WA School of Mines: Minerals, Energy and Chemical Engineering

Optimization of Underground Development Advance – A Pragmatic Approach to a Multivariate Problem

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DECLARATION

To the best of my knowledge and belief this thesis contains no material previously published by any other person except where due acknowledgement has been made.

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ABSTRACT

In the mining industry, often decisions need to be made rapidly based upon the experience and general knowledge of the management team making the decision. Whilst it is reasonable to make a quick decision due to operational requirements and constraints, when it comes to solving problems of a grander scale, more considered approach is appropriate. A holistic, pragmatic approach considers data driven decision making and empirical data but also leans into general knowledge and experience when the data is insufficient to draw statistically significant conclusions.

A long standing challenge at Agnew Gold mine in Western Australia was the tendency of the development blasts to result in sub-optimal advance rates. A key difficulty in determining the root cause of the failures included the wide array of available design parameters required to be considered by the operators in determining the final blast design – ground type, bulk explosive, bit and steel sizes, blast pattern and sequencing.

Development blast data was collected from Agnew Gold Mine in Western Australia during a 6 month period for the purpose of optimising the drill and blast practices and design based on data, observations and blast theory. Quantitative, statistical and empirical analysis was conducted using the data and where the data was inconclusive, established blast theory was leveraged to draw inferences from the data to aid in driving a design decision.

A standardized design was generated based on the findings and rolled out across all crews. The implementation of the optimised design was able to deliver an average cut length 0.3m greater than the operational average while also reducing the standard deviation by 25%, and simultaneously resulted in a cost reduction in consumables of approximately 15% per metre advance.

This study presents a holistic, pragmatic approach to drill and blast optimisation, utilising a hierarchy of considerations for optimised blast design and demonstrates its effectiveness through its successful application in highly laminated geology at an underground gold mine in Western Australia.

Keywords: Drill and Blast, Optimization, Development Advance, Development Cuts, Underground Mining, Pulling Cuts

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1 INTRODUCTION

Development advance is one of the key performance metrics of any underground mine - it directly impacts the mine schedule and is the most significant contributor to the cost of an underground mining operation. Optimizing development drill and blast practices can significantly reduce associated costs, improve the rate of advance or “Pull” of each cut, and reduce re-work.

A long-standing operational challenge at Agnew Gold Mine (“Agnew”) in Western Australia has been the inability to effectively blast full cut lengths during lateral development, particularly when developing through heavily laminated geology. Often target rock is left behind as butt which requires mechanical scaling and re-work after the initial blast.

1.1 Problem Statement

A major challenge at Agnew in determining causes of failure was the lack of standardization of development blast designs; jumbo operators made the plan at the face and drilled according to their personal experience, and the charge up operators were encouraged to charge the holes with whatever explosive was available at the time (which was also dependent on machine maintenance schedules). This led to a large degree of variation between cut design and performance. The different choices operators faced in their designs included long drill steels vs short drill steels; Hex Steels vs Round steels; Small diameter vs Large diameter bits; 5 row patterns vs 7 row patterns (and everything in between); Burn Cut / Reamer formation, Small cast booster vs Large cast booster vs Packaged emulsion; Blasthole sequencing; and, ANFO vs Emulsion.

Previous attempts at solving the problem were outlined in internal company reports, external reports conducted by suppliers that was clearly impacted by product bias, and an unlimited supply of operator opinions that all offered vastly different conclusions on the root cause. The variability between blast design inputs made it difficult to repeat any of the findings with any degree of confidence, which alluded to the multi-variate nature inherent in drill and blast operations.

Major areas to consider in any drill and blast design include geology, pattern design, drilling practices and explosive selection. Freezing all inputs and investigating each parameter in isolation is a time consuming and costly exercise, and often there are too many variables to rely solely on quantitative multi-variate analysis. Due to the large array of inputs available at Agnew as well as the time-sensitive nature of underground mining; an alternative method of optimization is required – a pragmatic approach to optimization.

1.2 Research Aim

This research presents a pragmatic approach in solving the challenges at Agnew and proposes a hierarchy of considerations for optimized blast design.

A pragmatic approach to optimization does not rely purely on data analysis, but also draws on operational observations and the application of blast theory to help extract insights from data that may not be readily extractable with pure quantitative analysis.

