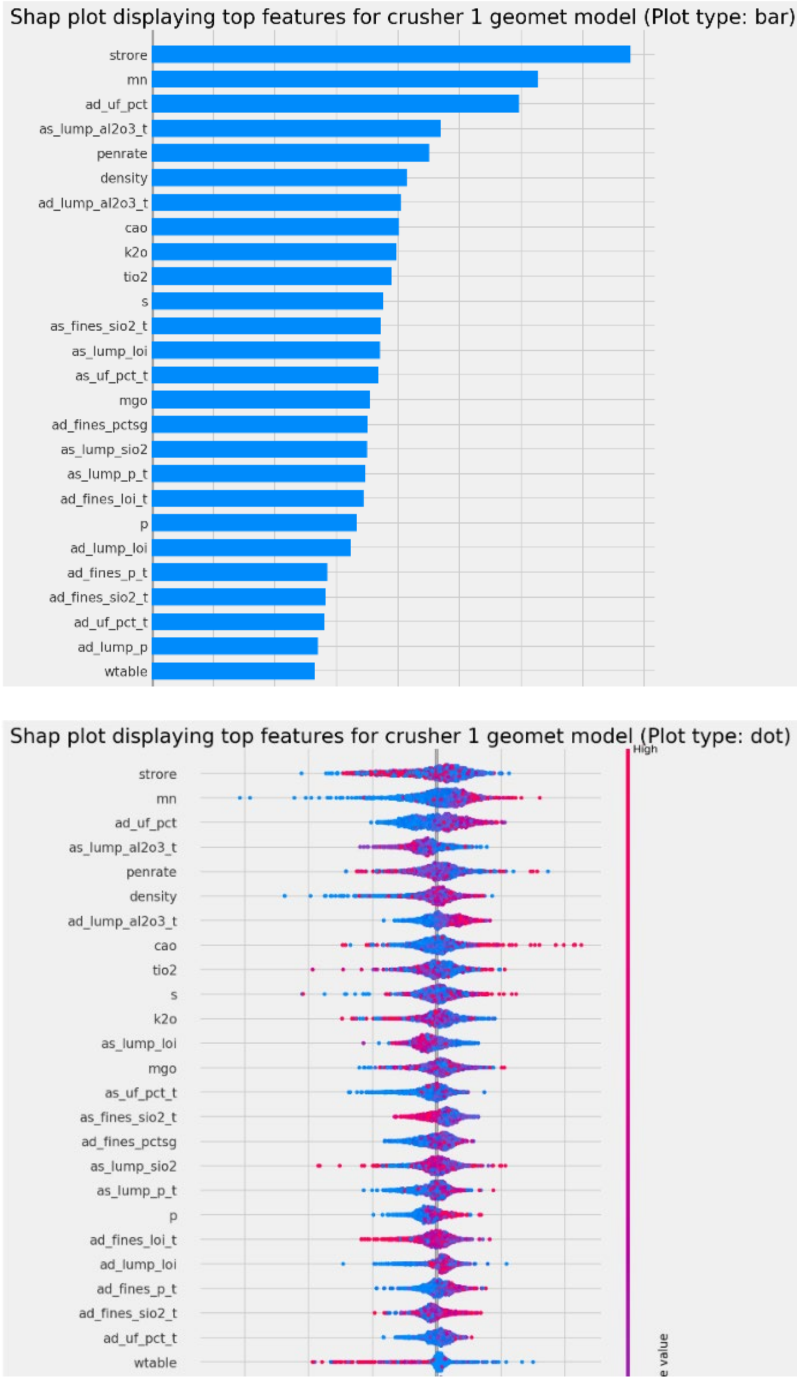



**FIG 10** – Example MAXTA machine learning interpretability using: (i) variable importance chart, and (ii) SHAP value plot.

The most significant feature of these charts, especially the SHAP value plot, is that it highlights the complex interactions and inter-relations between variables that exists in the data. This means that at certain times a variable can pull the prediction in the negative direction and at other times it pulls the prediction in the positive direction depending on surrounding values of other variables in the model. Unlike conventional regression models where impact of a variable is singularly defined by the coefficient magnitude and direction.

## Sustaining the benefits of Mine to Mill

One of the reasons a digital solution for Mine to Mill is sustainable is that once deployed the machine learning models can be updated with new data at regular intervals. In the field of data science modelling the updating of models with new data sets is called retraining the model, and this process is typically automated with a model verification steps to ensure the new model is an improvement over the old model. Whilst live processing plant digital twin ML models are retrained continuously via secure API (application programming interface) at this time most operations do not have a data architecture capable of live digital twinning. In the mine, it is more common for data to be offline and so model retraining is done offline. That said, an increasing number of mining operations are building data architectures suitable for connecting (via API) digital twin applications such as the MAXTA orebody learning software.

This facilitates more accurate scheduling from life-of-mine planning scale blocks/polygons to short-interval-control scale blended stockpiles and direct tip combinations of blocks.

The ability to link the block model to downstream performance increases the value of the block model because it enables the downstream processes to be optimised for geological variability (The Australian Mining Review, 2020). Munro (2016) states

> *I dream of people at the morning meeting in every concentrator reviewing the 3D representations of the orebody, hearing a metallurgist say 'our current feed is from bench/stope AB where the predominant rock is andesite, grindability data for this rock type show a Bond work index, Axb and drop weight index, so the predicted throughput is...*

This is now possible with machine learning model described here, furthermore these predictions are based on actual plant performance and when combined with test work for ore characterisation and simulation outputs, can provide even stronger links to geology thereby advancing the Mine to Mill initiative.

The application of orebody learning software for the generation of a digital twin model where select variables can be adjusted to enable 'what if' simulations for various scenarios can generate information for real-time and near real time operational decision support.

As shown, the software can be used in optimisation of drill and blast inputs (burden, spacing and design powder factor) to better suit specific rock types where variance in uniaxial strength, Young's modulus and gold deportment are common. Like many Mine to Mill investigations, the drill and blast inputs can be optimised for improve mill throughput. Whereas identification of optimum grinding circuit settings (mill speed, feed density, mill load) and flotation circuit parameters (reagent dosages, mass pull) can be identified and recommended as operational guidelines/decision to support and ensure maximum recovery. Further, new operating data can be automatically ingested for model retraining to continually update model outputs.

Orebody learning software requires operational data. For greenfield deposits, orebody learning is not applicable. Rather, more common and established approaches to Mine to Mill and particularly geometallurgy are applicable.

The software is designed to automatically update and retrain prediction and optimisation models as new data becomes available. It can be embedded in current workflows and the results and recommendations can be tied into existing interfaces and software to minimise training requirements.

## CONCLUSION

The successful and profitable operation of a mine depends not only on the performance of the individual processes, but on the performance of the operation as a whole. Mine to mill methods over the last 30 years have proven increases in productivity can be as high 20 per cent when a holistic approach to optimisation is applied. However, challenges remain around sustaining the application of Mine to Mill methods and most of these issues are around change management.

Orebody learning is a digital solution to value chain optimisation, a complement and alternative to conventional Mine to Mill systems. Orebody learning software avoids the extensive and repeated sampling, test work and statistical and empirical modelling around traditional Mine to Mill systems. Orebody learning uses 3D geological data with operational performance data from all stages of the mining value chain including geology to mineral processing. It enables:

- Accurate geometallurgical, materials handling and recovery inputs for mine planning and optimisation results.

- Continuously learning as new data becomes available simulation and optimisation tools for engineers across the mine value chain.
- The use of digital twin to conduct 'what if' simulations of various scenarios, eg drill and blast design simulation.
- Recommended set points that are optimised for the specific ore being fed to the plant.

Orebody learning software is seen as a more sustainable and likely adaptable approach as it can be used to enhance current workflows, interfaces and software already being used on-site.

Orebody learning software can be implemented in such a way that the digital twin models create a feedback loop that is essentially live to operations enabling fast and efficient production based decisions to be made that optimise different stages of the operations. The software facilitates strengthening of the ties between the connected disciplines by sharing and merging data, no matter the source of the data. This can lead to better informed operational decisions and improved productivity to that achieved by traditional Mine to Mill systems.

## ACKNOWLEDGEMENTS

The PETRA team would like to acknowledge and show appreciation for the support, time and commitment of industry champions who promote and support the uptake of practical digital solutions at their operations, special mention and thanks to our co-authors James Carpenter, XiaoMing Hu and Rob Embry.

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# Social license to operate

# Social license to operate – circular economy

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## ABSTRACT

### Social licence to operate

Novum Energy Australia Pty Ltd (Novum) has set itself the target of providing a reclamation technology with ultra-low carbon emissions across operational TVR plants in Georgia, USA (owned by Green Carbon Inc.), and Fort McMurray, Alberta, Canada (owned by TTRC located at Suncor Energy Inc.). Novum already has a proven operational record of 10 years for rubber waste reclamation in USA and Canada and has been planning to expand its ability to Australia (ie Qld, NSW, and WA). While Novum is continuing with fuel oil, and recycled carbon black (rCB) reclamation as high-quality base products for new applications, research in new applications and improved quality is progressing well, increasing the potential in sourcing of renewable energy and materials as replacements in a circular mining economy.

Novum is to fulfill an important part in setting 'zero waste to landfill' targets with major mining companies in Australia. An example of one such initiative is Novum partnering with Anglo American to construct a waste rubber reclamation plant in central Queensland. This plant will process mining-related waste rubber to produce by-products for reuse by the same industry. This will mitigate issues that sites have with waste tyre and conveyor storage and contribute to the sustainability goal of reducing waste to landfill.

The TVR process, and accounting for recycling credits from steel, energy generation, and reclaimed Carbon Black, a net reduction of $CO_2$ equivalent emissions is realised, where 982 kg $CO_2$ equivalent emissions are reduced by processing 1 t of waste tyres. The reuse of products from end-of-life (EOL) tyres has the potential to make a substantial contribution in reducing carbon emissions and fossil fuel consumption. As example, if 3 Mt of rubber tyres were processed in a year, this would result in a reduction of about 2.95 Mt of $CO_2$ eq emissions or an energy savings of 92 342 MJ/kg.

By participating in the regional areas, close to waste rubber supplies, Novum also recognises that it is part of an integral part of the cumulative regional performance management system and that it will need to align with international best practice in terms of training, job creation, and providing opportunities for new support businesses, and thus in essence supporting features for the mining industry to achieve a 'social licence to operate'.

## INTRODUCTION

Heavy industry, specifically mining, generates a significant amount of waste rubber products that are hard to dispose of, let alone recycle. Solving this growing problem will make the mining industrial sector more sustainable and promote related circular economies. Previous proposed recycling solutions are costly, cumbersome, high in GHG (Greenhouse gases) emissions, and energy intensive, while activity is completed away from the region.

Novum Energy Australia Pty Ltd (Novum) operates a proven, patented process that is an integral part of a waste rubber circular economy. Through its Thermo-Vacuum Recovery (TVR) technology that Off-the-Road (OTR) tyres and industrial all types of conveyor belts (Figure 1) are converted into clean, carbon credit-worthy, saleable, low emission products.