1.3 Hierarchy of Considerations for Optimized Blast Design

To ensure every parameter in the blast design is considered in its entirety and specifically for its purpose, a hierarchy of considerations for optimized blast design is proposed.

The hierarchy of considerations outlined in Table 1-1 is a framework for iterative reflection at all stages of the optimization process (Figure 1-1) – during data collection, analysis and improvement.

1. Understand the Problem	Consider all parameters in a blast design with respect to their role in the blast fracture mechanism. What are you trying to achieve and how does it relate to rock fracture? Identify metrics for success.
2. Consider Operational Evidence	Which parameter selections are involved in the best performing blasts? Is there enough evidence to suggest they could be a critical factor to success?
3. Consider Established Blast Theory	Where there is not enough evidence within the data set to drive a design decision, the decision should be made upon established blasting theory.
4. Consider Cost	Cost analysis is conducted to ensure the practicality of the design and benefit to the business.

Table 1-1 - Hierarchy of Considerations for Optimised Blast Design

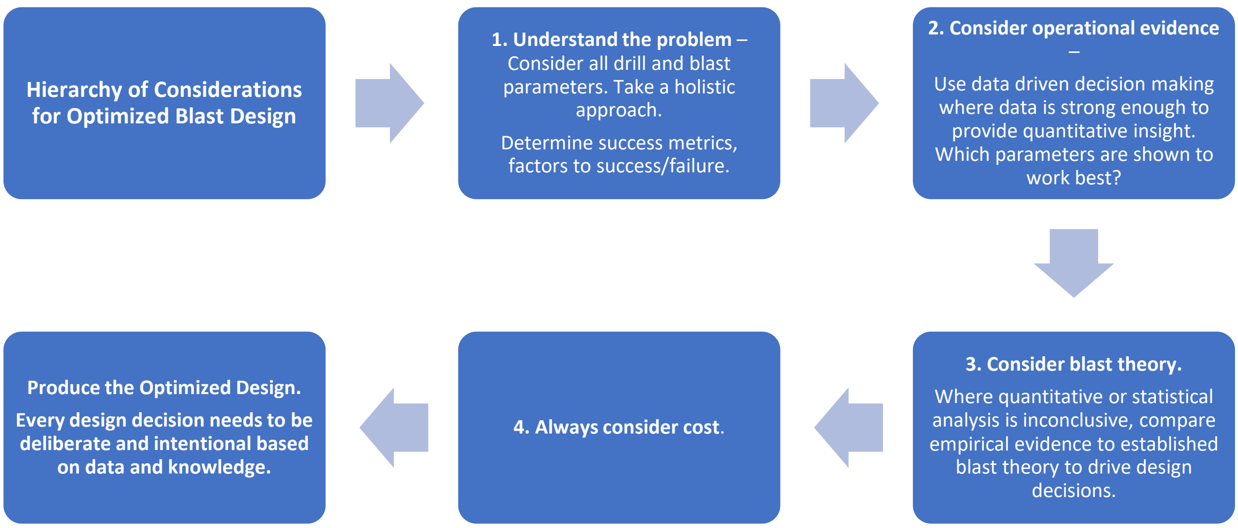


Figure 1-1 - Hierarchy of Considerations for Optimised Blast Design Process Flow

2 MINE SITE INFORMATION

Agnew underground gold mine, owned by Goldfields, is 375km north of Western Australian mining town Kalgoorlie, approximately 1000km north-east of Perth, and produces over 250,000 ounces of gold per annum. The mining operations at Agnew consist of the consolidated Waroonga and New Holland underground complex, accessible from their respective open pits. Waroonga is operated by contract miner Barminco, while New Holland is an owner-operated operation (Goldfields, 2021).

The Waroonga complex contains the Main, Kim and Rajah lodes found near the contact with Scotty Creek Sandstone sediments and mine conglomerate sequence, characterized by variably deformed laminated quartz veins and breccia (Mindat, 2021). Woolley (2015) measures the average joint spacing of the sandstone domains as 0.1m-0.5m.