**FIG 1** – Waste domestic and truck tyres, conveyor belting, tracks, and rubber hoses (Rome, Georgia, USA).

## TVR PROCESS DESCRIPTION

### Introduction

In the last decade, multiple strategies including pyrolysis, gasification, de-vulcanisation, hydro-generation, chemical degradation, and catalytic cracking have been adopted for the deconstruction of waste rubber (ie scrap domestic tyres) (Deinma, Oluranti and Augustine, 2020). Although all technologies are still of interest today, pyrolysis is widely and extensively researched at laboratory and pilot plant-scale, in batch and continuous processes, including those induced by thermal and more recently, microwave heating.

Recently, however, industrial scale plants have come online given process advantages, changes in regulations, and technological maturity offered by suppliers. Pyrolysis in general, and other enhanced but similar technologies (ie Novum TVR) are applied to address waste rubber and is a sustainable source of fuel oils and petrochemical substitutes. It may help in compensating the progressive consumption of petroleum-based products, valourising an important and complex waste (large tyres, conveyor belting, hoses, and pipes) in the circular economy framework, and to divert rubber waste from landfill. The current fashion of crumbing rubber being applied to roads, pavements and even playgrounds is not a long-term solution, as 'spreading waste around' does not address the environmental problem of volatile emissions and contamination outside of landfill or open cut mines.

Further reference is made to the Tyre Particle Health, Environment and Safety Report, prepared by Tyre Stewardship Australia, February 2022, in which a literature review is offered to support premature views and decisions of applying crumbed rubber in roadbuilding and playgrounds. The point has to be made that if rubber in landfill or as micro-sized particles in water ways is a problem, why would crumb rubber in its current applications be allowed.

In general, pyrolysis is a thermochemical process during which rubber (polymers) is deconstructed in the absence of oxygen, as volatile matter, from the carbonaceous structure fixed by the carbon black (CB), added inorganic elements and steel wiring used in tyre manufacture. Thus, a mixture of gaseous products and condensable hydrocarbons are produced, as well as solid fractions (rCB and steel). This leads to a removal of waste rubber, producing usable products which have the potential for application in a true circular economy.

### Raw material

Rubber waste streams in their original use forms can be processed without any preparation (ie cutting, grinding, shredding, and chipping) prior to processing. This makes the TVR technology of Novum unique in that there is a huge saving in handling time and energy (read less waste) in pre-treating waste (Figure 2). The TVR system is a batch system, and the GCR-103 reactor can process up to 13 600 kg per batch. Whole OTR tyres/baled (Figure 3) commercial tyres, conveyor belting, and tracks are placed in the reactor for processing.

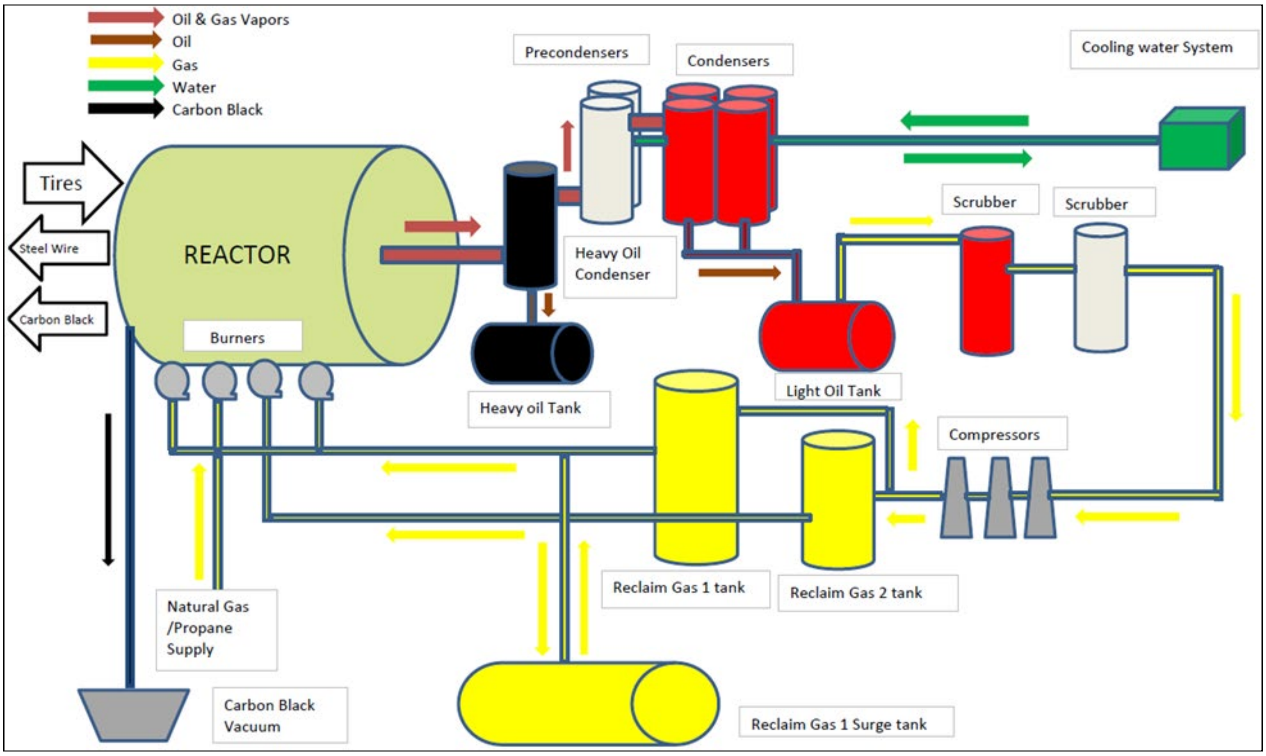


**FIG 2** – Diagram depicting the TVR process, with special reference to movement of oil and gas, water and rCB.

**FIG 3** – Photo of the front-end of a GCI-103 reactor in Rome, GA, USA. Note the mixed waste rubber source.

## Thermal reactor operations

The TVR reactor system (Figure 2) thermally depolymerises waste rubber in a reduced oxygen atmosphere. Oxygen is removed from the reactor chamber prior to the heating process by purging the chamber with nitrogen. The external wall of the reactor drum is heated by PLC (Programmable Logic Controller) controlled burners that are fuelled by natural gas, propane and/or scrubbed reclaim gas derived from the process.

The burners are controlled to produce a heating curve with predetermined temperatures over time so as to optimise the quantity and quality of the by-products. The reactor system is totally closed to the atmosphere and is operated under a negative pressure slightly below atmospheric. 100 per cent of the vapour generated by the process is captured and stored as gas and fuel. The TVR system utilises a cylindrical steel reactor that rotates during the heating and cooldown process. Rotating the reactor facilitates homogeneous heating of the feed stock. The total production cycle for the GCR-103 model, including loading, heating, cooling, and unloading will take approximately 16 hours. During the operation cycle the reactor system is monitored and controlled from a central control room.

## Recovery of oil and gas

As the reactor heats up it will begin to produce gas vapours that are pulled from the reactor under a vacuum created by two or three gas compressors. As the vapours travel through an external cooling circuit, they will condense and separate from the vapour stream at different temperatures. The heavy oil is first to separate. It is collected in an intermediate holding vessel and later pumped to an outside storage tank. The typical viscosity index of this oil is 40. The vapours continue through two pre-condensers and then a set of four water cooled condensers where the separation of the light oil takes place. This oil is collected in another intermediate holding tank. The viscosity index of this oil is 14 (similar to diesel fuel). The oil is later pumped to an outside storage tank. The vapours that remain travel through two scrubbers and filters to remove any remaining liquid and contaminates and are pumped to a Reclaim Gas storage tank. This tank holds the Reclaim Gas and can be mixed with natural gas or propane at the reactor burners. A secondary storage tank located outside the building will hold additional quantities of reclaim gas until it is used to heat the reactor.

## Carbon black and steel handling

The reactor is stopped for removal of the carbon black and steel when the cooling phase reaches a temperature of approximately 90° Celsius. The carbon black is now removed with a vacuum system equipped with a 100 HP vacuum pump and a high efficiency particulate air (HEPA) filter unit. The reactor drum is rotated as the vacuum control circuit alternates between a vacuum cycle and as the carbon black is extracted, pushing the rCB towards the vacuum port. It dumps the material from the vacuum hopper into a 1 t capacity bulk bag. The bag is then weighed and stored in an enclosed area. The reactor door is now opened, and the steel wire is removed with a telehandler with an extended boom. The boom and fork are extended into the reactor and the wire mass (Figure 4) is lifted out. It is placed in a carrier and is then weighed and placed in a dedicated dumpster for sale to a scrap metal dealer.

**FIG 4** – Photo of the steel after the process had run its full course. Front-end of a GCI-103 reactor. Note the mixed steel in the reactor at the end of the process.

## Wastewater and emissions

Water (due primarily to rainwater in the tyres) also comes out of the tyres during the vapourisation process. This water is separated from the light oil using a centrifuge. The light oil condensers are cooled to approximately 30°C or less by circulating water through the condenser water jacket. The cooling water is circulated through evaporative cooling towers by one of two redundant water pumps. There is no wastewater stream from the cooling circuit. Water that evaporates off in the cooling towers is made up by adding water to a reservoir tank within the cooling circuit. One of the systems scrubbers contains a citric acid solution to remove ammonium bicarbonate from the gas stream. The acid solution in the scrubber is replenished with citric acid to maintain a pH of approximately three. Over time the solution must be replaced, and the waste solution must be disposed of.

The flue gas and air quality in the reactor building was evaluated at the Rome, Georgia facility in April 2015. An independent testing firm (Advanced Industrial Resources Inc.) completed the testing over a two-day operating period. Their results showed that all emission levels were below the US Permit Threshold limits. These results show that in the US, no flue gas scrubber or thermal oxidiser is needed to operate our reactor system.

## Process statistics

The Fort McMurray plant, Alberta, Canada, which is owned by TTRC Inc. processed 13 022 t of waste OTR tyres (960 loads) in 2018/2019. The waste rubber equites to approximately 1365 large OTR tyres or the equivalent of 974 000 car tyres. The plant was operational for 320 days of the year (ie 88 per cent operational time), during which a total of 5 603 273 litres of fuel oil, 4769 t rCB, 2045 t steel, and 496 t process gas was produced.

In 2019, a second TVR plant located in Rome, Georgia, USA (owned by Green Carbon Inc.), operated for 260 days of the year (55 per cent operational time). Based on the waste rubber supply of 292 t (67 loads), 123 710 litres of fuel oil, 117.2 t rCB, 35.2 t steel and 29.3 t process gas. The Rome plant has been operational as an R&D facility.

# PRODUCTS RECOVERED

## Introduction

Historically, the recycling industry (rubber) has been focusing on the waste disposal process, and not specifically on the quality of products. Novum has proven that their TVR technology is viable and assumed a business plan and model with the focus on the production of: (a) high quality base products with initial offtake, and the ability to upgrade in quality through beneficiation, and (b) unique applications which offset the production of virgin commodities like diesel, fuel oil, carbon black, process gas and steel, creating products for local use and thus develop circular to develop a circular economy withing the local communities.

### *Fuel oil*

The major component in waste rubber products is a mixture of natural rubber and synthetic polymers. The cross-linked thermoset resulting from the vulcanisation process of the thermoplastics resists biological or thermal deconstruction of tyres and conveyor belting. Combustion of rubber in open atmosphere releases dioxin, furans, polyaromatic hydrocarbons (PAHs), oxides of nitrogen, and volatile particulate matter into the environment (Czajczynska *et al*, 2020). Work by Banar *et al*, 2012; Pilusa, Shukla and Muzenda, 2013; Ahmad and Ahmad, 2013; Nabi, Masud and Alam, 2014; Neto *et al*, 2019; Rodriquez *et al*, 2011; Williams and Cunliffe, 1998) provide a balanced introduction regarding the process on rubber-derived fuel oil recovery and attempts to ungraded applications during the recent decade.

Novum produces fuel oil from vapour, which after various stages of condensation by cooling, produces a heavy and light fuel oils. Table 1 represents oil specifications according to major and trace component contents for heavy and light TVR fuel oil. Table 2 summarises a comparison of fuel properties determined for a range of samples produced by TVR (Intertek Report of Analysis: US320–0063093, Batch #4 Bubbled Heavy oil; Intertek Report of Analysis: US320–0063093, Batch #4 Bubbled Light oil; GCL Analysis: Batch #30549: Light Oil Sample GCLAB1393 (2019); SPL Certificate of Analysis: Report #1030–21110942–001A: OTR029, OTR030, OTR031 Sample (7/01/2022). The data is further comparison with standard diesel, D2 diesel, marine fuel oil (MFO)/diesel fuel oil (DFO) and heavy fuel oil (HFO) (Chevron, 2018).

**TABLE 1**

Representation of oil specifications from a mixed rubber feedstock. Analyses for light and heavy oil fractions are presented as Major and Trace component content.

Novum Energy Australia: TVR Oil specifications as tested on mixed rubber feedstock

| Light Oil Major Components | |
|---|---|
| Carbon Range C5 - C29 | Wt. % |
| Paraffin | 2.755 |
| Isoparaffins | 54.391 |
| Naphthenics | 6.846 |
| Aromatics | 32.515 |
| Olefins | 3.494 |

| Heavy Oil Major Components | |
|---|---|
| Carbon Range C3 - C30+ | Wt. % |
| Paraffin | 5.865 |
| Isoparaffins | 77.225 |
| Naphthenics | 1.631 |
| Aromatics | 13.034 |
| Olefins | 2.245 |

| Light Oil Major Trace Components | |
|---|---|
| Carbon Range C7 - C20 | Wt. % |
| N-Hexane | 0.022 |
| Benzene | 0.171 |
| Ethyl Benzene | 2.213 |
| Toluene | 3.059 |
| Meta-Xylene | 3.224 |
| Para-Xylene | 0.956 |
| Ortho-Xylene | 0.911 |
| Xylenes | 5.091 |
| Pristane | 0.059 |
| Naphthalene | 0.417 |
| 1-Methyl Naphthalene | 0.352 |
| C17 | 0.233 |
| C18 | 0.023 |
| Phytane | 0.062 |
| 2-Methyl Naphthalene | 0.032 |

| Heavy Oil Major Trace Components | |
|---|---|
| Carbon Range C10 - C22 | Wt. % |
| N-Hexane | 0.05 |
| Benzene | 0.061 |
| Ethyl Benzene | 0.356 |
| Toluene | 0.563 |
| Meta-Xylene | 0.69 |
| Para-Xylene | 0.099 |
| Ortho-Xylene | 0.162 |
| Xylenes | 0.951 |
| Pristane | n.a. |
| Naphthalene | 0.232 |
| 1-Methyl Naphthalene | 0.373 |
| C17 | 0.25 |
| C18 | 0.252 |
| Phytane | n.a. |
| 2-Methyl Naphthalene | 0.022 |
| 2,24-Tri Methylpentane | 0.05 |

**TABLE 2**

Comparison of fuel properties determined for a range of raw samples produced from TVR. Note the further comparison with final standard diesel, D2 diesel, marine fuel oil (MFO), diesel fuel oil (DFO) and heavy fuel oil (HFO).

| | | | | TVR FUEL PRODUCTS, STANDARDS AND REFERENCES | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| TVR FUEL OIL ANALYSIS | | | | OTR029 | OTR030 | 0TR031 | STD | D2 | MFO / | HFO[4] |
| TEST # | PROPERTY | UNITS | METHOD | LFO[2,3,5] | HFO[1,5] | LFO+HFO[5] | DIESEL[4] | DIESEL[4] | DFO[4] | |
| 1 | Density @ 15°C | kg/l | ASTM D4052 | 0.869 | 0.965 | 0.916 | 0.828 | 0.876 | 0.840 | 0.989 |
| 2 | Flash Point | °C | ASTM D93 | 0 | 10 | | 42 | 52 | 60 | 45 |
| 3 | Kinematic Viscosity @ 40° | cSt | ASTM D445 | 0.962 | 6.696 | 1.994 | 2.59 | 4.1 | 6 | 7 |
| | | SUS | ASTM D445 | 28.98 | 47.82 | 32.59 | | | | |
| 4 | Kinematic Viscosity @ 50° | cSt | ASTM D445 | 0.865 | 5.073 | 1.701 | | | | |
| | | SUS | ASTM D445 | 28.66 | 42.68 | 31.62 | | | | |
| 5 | Pour Point | °C | ASTM D97 | -60 | -60 | -48 | 3 | 30 | | |
| 6 | Acidity | number | ASTM D664 | 22 | 11.2 | | | | | |
| 7 | Sulphur | % mass | ASTM D4294 | 0.490 | 0.839 | 0.635 | 0.32 | 0.05 | 0.68 | 2.3 |
| 8 | Ash Content | % mass | ASTM D482 | 0.057 | <0.001 | 0.037 | 0.00 | 0.01 | | |
| 9 | Water by distillation | Vol % | ASTM D95 | 0.2 | 0.7 | 0.4 | | | | |
| 10 | Sediment by Extraction | % mass | ASTM D473 | 0 | 0.3 | 0.2 | 0.00 | 0.05 | 0.1 | |
| 11 | Gross Calorific Value | MJ/kg | ASTM D240 | 43.41 | 42.35 | 40.89 | 45.22 | | 42.30 | 42.30 |
| | | BTU/lb | | 18,663 | 18,207 | 17,580 | 19,441 | | 18,186 | 18,186 |
| 12 | Boiling Point | Initial °C | ASTM D7213 | 26.0 | 65.5 | 32.0 | | 282 | 180 | |
| | | Final °C | | 504.4 | 600.0 | 576.0 | | | | |
| 13 | Carbon Residue | % mass | ASTM D189 | 0.92 | 3.48 | 2.18 | 0.08 | 0.35 | | |
| 14 | Cetane Number (CN) | number | ASTM D613 | 5.3 | 27.3 | 21.0 | 48 | 41 | 50.7 | 35 |
| 15 | PAH | % mass | ASTM D6591 | 8 | 11 | 8 | 6 | 11 | 30.6 | |
| 16 | PAH range analysis | ppm | GC/MS USEPA 8 | | | | | | | |
| 17 | Iron | ppm | ASTM D5185–18 | 27 | 37 | | | | | |
| 18 | Sodium | ppm | | 1.5 | 1.4 | | | | | |
| 19 | Nickel | ppm | | <0.1 | <0.1 | | | | | |
| 20 | Potassium | ppm | | 0.2 | 0.3 | | | | | |
| 21 | Vanadium | ppm | | 0.1 | 0.1 | | | | | |
| 22 | Carbon | % | ASTM D5291-C | 84 | 87 | | | | | |
| 23 | Hydrogen | % | | | | | | | | |
| 14 | Nitrogen | % | | <0.75 | <0.75 | | | | | |
| 25 | Oxygen | % | | | 1.9 | | | | | |
| 26 | Fluorine | ppm | ASTM D7359 | | | | | | | |
| 27 | Chlorine | ppm | | 43 | 55 | | | | | |

Based on the comparison (Table 2), the reclaimed fuel oil samples depict high sulfur and Poly-cyclic Aromatic Hydrocarbons (PAH) contents, and an associated low flash point. These attributes limit the use as fuel in ANFO (Ammonium Nitrate Fuel Oil) emulsions and use in stand-alone fuel applications. Novum improved the fuel quality by resorted to several solutions: (a) water, ultra-light fuel fractions (ie benzene, naphthalene etc) and sulfur associated with PAH are removed by an in-line centrifugal separator; (b) hydrotreatment of the fuel, and (c) by addition of scavenger additives to react and isolate sulfur (thus decreasing the PAH content). By these treatments, the low flash point is increased to between 60–72°C, while the sulfur and PAH contents is decreased to less than 300 ppm and 2 per cent respectively.

### *Reclaimed carbon black (rCB)*

In the simplest terms, carbon black is elemental carbon in the form of extremely fine particles having an amorphous molecular structure. Buried within the amorphous mass is an infrastructure of microcrystalline arrays of condensed rings. These arrays are like the layered condensed ring form exhibited by graphite. The orientation of the arrays within the amorphous mass appears to be random, consequently a percentage of strands have open edges of their layer planes at the surface of the particle (Figure 5) (Voll and Kleinschmidt, 2002; The Fredonia Group, 2010).

XRD and XRF analysis of reclaimed tyre carbon black (RTCB) showed that the main contaminates are silica, sodium and potassium salts, zinc sulfide, and calcium minerals (Figure 6). SEM micrographs of N330 and RTCB particles are shown in Figure 7.

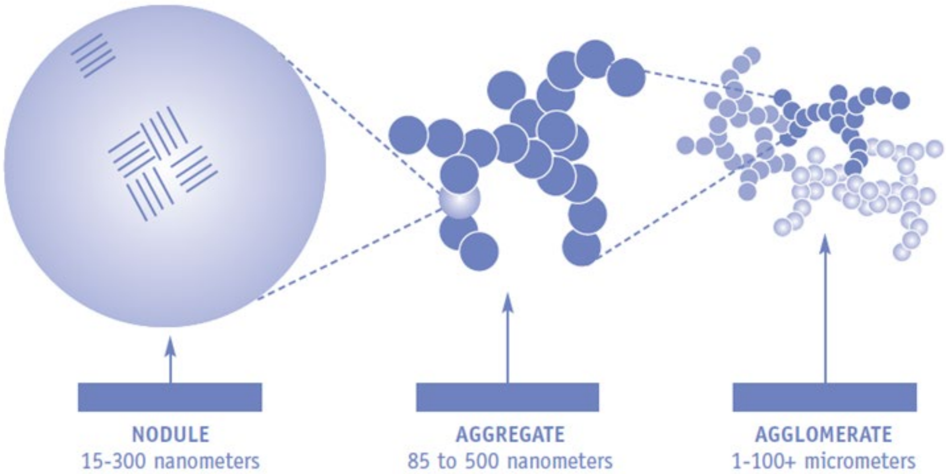


**FIG 5** – Diagram displaying the structure of Carbon Black, which consists of nodules which vary in diameter of 15–300 nm (0.000001 to 0.0003 mm), aggregates, which vary in diameter of 85–500 nm (0.000085 to 0.0005 mm) and aggregates which range between 1–500 µm (0.001 to 0.5 mm) in diameter. Also, note the random orientation and open edges of the stands.