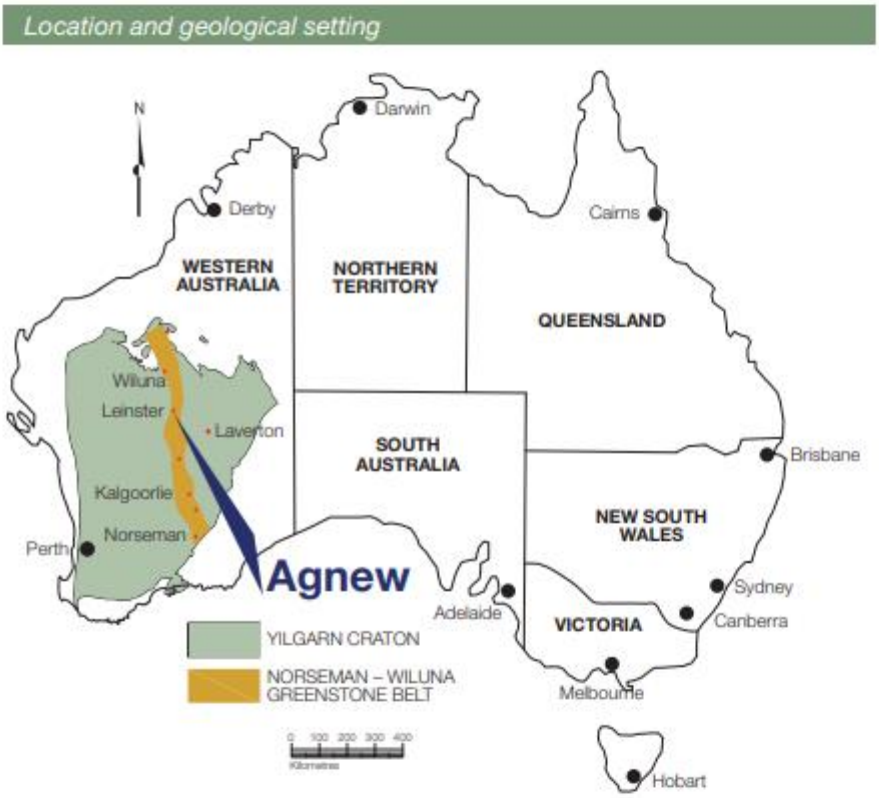


Figure 2-1 Location and geological setting (Goldfields, 2012)

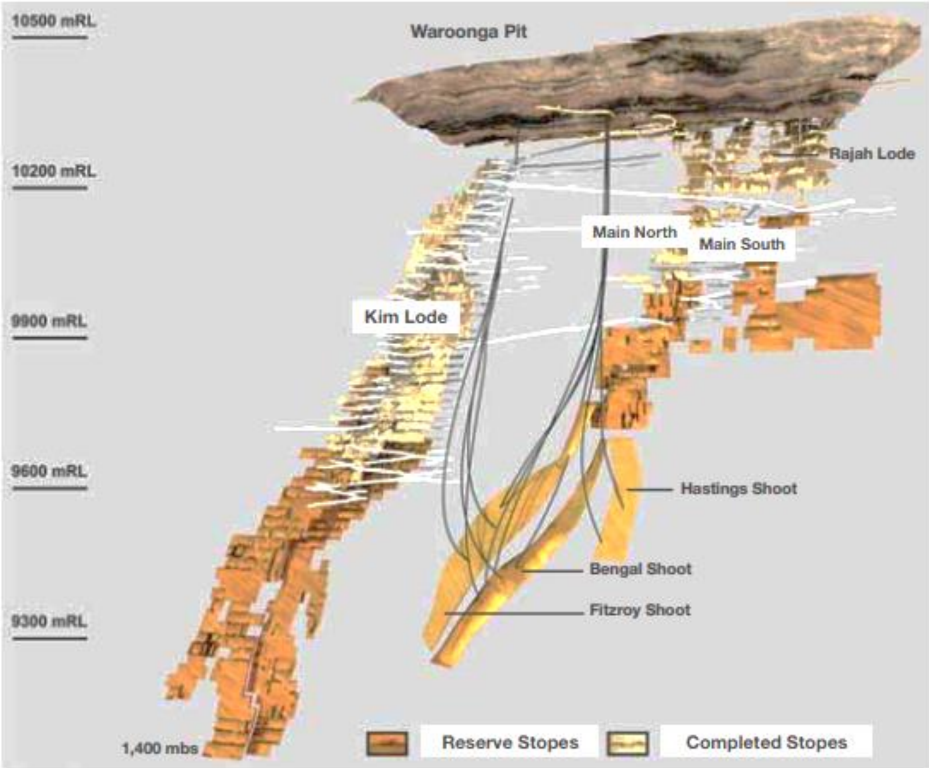


Figure 2-2 – Mine Model of Agnew Waroonga ore body (Goldfields, 2012)