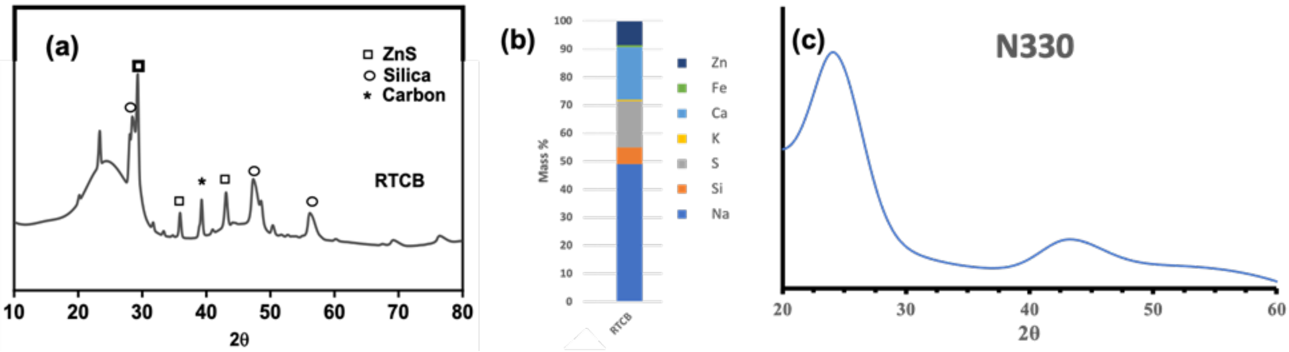


**FIG 6** – (a) X-ray diffractogram of reclaimed tyre carbon black (RTCB ('recycled tyre carbon black')) or rCB particles, (b) X-ray fluorescence of RTCB, mass fraction of contaminates, (c) X-ray diffractogram of N330 carbon black (Smith, 2021).

Importantly, the analysis of waste rubber changes based on product type or design (ie moisture, binders, or remnants of binder material). In the case of conveyor belting and OTR tyres, major elements like $Na_2O$, $CaO$, $MgO$, $K_2O$, $Al_2O_3$, $SiO_2$, $P_2O_5$, $TiO_2$, $Mn_2O_3$ and $Fe_2O_3$, and trace elements have also been determined as Cr, Sn, V, Cd, Pb, Co, Br and S (Table 3).

Other non-organic elements consist of refractory dust, metallic oxides, coke, and water-soluble salts. The levels of these materials are controlled within specified limits and are usually not significant in the final rubber product.

The most recent report on the repurposing of zinc in rCB as a fertiliser resource (Greer, Wiebe and Bremer, 2021) shows that Novum's proprietary TVR technology results in a high quality rCB that holds a range in nutrients, including Zn, which can be repurposed to fertiliser (Table 3). A circular economy of nutrients has always been part of the agricultural system as manures, composts, and other biological products. However, with three billion tyres sold worldwide per annum, containing more than 2.2 Mt of ZnO, repurposing this nutrient to agricultural soil is a much-preferred scenario compared to effectively stranding it within rubber crumb/aggregate products, reformed rubber ramps, curbs, and mats (Shercom Industries Inc., 2020).

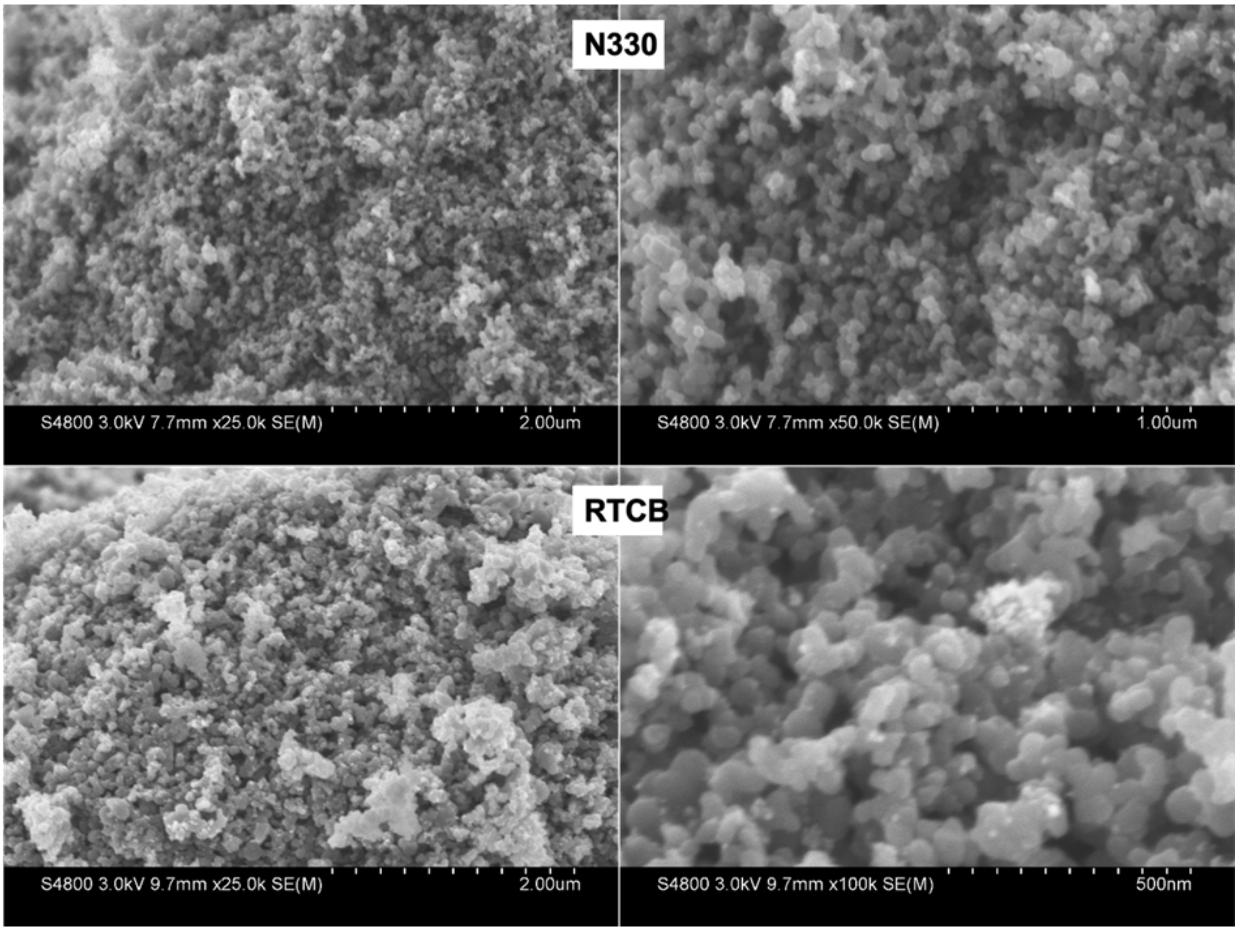


**FIG 7** – Scanning electron micrographs of N330 carbon black particles and Novum rCB particles (Smith, 2021).

Further new evidence has reported raw tyre rubber will experience leaching/chemical transformations that deleteriously impacts the environment (Tian *et al*, 2020). Given the efficiency of oil and gas extraction with the Novum's TVR process, the rCB appears to have very few volatile organic components left in the product. As such land application as a beneficial plant nutrient appears to be a very promising win-win.

Ongoing research conducted by Novum (Smith, 2021) demonstrates an energy and cost-efficient method to purify rCB recovered specifically from the TVR process, thus enhancing the quality for recycling back into tyres or conveyor belting. Compared to conventional methods, the methods explored demonstrate improvements in product purity by 40–60 per cent, a reduction in water requirements by 50 per cent, reduction in processing costs by 80–90 per cent, and processing/embodied energy by 70–95 per cent.

A techno-economic feasibility analysis of a 10 t/h pilot scale operation shows that these new demineralisation methods explored are potentially economically viable. Further development and implication of this technology would help enable up to 30 per cent offsets of Carbon Black (CB) by rCB.

**TABLE 3**

Table representing proximate and chemical analyses. Reference is also made to the Ash mineral analyses of the reclaimed Carbon Black (rCB) produced from a range of waste rubber supplies.

| SAMPLE TYPE | PROXIMATE ANALYSIS | | | | | |
|---|---|---|---|---|---|---|
| | ID | Batch # | rCB Water % | rCB Ash % | rCB Volatile | Fixed Carbon |
| Tracks, domesitic tyres mixed | MIX01 | 300720 | 0.956 | 14.77 | 4.074 | 80.20 |
| Conveyor belting 1 | CB01 | 300421 | 4.823 | 26.00 | 6.139 | 63.04 |
| Conveyor belting 2 | CB02 | 300421 | 5.055 | 25.84 | 5.834 | 63.27 |
| Bridgesone 63'' OTR 1 | OTR01 | 300720 | 0.959 | 19.20 | 4.223 | 75.62 |
| Bridgesone 63'' OTR 2 | OTR02 | 300720 | 1.455 | 19.06 | 3.091 | 76.39 |

| SAMPLE TYPE | rCB Chemical Analysis EDXRF ASTM D4294 (content expressed as Wt.%) | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | ID | Ca | Zn | S | Si | Al | Fe | Ti | Cl | Sn | Cd | P | Br | Cu | Pb |
| Tracks, domesitic tyres mixed | MIX01 | 0.65 | 0.55 | 0.85 | 0.50 | 0.07 | 0.086 | 0.028 | 0.014 | 0.039 | 0.031 | 0.015 | 0.008 | 0.008 | n.a. |
| Conveyor belting 1 | CB01 | 0.70 | 0.50 | 0.60 | 1.40 | 0.80 | 0.300 | 0.002 | 0.500 | 0.032 | 0.029 | 0.019 | 0.001 | 0.002 | 0.100 |
| Conveyor belting 2 | CB02 | 0.70 | 0.50 | 0.60 | 1.40 | 0.70 | 0.200 | 0.002 | 0.500 | 0.033 | 0.029 | 0.019 | 0.001 | 0.002 | 0.100 |
| Bridgesone 63'' OTR 1 | OTR01 | 0.30 | 0.60 | 0.90 | 1.70 | 0.06 | 0.024 | 0.002 | 0.008 | 0.027 | 0.029 | 0.016 | 0.016 | 0.005 | n.a. |
| Bridgesone 63'' OTR 2 | OTR02 | 0.30 | 0.60 | 0.80 | 1.60 | 0.05 | 0.051 | 0.003 | 0.008 | 0.032 | 0.030 | 0.016 | 0.014 | 0.009 | n.a. |

| SAMPLE TYPE | Ash Mineral Analysis EDXRF ASTM D4294 (content expressed as Wt.%) | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | ID | $Na_2O$ | MgO | $AL_2O_3$ | $SiO_2$ | $P_2O_5$ | $SO_3$ | Cl | $K_2O$ | CaO | $TiO_2$ | $CrO_3$ | $Mn_2O_3$ | $Fe_2O_3$ | ZnO | SrO |
| Tracks, domesitic tyres mixed | MIX01 | 5.237 | 1.070 | 4.025 | 34.758 | 0.922 | 13.809 | 0.103 | 0.696 | 6.941 | 1.157 | 0.008 | 0.017 | 1.807 | 29.446 | 0.003 |
| Conveyor belting 1 | CB01 | 1.212 | 0.209 | 17.277 | 43.967 | 0.196 | 7.341 | 0.097 | 0.240 | 5.940 | 0.042 | 0.004 | 0.025 | 3.743 | 19.701 | 0.004 |
| Conveyor belting 2 | CB02 | 1.188 | 0.183 | 17.400 | 44.649 | 0.265 | 7.472 | 0.107 | 0.244 | 5.734 | 0.052 | 0.005 | 0.024 | 3.633 | 19.043 | n.a. |
| Bridgesone 63'' OTR 1 | OTR01 | 3.524 | 0.132 | 1.266 | 57.257 | 0.580 | 6.231 | 0.054 | 0.654 | 2.160 | 0.045 | 0.003 | 0.008 | 0.406 | 27.680 | 0.004 |
| Bridgesone 63'' OTR 2 | OTR02 | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. | n.a. |

## APPLICATIONS AND THE FUTURE

### Light and heavy fuel oil

Reclaimed fuel can be applied in various renewable technologies and products (Figure 8) without or after limited, cost-effective beneficiation:

- 75 per cent bio-based light (LFO) and 71 per cent bio-based heavy (HFO) fuel can be applied as standard diesel for blending, D2 diesel, marine fuel oil (MFO)/diesel fuel oil (DFO) and heavy fuel oil (HFO).
- D2 type fuel can be used as part of an ANFO emulsion, whereby the low PAH and sulfur content will not react adversely with the AN (ammonium nitrate) prill. An associated high flash point will ensure the safety of the emulsion in the field. The use of Novum's D2 fuel can also be potentially used as an efficient binder and wetting agent in CHPP (Coal Handling and Preparation Plants).
- Base fuel oil for solvents and other hydrocarbons (Naphtha, Paraffin, Benzene, D-Limonene).
- Generation of electricity in generators.