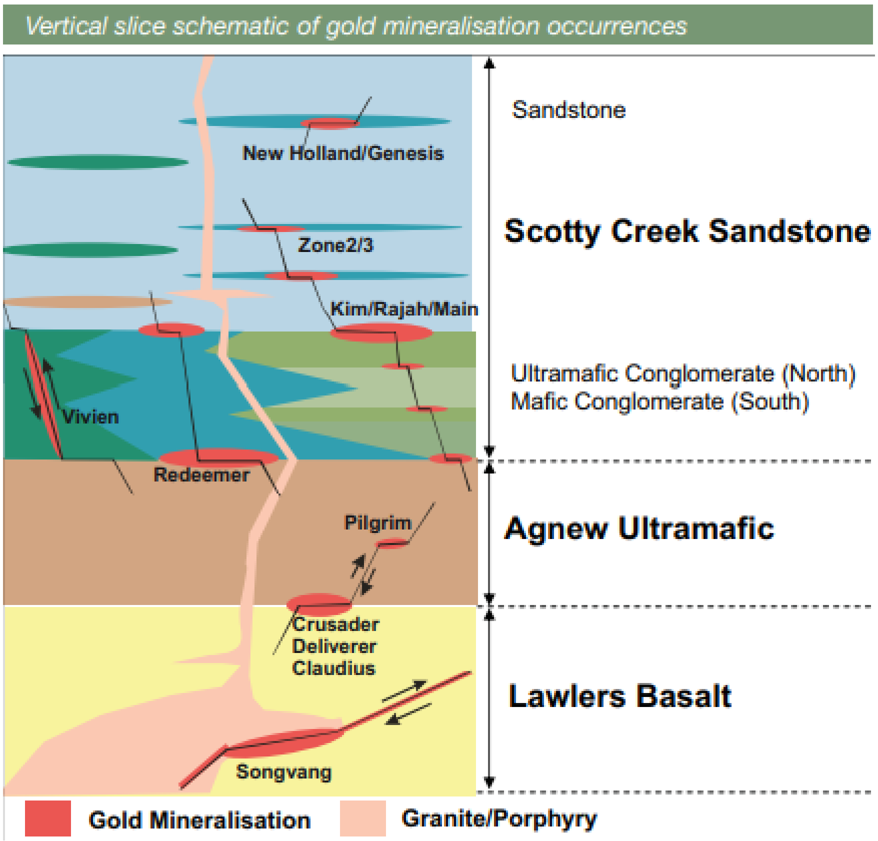


Figure 2-3 Vertical slice schematic of gold mineralisation occurrences (Goldfields, 2012)

3 BACKGROUND & LITERATURE REVIEW

3.1 A BRIEF HISTORY OF MINING AND MINERAL EXTRACTION

The extraction of minerals for use in society dates back to at least the Bronze age (Circa 3000BC) when civilisations capable of melting terrestrial ores began smelting copper and tin to fashion items such as tools and weaponry. As history progressed and records began to be kept, descriptions of mining techniques can be found in one of the largest known historical pieces of literature from the ancient Roman time period – Naturalis Historiae by Pliny the Elder. In the record, Pliny describes the use of fire setting and vinegar-quenching to break rock (Pliny the Elder, 1984). Later works such as Georgius Agricola's De Re Metallica first published in 1556 also document rock breakage methods that includes gads and mauls being hammered into rock cracks to expand them (Agricola, 2011). Rock breakage mechanisms did not change much until gunpowder was introduced circa 1670 (Darling, 2011, pp. 3). The introduction of gunpowder provided means to easily break rock however the process remained dangerous as the gunpowder was poured into blast holes and ignited by a spark.

With the introduction of Safety fuse by William Bickford in 1831, dynamite in 1867 by Alfred Nobel, and compressed-air-powered drills in the 1860s – the safety and productivity of mining increased significantly, with ongoing refinements in drilling and blasting, and the development of mechanized mining techniques bringing us to the present day (Darling, 2011). Before further discussing the refinements and optimization of drill and blast techniques in an underground development context it is first important to understand the fundamental science of drilling and of explosives and blasting.

3.2 DRILLING

In the age of mechanised mining (and more specifically "drill and blast"), drilling involves the breakage and subsequent removal of rock, usually with high pressure air or flushing water, to produce boreholes with which to plant explosives. There are many types of rock drilling methods available including directional drilling, drag-bit drilling, rotary drilling, and percussive drilling. The industry standard for development in hard rock underground mining is the rotary-percussive drilling method utilizing top-hammer Jumbo drills.

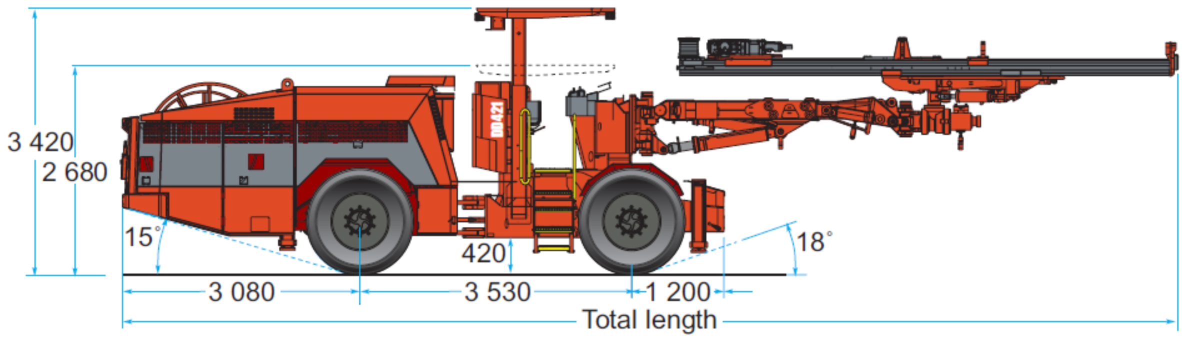


Figure 3-1 – The Sandvik DD421 Development Drill Rig, also known as a Jumbo drill, uses a hydraulic top-hammer (Sandvik, 2021).

Rotary-Percussion drills use a mechanical hammer (typically driven by hydraulic piston) to bring a drill bit up and down in a continuous cycle, whilst simultaneously rotating as the drill bit strikes the rock. The percussive energy of the hammer is transferred to the shank which is coupled to a drill steel, and the energy is then transferred from the drill steel to the drill bit. The hammer simultaneously rotates the shank, which in turn rotates the steel and the bit.

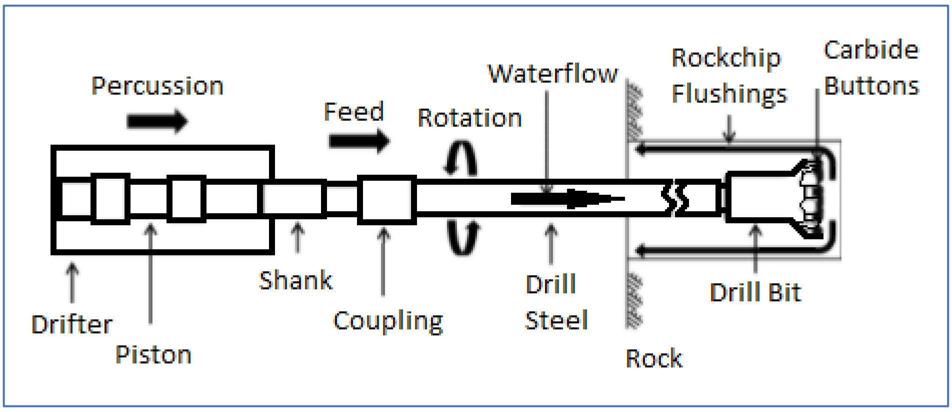


Figure 3-2 - Rotary-Percussive Drilling Energy Chain (Kim & Kim et al 2020).

As the drill bit strikes the rock, the button attached to the bit transmits a sudden burst of force, causing the button's sharpness to embed in the rock. This results in the rock near the embedded button being crushed by the shock wave and compressive pressure generated instantly. As the shock wave moves through the rock, it causes shear and tension cracks to form in the surrounding rock. Some of the cracks can lead to rock chips breaking off, particularly close to the free surface. This phenomenon is heightened when the spacing between the impacted buttons and the rotation speed is optimised.