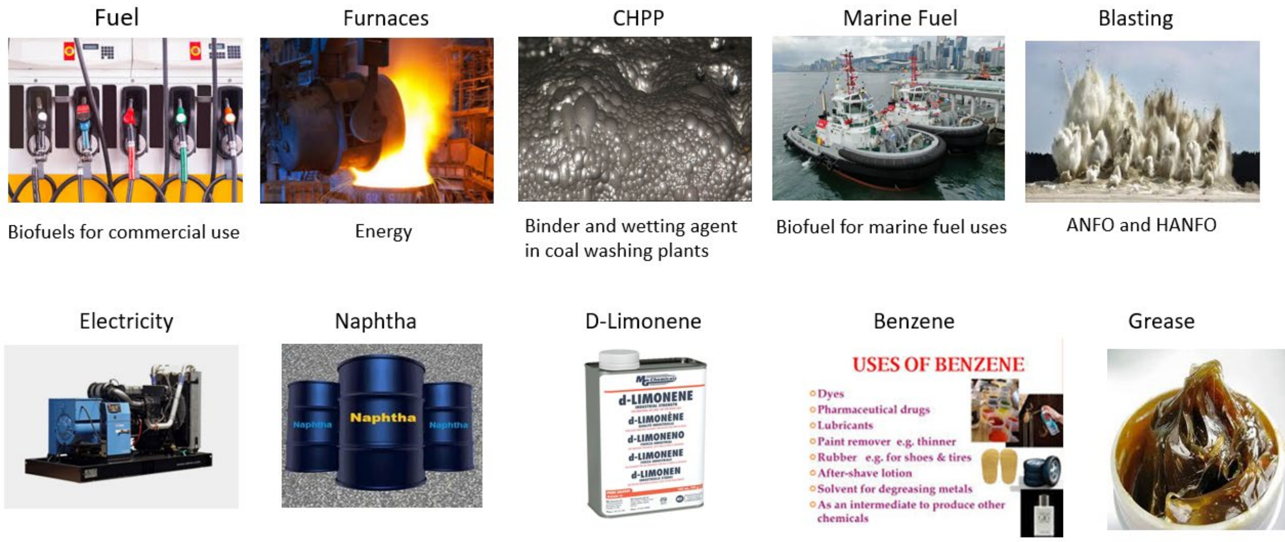


**FIG 8** – Solutions for diverse markets – fuel oil product specialties have a large number of applications.

### Reclaimed carbon black (rCB)

rCB requires a variable degree of modification (demineralisation, screening, milling, briquetting, pelletising for a range of end-use applications (Figure 9). However, the quality of rCB can be controlled or enhanced by mixing waste rubber feedstock for certain chemical compositions. The process is also important, with special reference to maximum heat, resident time in the reactor, heating rate and cutting points.

These process configurating will apply for the production and quality specifications for light and heavy fuel oils and will thereby have an effect on the ultimate rCB quality. Types of feedstocks and the ultimate rubber use are important, as seen above in previous *Reclaimed carbon black (rCB)* section. rCB composition will be different between domestic tyres, OTR tyres, conveyor belting and other rubber-based waste. Special reference is made to conveyor belting, where steel cord is included for ultimate strength and carry potential. These steel cord vary between 2 mm to 10 mm and need to be 'glued' to ensure the rubber-steel bonds. During pyrolysis, these additional additives (ie glue) can be problematic.

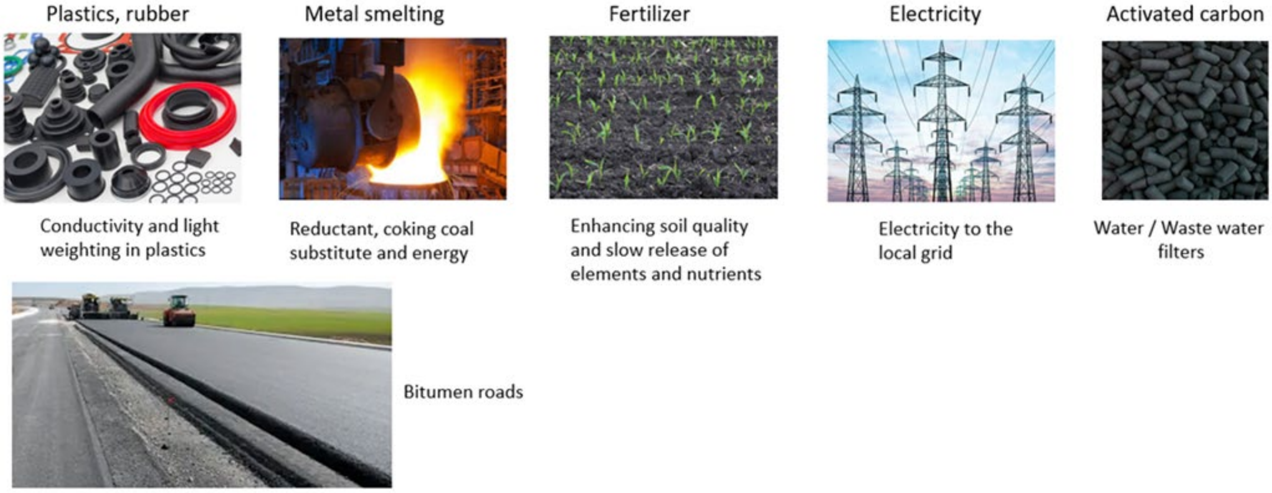


**FIG 9** – Solutions for diverse markets – reclaimed carbon black (rCB) product specialties have a large number of applications.

### *Metallurgy*

Iron and steelmaking are the most important industrial sectors which have a great impact on the global growth and economy. The steel production is sharply increased in the recent years to reach more than 1662 Mt in 2014, up by 1.2 per cent compared to 2013 (World Steel Association, 2015). By 2050, steel usage is expected to increase to become 1.5 times higher than present levels to meet the growing population. Steel manufacturing is also one of the largest energy and carbon-consuming sectors. The global energy consumption in steelmaking is estimated to be about 20 per cent of the annual industrial energy requirements. Fossil fuels represent the main source of heat and reducing agents in steelmaking and so is major contributor to global anthropogenic $CO_2$ emissions.

$CO_2$ emission from iron and steelmaking was 2.3 Bt in 2007 while it is expected to reach 3.0 Bt in 2050 (International Energy Agency, 2010, 2021). The reduction of specific energy consumption and gas emissions have become top priorities for the iron and steelmaking industry due to the dynamic growth of energy prices as well as the commitment of governments to decrease $CO_2$ emissions according to Kyoto Protocol (Birat and Hanrot, 2005).

Reference is made to Novum Energy Australia (2020a, 2020b) summarising that from one plant (set-up with one GCR-104 and two GCR-103 reactors) a waste rubber source of 13 500 t can support the annual production of 4 701 722 L of HFO+LFO (heavy and light fuel oil), 3177 t rCB, 1525 t steel and 387 193 m$^3$ syngas. Of importance is that rCB is a source of energy with the following benefits when introduced into blast furnaces:

- rCB is a cost-effective, high energy, low emission energy and reductive source to be applied in various smelting operations.
- Compared to plastic energy sources, rCB does not require expensive collection and sorting.
- Also compared to charcoal energy sources, rCB does not require expensive stumpage, hauling, chipping, transportation, and two phases of carbonisation phases.
- Novum is of the view that while the mining industries have available waste rubber sources, they [mining companies and processing industries] should refrain of using wood resources for energy and reduction requirements.
- Milling and drying of rCB is not required, although rCB will need to be pelletised.
- rCB is custom-made and can be supplied as fines (32–250 µm) or in granulated (2–6 mm) form or briquettes (40–55 mm).
- low moisture content is preferred between 2 to <13 per cent for Pulverised Coal Injection (PCI) coal, while compared to rCB, an average of only 3.40 per cent lowers transport and drying costs.

- the use of rCB leads to a lower consumption of both coke and pulverised coal.
- rCB compares well with Low Volatile bituminous coal, with a Replacement Ration (RR) = 0.90, RAFT (Raceway Adiabatic Flame Temperature) change of 1.00°C/kg and a Calorific Value = 33.5 MJ/kg.
- coke or PCI coal can be replaced by injecting rCB (between 0.8 and 1 kg/kg of coke).
- rCB leads to lower energy consumption in production of tHM (ton hot metal unit).
- rCB leads to a decrease in carbon dioxide ($CO_2$) emissions (low H/C ratio).
- rCB leads to a high energy efficiency of <90 per cent.
- The sulfur and alkali contents of rCB is lower than average PCI coal.
- rCB leads to low emissions of dioxins and furans.

A second use for rCB as a solution is as a substitute for metallurgical coal or specifically Nut Coke in blast furnace applications without the availability and cost constraints. Novum's reclaimed Carbon Black product (Novum Green rCB) has properties similar to coke and offers the potential to reduce cost of iron and base metal production.

The flexible pellet size can enhance burden permeability and thus increasing the blast furnace productivity by 2.5 per cent. In the cohesive zone, where the ferrous burden softens and then melts, permeability is hampered. It has been found that by charging the ferrous burden with <20 per cent (20–40 mm) rCB pellets, the permeability in the cohesive zone is improved to such a degree that 'reduction retardation' does not occur.

- rCB pellets have high strength and low reactivity, similar to metallurgical coke.
- 20–40 mm rCB pellets replace nut coke, providing a positive effect on burden permeability.
- All of the points above are applicable.

### *Roadmaking*

Road traffic combined with large ranges of temperature is leading to urgent attention to improve asphalt performance and durability. Commercial carbon black (CB) was first used as a bitumen modifier about 30 years ago by the Australian Carbon Black Pty Ltd in collaboration with the Australian Road Research Board (Aliotti, 1962; Bahia, 1994). The origin of this idea was based on the concept that commercial carbon black is known to reinforce rubber polymers, thus improving their durability and UV protection. In fact, to attain high road performance and to be convenient for specific climate and traffic conditions, binders must be modified.