Different rock characteristics such as its geophysical properties (e.g. rock strength, hardness and brittleness) and geological properties (e.g. structures and jointing) impact the effectiveness of a rock drill, and thus intact rock properties bares the most significance on bit selection. To assist with this, manufacturers of drill bits offer various bit and carbide configurations, for varying ground types (Appendix A).

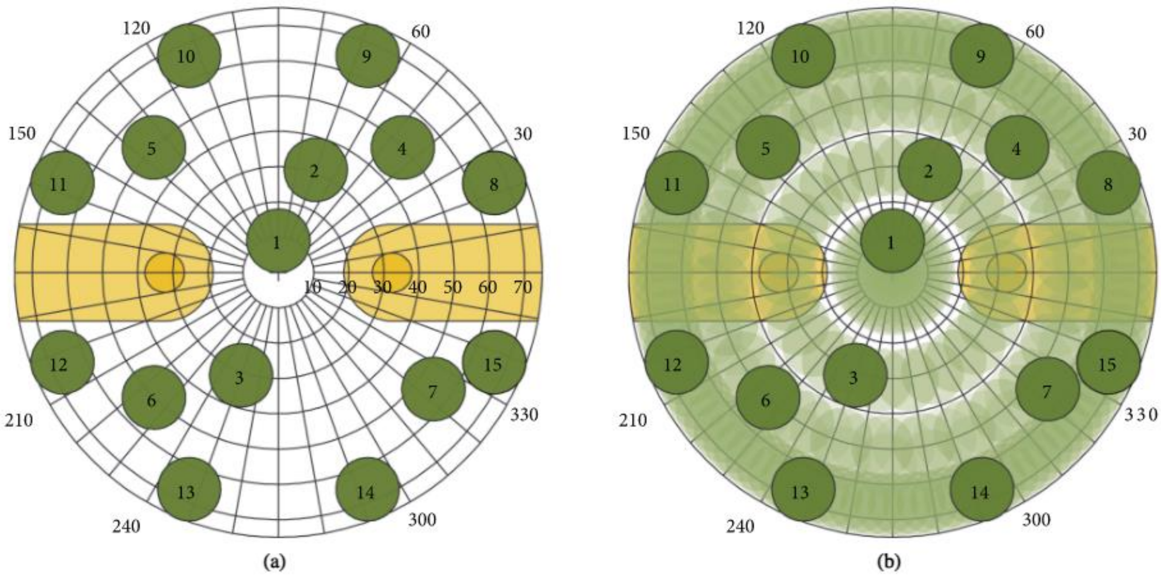


Figure 3-3 - (a) Drill Bit Button Configuration, and (b) The impacts shown after each rotation between percussive strikes (Kim & Kim et al, 2021).

Drill steel selection can impact both the depth of the borehole (steel length) and accuracy/hole deviation (cross sectional profile of the steel). A hex steel is typically cheaper, lighter, and easier to handle than a round steel. A round steel is more rigid and is used to reduce the risk of hole deviation during drilling. The increased rigidity is due to the round profile as well as the additional material required to achieve the profile. Due to the additional material however, it is typically more expensive and heavier to handle. The increased rigidity also assists with efficient flushing of water and cuttings from the borehole (Ace Drilling, 2021).