Novum investigated the source and nature of Carbon Black (CB) types (ie CB and rCB) and bitumen-related physio-chemical properties in an attempt to show that mixing the two (bitumen and CB/rCB) will: (a) improve the range of bitumen uses in certain conditions, (b) renew and expand current applications of bitumen with specific binders, (c) environmental management and bitumen producers now can rely on the road construction industry to develop markets for waste materials which can be cheaply used as reinforcing agents, (d) illustrate that the application of polymers and waste tyre-derived carbon black (rCB) supports an increase in the rigidity and the elasticity of binders, and that (e) the addition of rCB/CB further reduces cracking, bleeding and rutting potential for heavy road traffic pavement sections (Chebil, Chaala and Roy, 1997).

Novum's view is that the application of rCB as binder to bitumen will support the modern-day and future roadbuilding in a smaller carbon and circular economy setting:

- rCB binder-bitumen roads have an extended life of at least 60 per cent.
- An assumed increase in cost due to binder modification is outstripped by the bitumen quality and longevity of the rCB-bitumen.
- rCB-bitumen is seen as a specific product to be applied in areas of high traffic frequency and loading, thus preventing cracking, rutting, and bleeding.

- Road construction companies, by using rCB-bitumen products will be able to claim Carbon Credits for diverting waste from landfill.
- rCB and fuel oil is produced in the TVR process which offsets the use of expensive new/virgin products in a near neutral carbon way.
- Novum can produce sufficient volumes to support an extensive roll-out of this specific product.

### *Fertiliser*

Reference is made to a recent patent (WO 2021/122500 AI) lodged under the International Application Published under the Patent Cooperation Treaty (PCT), World Intellectual Property Organization, dated 15 December 2020 by BASF SE in which the present invention comprises the use of Carbon Black for soil conditioning (worked into the topsoil), eg to promote growth of plants, to promote soil drainage and to prevent erosion, evaporation and silting up. The above reference is important as it supports Novum base case of applying rCB as a slow-release carrier fertiliser.

A report by Novum Energy Australia (2021) shows that (i) the ash content in rCB is no longer seen as an obstacle, and (ii) that in certain applications, the ash and its specific inorganic components can be seen as a benefit. In Novum Energy Australia (2020c), the report refers to rCB upgrade methods and zinc recovery, it is shown that up to 76 per cent of the ash content held by rCB can be removed. This process can reduce all possible deleterious components like S and sulfur-associated polycyclic aromatic hydrocarbons (PAH) to below 0.04 wt per cent. It must be noted that the inorganic ash content (except for sulfur and S-related components mentioned) and thus elemental content can have a positive effect on plant growth when leached from the rCB in fertiliser form.

The application of rCB as a slow-release carrier fertiliser, a soil conditioner (worked into the topsoil), and has the ability to promote growth of plants, control soil drainage and to prevent erosion, evaporation, and silting.

In preparation of the rCB for fertiliser applications, Novum is prescribing:

- A milling process to liberate elements held by the carbon aggregate – agglomerate structure. Thereafter, demineralisation under Moderate Temperature and Pressure (MTP) conditions was found to be the most effective treatment method explored by Novum. The rCB material is rinsed, dried, and then milled for 90 seconds.
- A further process of re-heating of the rCB will lead to the activation of the material. Activation of rCB will increase the Total Pore Volume from 0.203–0.501 $m^3/g$, which provides the potential for the rCB to absorb 148 per cent more additives/fertiliser components.
- The pelletising (by disk pelletiser) will produce pellets (2–4 mm in diameter). The binder can vary from water soap, wax, gypsum, and carbonate, which will, in time and based on soil and climate conditions, dissolve, and so release different organic or inorganic fertiliser additives, eg agrochemical active substance from the groups of fungicides, bactericides, herbicides and/or plant growth regulators.

# CIRCULAR ECONOMY

## Introduction

As per definition, creating reusable products from products that use natural resources, or recycled to displace demand, and in the process of reducing emissions forms the basis of a circular economy. The effect of the circular economy is enhanced by generating this capability in regional area and dispersing the products locally for application in local or new industries.

The process of circular economies requires efficient management of waste to create true value by creating products from waste to be used across various applications. Novum has a primary aim of exploiting new opportunities for reclaimed carbon-based product.

The circular economy is shaping our thinking both around how we can do things differently and how we can do different things in times of unprecedented change – notably, in the fields of digital, materials science and biological science.

Anglo American (Anglo) is embedding circular economy principles into their thinking and decision-making and are applying them on an increasing scale as we seek to create greater value throughout the mining value chain through more 'circular' activities (eg fuel, for energy and processing, plastic and rubber products, fertiliser and soil enhancers required for the reclamation of mining areas (Figure 10). Developing opportunities in new industry in regional locations is shaping our thinking. An added key factor of diverting waste from landfill is a key factor that participants can physically contribute to from an ESG (Environmental, Social, and Governance) focus.

**FIG 10** – Waste 63' OTR tyres deemed to be diverted from landfill.

As a case study of Novum's approach in this new field of responsibility, Anglo American's Purpose, which is 're-imagining mining to improve people's lives' rings true. In Anglo American's (2020) Sustainability Report it is reiterated that with their company values and guiding strategy, they [Anglo] approaches climate change as a commitment – because it is part of the global response to climate change, and also because it is the right thing to do.

Anglo's adoption of zero waste to landfill (Figure 10) commitment started in 2020 by building a comprehensive baseline database of waste metrics to inform the specific requirements like internal standards and systems to assist in driving progress toward alternative solutions. Such an initiative was to partner with Novum Energy Australia Pty Ltd to construct a rubber reclamation plant in central Queensland. This plant will process waste rubber to produce products for reuse and will enable pick-up and transfer of rubber waste from all Metallurgical coal mining sites.

This will mitigate issues that sites have with waste tyre and conveyor storage and contribute to the sustainability goal of reducing waste to landfill. Novum's view and an important part of their Anglo business plan is to address the large contingent of waste rubber and plastic produced by the mining industry, as a sustainable source for its TVR plants for the period of operation. Key areas Novum addresses include the management of waste by diverting it from landfill or burial in open cut pits, avoiding unnecessary transport emissions by treating locally, creating new industries in regional areas to enhance local economic development and contain the emissions generated by transport, produce products for direct reused locally or beneficiated to feed into a mining circular economy, and to scale the reclamation plants to match product types, volume, and ongoing waste volumes.

## SOCIAL LICENSE/ENVIRONMENTAL, SOCIAL AND GOVERNANCE (ESG)

Mining operations qualify in opportunities to commit to a circular economy (Figure 11). Most are qualified in three areas:

- generate waste, and thus providers of waste feedstock (rubber and plastics);
- consumers of product (simple application of a range of products) for energy, processing fluids, fertiliser for the reclamation of mined-out areas, and

- participants in supporting a sustainable circular economy (ie 'closing the loop') from waste rubber and plastic to high quality, usable products of value.

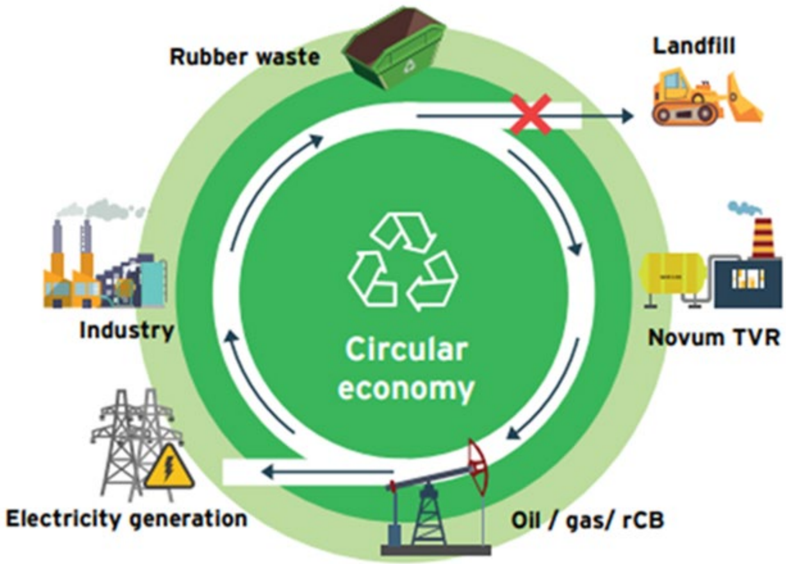


**FIG 11** – Circular Economy showing the cycle where waste rubber is not diverted to landfill, but is provided to Novum for reclamation, so producing fuel, process gas, rCB and steel, which becomes sources to electricity production, energy and fuel in the same industry which produced the waste.

As published, mining companies are committed to ESG targets and understand the right to operate in regional areas as a result of these commitments to achieve a Social License to Operate.

> *Social license comes down to the core of society acknowledging that they as a collective whole don't really understand a particular industry well enough in order to regulate it, so they trust that industry will do the right thing and regulate itself.* (Bolton, 2021)

Dairy Australia project officer Sarah Bolton presented on the topic of social licence at the Grower Group Alliance 2021 Forum held in Perth recently. Bolton further said:

> *But if that particular industry fails to self-regulate in a way that the community approves of, then it is likely that the community will intervene, despite their lack of understanding, in order to make sure that what that industry is doing is something that people will approve of.*
>
> *There are a whole lot of different approaches people have suggested we can go about preserving the importance of public trust, given how much it means to an industry and the monetary that is attached to being able to preserve our way of doing things.*

## CONCLUSION

While the constraints of a circular economy are realised, the governing Social Licence requirements continue to be improved and enhance through better waste management. The TVR technology provides key facets and options to provide a sustainable solution that matches operational aspects of the mining region.

The ability to include aspects of waste management improvements into a company's ESG program provides additional steps to carbon neutrality and improvements in Social Licence. The first steps, however, is (a) to recognise that there are proven, patented technologies available which can transform waste rubber into clean energy and other high quality raw materials. If the processes (b) produce low to no emissions and rely on low energy input, and that the transportable, scalable, and adaptable plants (ie build the plant in the industry region, scaled to volume requirements), it is then that dirty industries can be transformed into producers of raw and beneficiated minerals and clean energy from waste. Removing a costly problem, replacing it with a sustainable solution.

## ACKNOWLEDGEMENTS

Novum Energy Australia continues to collaborate with supporters within the mining industry that have engaged to provide sustainable waste supplies enabling establishment of these circular economy principals.

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# Social licence to operate – number one issue for mining companies

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## ABSTRACT

Social licence to operate is becoming part of a mining company's DNA in the same way as health and safety. Consequently, mining companies are receiving increasing pressure from their shareholders to ensure that they add enduring value to the local communities in which they operate. Apart from the shareholders, a company has responsibility to suppliers, customers, employees, the government and the community, which means under the social responsibility model they must be accountable to the other stakeholders.

External and internal challenges in mining are increasing the difficulty and complexity of mines to operate. The external mining environment has a big impact on mining operations and is driving matters such as the focus on safety, responsible mining and resource management, sustainability issues, profitability and productivity.