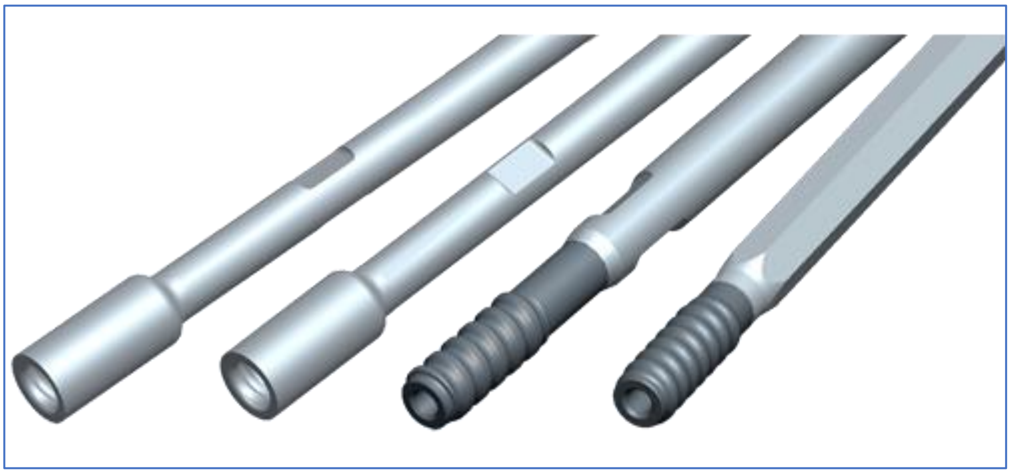


Figure 3-4 - Drill steels. 2 x Speed rods (round), 1 x Extension rod (Round), 1 x Extension Rod (Hexagonal) (Ace Drilling, 2021).

Once the drill rig is aligned to the blast hole, the operator can adjust their percussion (on a Sandvik DD421 Drill Rig they may choose between normal percussion or high percussion), rotation pressure and feed pressure. Having the feed pressure up too high can cause the drill steel to bend inside the hole, tilting the drill bit and leading to deviation (Boart Longyear, 2021-A). Gravity can also cause the drill steel to sag in the middle, causing an upwards deviation of the drill bit. (Boart Longyear, 2021-A) – which is why deviation is reduced with more rigid steels. Having the rotation pressure up too high can bind up the components and make them hard to separate. High percussion is useful for breaking rock with a high compressive strength – but using it in rock that does not require the additional breaking force can lead to early wear and failure of the component parts and the drifter itself.

The carbide buttons on the drill bit can also become worn, reducing the penetration rate of the drill string. If the penetration rate drops while the feed pressure is wound too high, this can lead again to the bowing of the drill steel and deviation of the borehole. When buttons are worn fewer rock chips are generated, and a greater proportion of the energy is reflected back through the drill string which reduces the life of all of the drilling components and becomes a major contributor to deviation. (Boart Longyear, 2021-B).

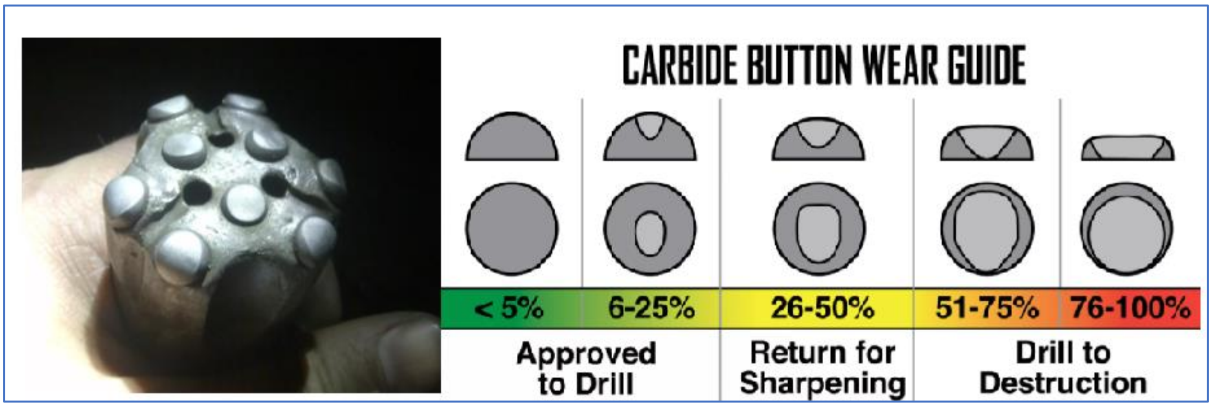


Figure 3-5 - Drilling consumables supplier Boart Longyear recommends proper maintenance of bits to ensure effective drilling (Boart Longyear, 2021-B).