Considering this, it is expected that the widespread implementation of Environmental, Social and Governance (ESG) related practices across the mining industry in the next ten years will increase, not just for the big players but for juniors as well. Companies, investors, governments and insurers who fail to act on ESG matters are to likely face greater risks. Demonstrating leadership in ESG will ultimately become a differentiating factor and market participants have much to gain from embracing ESG stewardship as part of their competitive advantage.

Stock Exchanges have an important role to play in advancing sustainability in the capital markets. It is for this reason that the Sustainable Stock Exchanges (SSE) initiative was founded to serve as a global platform for exploring how exchanges can enhance performance on ESG issues and encourage sustainable investment. The SSE initiative provides an effective platform for dialogue between the United Nations, stock exchanges, investors, companies and regulators in the mining industry.

The paper will explain why ESG reporting is crucial for the survival of a mining company and how the lack of transparency could be detrimental to its public image and shareholder relations. The paper will also identify bodies that regulate ESG reporting, as well as the available tools that provide guidelines on reporting ESG matters. Key ESG reporting parameters (at a global scale) will also be highlighted.

## INTRODUCTION

A Social Licence to Operate (SLO) refers to the level of acceptance or approval by local communities and stakeholders of organisations and their operations. SLO has evolved beyond the narrow focus on social and environmental issues as there is now an increasing expectation of true shared value outcomes from mining projects (Ernst and Young Global Limited, 2020).

The SLO concept has progressed fairly recently from the broader and more established notions of corporate social responsibility and social acceptability. It is based on the idea that institutions and companies need not only regulatory permission but also social permission to conduct their business. Obtaining environmental authorisations to operate from local governments and meeting regulatory requirements is no longer adequate. Any misstep can impact the ability to access capital or even result in a total loss of license.

The term social licence to operate was mentioned for the first time in 1997, at a World Bank meeting by an executive of Placer Dome (Boutilier and Thomson, 2011). The term describes an important requirement for the survival of the mining industry andrequires extractive industries to have a visible impact in a world of finite resources. To maintain the support of the host communities, companies must demonstrate transparency and responsibility, as well as encourage more open dialogue on the trade-offs between development and the environment.

In a survey of over 250 mining sector participants undertaken by Ernst and Young Global Limited, it was found that SLO was the number one top business risks facing mining and metals in the year 2019. As a result, Chief Executive Officers (CEOs) and boards have realised that the current approach to SLOis not broad enough and that the stakeholder landscape is changing, and miners need to adapt.

According to Slack (2008) the distinction between social license and consent is critical because accepting community consent as a basic operating standard sets a higher bar. If a community's actual consent is required before operations begin, companies must treat the community as more of a partner in project development, rather than as an obstacle to overcome. It also implies that a company must engage more holistically with a community, providing them access to critical information and allowing them adequate time to assess their needs and interests before making a decision about whether to accept a company's presence.

Frequently, the term SLO is understood to refer to 'the ongoing acceptance and approval of a project by local community members and other stakeholders that can affect its profitability' (Moffat and Zhang, 2014). SLO has also been embodied as a set of mutual relationships between operational stakeholders based on trust (Warhurst, 2001).

Moffat and Zang (2014) suggested that the degree to which a mining company addresses impacts on social infrastructure without engaging with comminites will affect trust in the company. In particular, the way companies engage with communities (ie contact quality and quantity) and treat community members (ie procedural fairness) will create community members' trust in a mining company, and thus their acceptance of its mining operation.

A depiction of this relationship is provided in Figure 1.

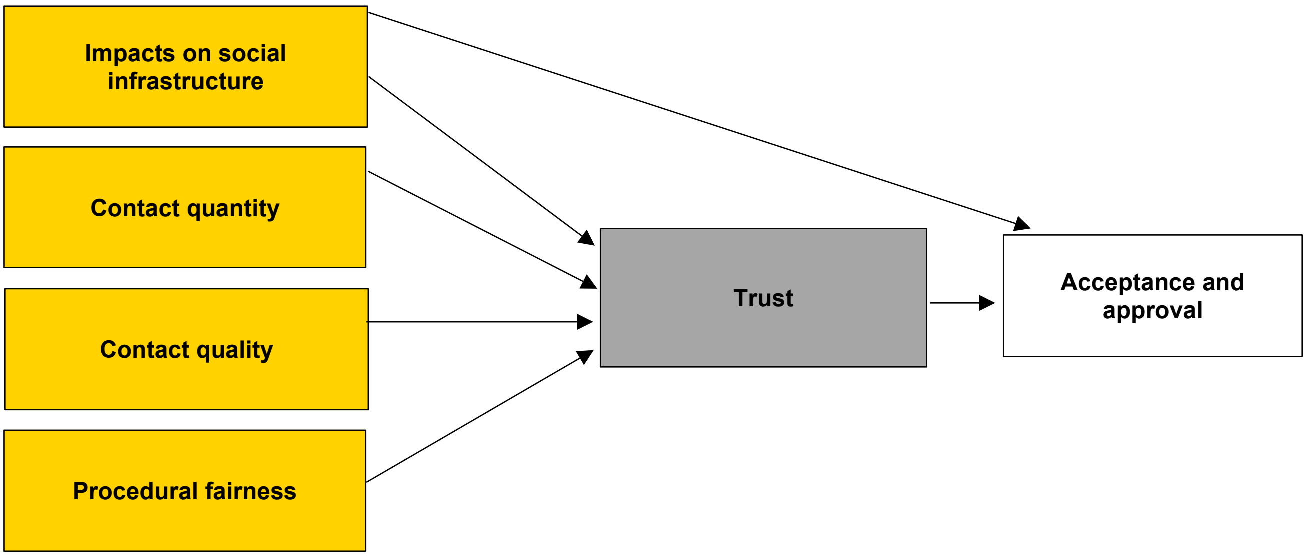


**FIG 1** – Overview of relationship between concepts (Moffat and Zhang, 2014).

## ESG REPORTING

Company disclosure and standards of reporting on ESG issues are critical for driving accurate investor information, public discourse, and regulatory guidance. In addition, transparency is critical to driving positive outcomes at both a financial and social level.

Due to the growing need of ESG reporting, voluntary environmental disclosure, responsible corporate behaviour, and financial transparency are vital factors in determining the sustainable competitiveness of companies (Barman, 2018).

Above all, financial transparency and ESG disclosure performance are now major concerns of company's stakeholders, investors, and consumers (Dhaliwal *et al*, 2014). To satisfy the needs of internal and external stakeholders, companies invest time and important resources in producing ESG reports, thus achieving transparency regarding the respective performance of a firm (Utz, 2019).

It will be necessary to keep the relevant stakeholders informed of the prgoress throughout the whole process, and to ensure that everyone is working towards the same strategic goals.

It is also importan to note that the different stakeholder groups may have differing levels of technical literacy. It will therefore be necessary to apply some of the following information sharing techniques in ansuring that the report and presentation is understood by the target market:

- In instances where the technical reports are in detail, short form reports can also be produced summarising only the key aspects of the project.
- Avoid publishing lengthy or complicated slides that could distract the public.
- Visual aids/diagrams are critical in order to keep the audience's attention and make sure the key points remain as people only remember a small percentage of what they read or hear.
- Images should integrate and clarify what is said verbally and not present additional information.
- The writings should be large and easily readable, using colour combinations to show how the initiatives relate to each other and to the larger strategic goals.

## DISCLOSURE REQUIREMENTS OF PERTINENT ESG ASPECTS

An ESG report must provide an overview of the processes and governance structures in place to support a company's commitment to doing the right business, the right way. It must also provide an update a company's progress towards enhancing positive social and environmental impacts, and mitigating negative impacts, through their core business activities.

Several frameworks exist which provide a global standard against which a companys ESG performance can be measured. Within the South African context, the South African Guideline for the Reporting of Environmental, Social and Governance Parameters within the Solid Minerals and Oil and Gas Industries (SAMESG Guideline, 2017) is recognised by the Johannesburg Stock Exchange (JSE) Reporting Index (RI) as an exchange body which regulates ESG reporting on the JSE. The SAMESG recommends that if any other ESG matters, outside of the context of Public Document dealing with Solid Minerals are to be reported on, then a separate, stand-alone, ESG specific document must be compiled along the lines described below (this relates to reporting results for Exploration, Resources and Reserves):

- Date of statement.
- General aspects, providing a description of organisational structure, systems, policies, procedures and management plans, and governance procedures in place to manage ESG issues.
- Key plans, maps and diagrams indicating the locality of sensitive receptors and sensitive areas within and around the project area.
- Legal aspects, outlining mandatory and/or voluntary standards within which the project target subscribes, including all environmental authorisations that have been issued and identified as required but not yet issued. This aspect also requires reporting on any penalties, fines and damages in excess of ZAR1 million, description of any pending administrative enforcement actions and known future liabilities.
- Environmental parameters, outlining context within which the project is located, and a description of the risks associated with any obvious environmental factors.
- External social and political parameters, providing an analysis and of external social and political context as well as the identification and mitigation of material risks.
- Internal social parameters, describing and assessing risks associated with any internal social factors and specific contextual details that could have a material effect.

- Conformance and compliance audits, providing a description of legal audits and ESG management systems conformance audits undertaken including a summary of material findings and management plans to address these findings.
- ESG liability, providing a description of closure, social obligations, rehabilitation plan, activities, remaining liability and compliance costs, including a description of mechanisms in place to address unplanned closure.
- Risk analysis, providing a description of the existence of a risk assessment process which has been undertaken to identify material ESG issues as well as a description of how the risk assessment process is integrated with the overall risk management framework.

## ESG FRAMEWORKS AND GUIDELINES

ESG investing is a 21st century phenomenon, with 48 of the top 50 economies, having some form of policy to help investors and the financial sector manage ESG risks, opportunities or outcomes. Within a global context, the volume of standards, frameworks and data requirements can appear overwhelming even for more experienced issuers. It can be a challenge to identify the indicators and standards that are most relevant to their investors.

There is a trend towards more mandatory ESG disclosure with emphasis on materiality of the reported information to meet growing investor demand for ESG information of key relevance to companies' operations. Some of the key regulations across the geographies are:

- European Union: In Europe, significant progress has been made to develop a comprehensive ESG legal framework. The Sustainable Finance Disclosure Regulation (SFDR) came into effect on 10 March 2021 and was created to set common rules for the Europen Union on: (i) how financial product manufacturers and financial advisers should inform end-investors about sustainability risks, (ii) how the impact of investments on the environment and society should be disclosed, and (iii) how financial products that are marketed as sustainability-related meet that ambition.
- Germany: Index provider S&P Dow Jones Indices (S&P DJI) announced on 5 May 2021 that it has been selected by the Federal Government of Germany to develop an ESG index which will serve as a performance benchmark for four of the government's Federal Special Pension Funds. The ESG index is set to incorporate the minimum standards for EU Climate Transition Benchmarks as described in Regulation (EU) 2019/2089 (known as the Low Carbon Benchmarks Regulation) and aligns with the landmark Paris Agreement.
- China: In 2019, the Hong Kong Stock Exchange (HKEX) introduced the mandatory requirement for all listed companies to issue a statement on the board's consideration of ESG risks, as well as how it determines what ESG matters are material to the business. As the stocks of nine of China's biggest mining companies are either traded on the HKEX or the Shanghai Stock Exchange, they need to apply ESG disclosure requirements to meet the needs of investors.
- South Africa: In 2009 the JSE introduced the mandatory integrated reporting. Companies listed on the JSE are required to implement integrated sustainability performance and integrated reporting.

In 2015 the SSE launched its database tracking which exchanges have published ESG guidance, which has now been updated with an analysis of the reporting instruments most frequently cited by the guidance documents, including instruments from: Global Reporting Initiative (GRI), International Integrated Reporting Council (IIRC), Sustainability Accounting Standards Board (SASB), CDP Worldwide, Task Force on Climate-related Financial Disclosures (TCFD), and Climate Disclosure Standards Board (CDSB). It shows that an overwhelming majority of guidance documents reference the GRI, followed by the IIRC and SASB instruments, which are each referenced by around three-quarters of guidance documents. Climate-specific reporting instruments such as TCFD and Climate Disclosure Standards Board (CDSB) are referenced by just under half of the guidance documents (Sustainable Stock Exchanges Initiative, 2020b).

**TABLE 1**

List of ESG frameworks and guidelines supported by the SSE (Sustainable Stock Exchanges Initiative, 2020a).

| ESG Framework/Guideline | Description |
|---|---|
| **General and cross-cutting** | |
| Global Reporting Initiative | The GRI is a multi-stakeholder non-profit organisation that develops and publishes guidelines for reporting on economic, environmental and social performance (sustainability performance). |
| International Integrated Reporting Council | The IIRC is an international cross-section of leaders from the corporate, investment, accounting, securities, regulatory, academic and standard-setting sectors as well as civil society. Integrated Reporting demonstrates the linkages between an organisation's strategy, governance and financial performance and the social, environmental and economic context within which it operates. |
| Sustainability Accounting Standards Board | SASB standards are used by companies around the world in a variety of disclosure channels, including their annual reports, financial filings, company websites, sustainability reports, and more. Investors, lenders, insurance underwriters, and other providers of financial capital are increasingly attuned to the impact of ESG factors on the financial performance of companies, driving the need for standardised reporting of ESG data. |
| CDP Worldwide | The CDP is an international non-profit organisation based in the United Kingdom, Germany and the United States of America that helps companies and cities disclose their environmental impact. It aims to make environmental reporting and risk management a business norm, driving disclosure, insight, and action towards a sustainable economy. The CDP challenges the world's largest companies to measure and report their carbon emissions. |
| Task Force on Climate-related Financial Disclosures | The TCFD was created by the Financial Stability Board (FSB) to develop consistent climate-related financial risk disclosures for use by companies, banks, and investors in providing information to stakeholders. |
| Climate Disclosure Standards Board | CDSB is a non-profit organisation working to provide material information for investors and financial markets through the integration of climate change-related information into mainstream financial reporting. CDSB operates on the premise that investors and financial institutions can make better and informed decisions if companies are open, transparent and analyse the risks and opportunities associated with climate change-related information. |
| **Other** | |
| United Nations Global Impact (UNGI) | The UNGI is both a policy platform and a practical framework for companies that are committed to sustainability and responsible business practices. As a multi-stakeholder leadership initiative, it seeks to align business operations and strategies with ten universally accepted principles in the areas |

| ESG Framework/Guideline | Description |
|---|---|
| | of human rights, labour, environment and anti-corruption. It is the world's largest voluntary corporate responsibility initiative. |
| UN's Voluntary Principals on Security and Humans Rights | The only human rights guidelines designed specifically for extractive sector companies. They are a set of principals designed to guide companies in maintaining the safety and security of their operations within an operating framework that encourages respect for human rights. |

Figure 2 provides an overview of reporting instruments most frequently referenced in stock exchange guidance documents (Sustainable Stock Exchanges Initiative, 2020a).

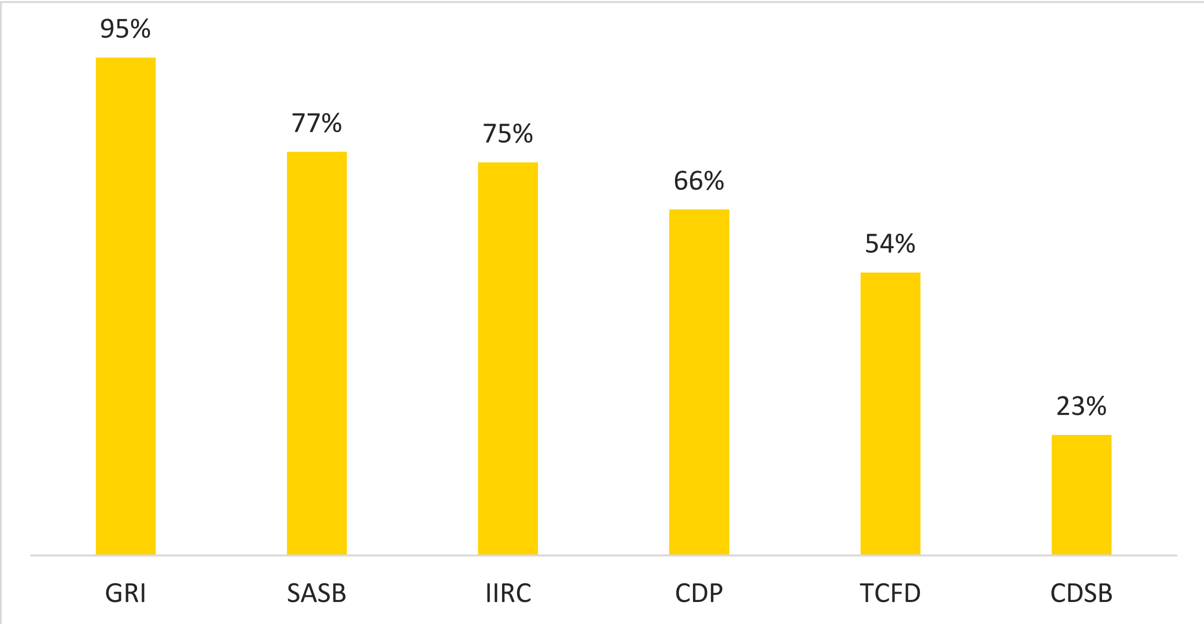


**FIG 2** – Reporting instruments referenced in stock exchange guidance documents (Sustainable Stock Exchanges Initiative, 2020a). NB: Global Reporting Initiative (GRI), Sustainability Accounting Standards Board (SASB), International Integrated Reporting Council (IIRC), CDP Worldwide, Task Force on Climate-related Financial Disclosures (TCFD), and Climate Disclosure Standards Board (CDSB).

Table 2 provides some of the typical issues which are disclosed in an ESG report.

**TABLE 2**

Disclosure of ESG aspects and other reporting requirements.

<table>
<tr><th>Pertinent ESG Aspects</th><th>Legal and Contingent Liabilities</th></tr>
<tr><td rowspan="3">Environmental Aspects<ul><li>Compliance with environmental legislation and licence conditions.</li><li>Use of processes, technologies or chemicals that have unacceptable impacts on the environment.</li></ul>Social Aspects<ul><li>Pending land/compensation claims</li><li>Relationships with stakeholders</li><li>Need for involuntary resettlement</li><li>Relationship with employees</li></ul>Governance Aspects<ul><li>Ethics</li><li>Executive pay</li><li>Board diversity and structure</li><li>Tax strategy</li><li>Political lobbying</li></ul></td><td>The market value of the contingent liabilities comprising:<ul><li>Environmental liabilities</li><li>Rehabilitation obligation</li><li>All statutory liability (including any historic liability) for the rehabilitation and closure of the mine, including the social closure obligations.</li><li>All of the other obligations in terms of the mining rights, the applicant's liabilities to the employees.</li></ul></td></tr>
<tr><td>Miscellaneous Aspects</td></tr>
<tr><td><ul><li>Country risk</li><li>Political risk</li><li>Legal risk</li><li>Reputational risk</li></ul></td></tr>
</table>

## BENEFITS OF ESG REPORTING

Investors and shareholders are aware of the long-term negative impact of ESG liability on potential growth and they are expecting more transparency from corporations. High standards of corporate ESG reporting are becoming ever more important for accurate investor information, public discourse, and regulatory guidance.

ESG reporting can bring a wide range of benefits to listed issuers' businesses – some of these benefits are:

- Improve their corporate governance, and in particular, strengthen their risk management by prompting them to assess ESG-related risks to their businesses, thus preparing them to better manage these risks.
- Save costs by prompting them to review, identify and address any inefficiencies in their consumption of resources such as energy and water.
- Recognise and capitalise on new business opportunities, which can in turn drive innovation (egdevelopment of greener, more resource-efficient products).
- Attract investors that incorporate ESG criteria into their decision-making.
- Enhance their share valuation and secure financing from lenders more easily, thus lowering their cost of capital.

## CASE STUDIES

The case studies provided below highlight the effects of lack of transparency on matters relating to ESG reporting. These are an indication that companies and investors who fail to act on ESG matters are likely to face greater risks of losing their social license to operate.

### Chile rules against BHP over Cerro Colorado's environmental permit

In an article published by Reuters on 14 January 2021, a complaint was lodged by the San Isidro de Quipisca Indigenous Agricultural Association over the granting of environmental approval in 2016 for

BHP Group's Cerro Colorado copper mine. The concerns were over the operation's water use, whereby a routine evaluation found that the mine's environmental project did not consider warnings by the local community members that Cerro Colorado was excessively using water and almost dried out surrounding wetlands.

The lawyers representing the Indigenous Group are recommending that BHP should cease operations, while the necessary environmental review is completed. BHP, however, is contesting that based on the same dispute by the country's Environmental Court, operations could continue while the review is being undertaken.

### Evicted Kenyans demand compensation after mining firm exits

In another article published by Reuters on 11 October 2018, Kenya Fluorspar Company (KFC) turned two acres of Tumo's land, which he inherited from his father into an opencast mine but did not pay compensation, stating that the government had acquired the land in the 1970s, decades before the firms involvement.

The firm's 21-year mining lease expired in March 2018 and the abandoned pits have filled with rain, marked with danger signs to deter locals from entering the area.

The government of Kenya will compensate thousands of Kerio Valley evictees, most of whom lost their land soon after 1971 when the state-owned fluorspar mining company was set-up.

## CONCLUSION

Companies that do business in a way which provides opportunities, generates meaningful employment, respects human rights and protects the environment can better manage risks, realise opportunities and help build more resilient communities that create and foster sustainable livelihoods. ESG disclosure and reporting forms a crucial part in the improvement on corporate ESG transparency and performance and if not adequately addressed, companies are likely to face greater risks of losing their social license to operate. Tools such as the samesg regulates esg reporting on the JSE and provides guidelines on reporting on ESG matters outside the context of public documents.

With mining in Africa on the rise and with more and more interest from international companies and investors, there is a need to further identify and understand business practices which can bring shared benefits for companies and for communities.

In order to ensure long-term sustainable solutions in community development and maintain healthy shareholder relations, one of the ways is to align expectations and needs from both the community and the mining company. It is also of highest importance to always remember that the local community is not a unified group of people, but rather a grouping of different individuals with their own characteristics and their own interests, which are directly or indirectly affected by mining activities which in turn may lead to environmental liabilities and reputational damage.

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