3.3 THE MECHANICS OF BLAST INDUCED ROCK BREAKAGE

The interaction between explosives and in-situ rock is a violent process that involves the rapid release of chemical and mechanical energy measured in the range of Gigapascals (Gpa). Detonation of an explosive charge inside a blast hole creates a stress wave that propagates through the surrounding rock causing fractures proportional to the magnitude of the wave. Initially the pulse is a compressive wave that becomes reflected off freefaces or discontinuities in the rock. Once the wave is reflected it becomes a tensile wave, which due to the low tensile strength of rock, provides a plausible mechanism for the development of slabs, spalls or damage within the rock medium. (Brady & Brown, 2004)

3.3.1 Wave Propagation

Waves travel in three dimensions, and the propagation of the wave induces transient displacements ux(t), uy(t), uz(t) in all three directions as illustrated in Figure 3-6, where t is time. Importantly, the displacement within a plane wave along plane yz is considered to be identical and independent of (y,z). The displacement of a plane wave can thus be expressed as:

$$u_x = u_x(x),\ u_y = u_y(x),\ u_z = u_z(x) \tag{1}$$

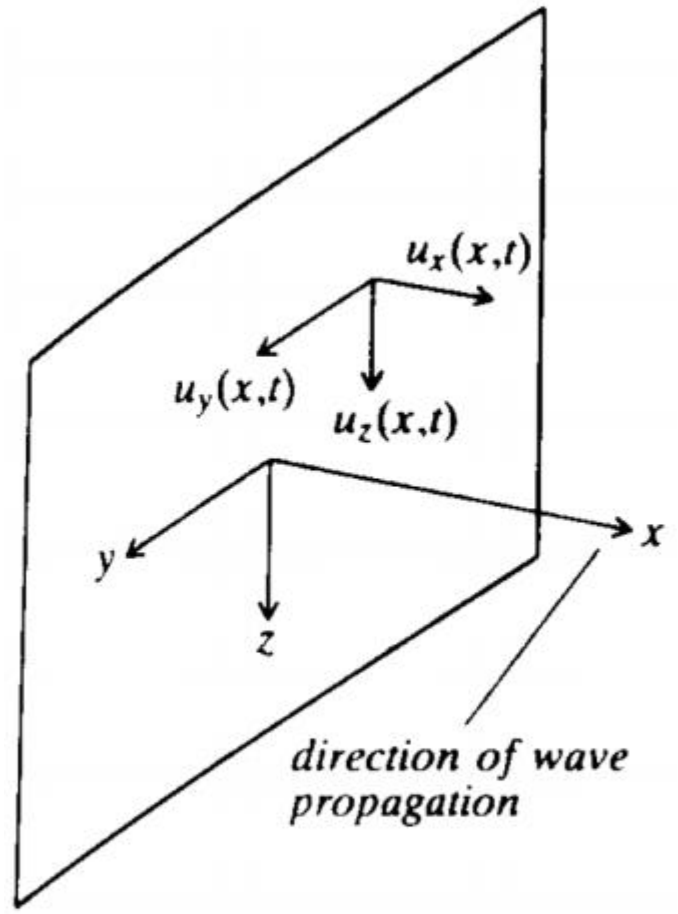


Figure 3-6 - Direction of blast wave propagation in directions x, y, z.

This explosive-rock interaction can be modelled as either a spherical charge or a cylindrical charge and the key difference between these two models is in the approach to generating the divergent dynamic stress waves (Dong, 2019).

Sharpe (1942) first modelled the elastic wave generated by an explosive as a spherical charge in a continuous, homogenous medium. This model ignores any discontinuities that are likely to be found within in-situ rock, and thus any resulting internal reflection or refraction of the wave. In doing so the model provides a simple basis for understanding the fundamental behaviour of the dynamic stress wave. Sharpe (1942) modelled the wave pressure in the rock as the same at any location within the rock at a given time and is proportional to the peak pressure exerted on the rock wall, which decays with time. This is expressed in Equation (2).

$$P = P_0 e^{-\alpha T} \tag{2}$$

Where P is wave pressure, P_0 is peak wall pressure, and α is decay constant.

While modelling a spherical charge gives a basic understanding of the propagation of a blast wave, in practice most charges are cylindrical due to the nature of the bore hole. While the spherical charge acts as a point charge where the blast wave propagates from a single point, the cylindrical charge detonates at one end and the blast wave propagates through the charge column over time. This behaviour is more complex to model however Starfield & Pugliese (1968) formulated a solution that simplified the problem by modelling a cylindrical charge as a series of discretised charge segments which each represent a singular spherical charge. This model correctly describes the divergence of the wave generated by the charge over a conical front, as illustrated in Figure 3-7. Results obtained in field experiments of cylindrical charges were in general agreement with the predictions from the simple model.

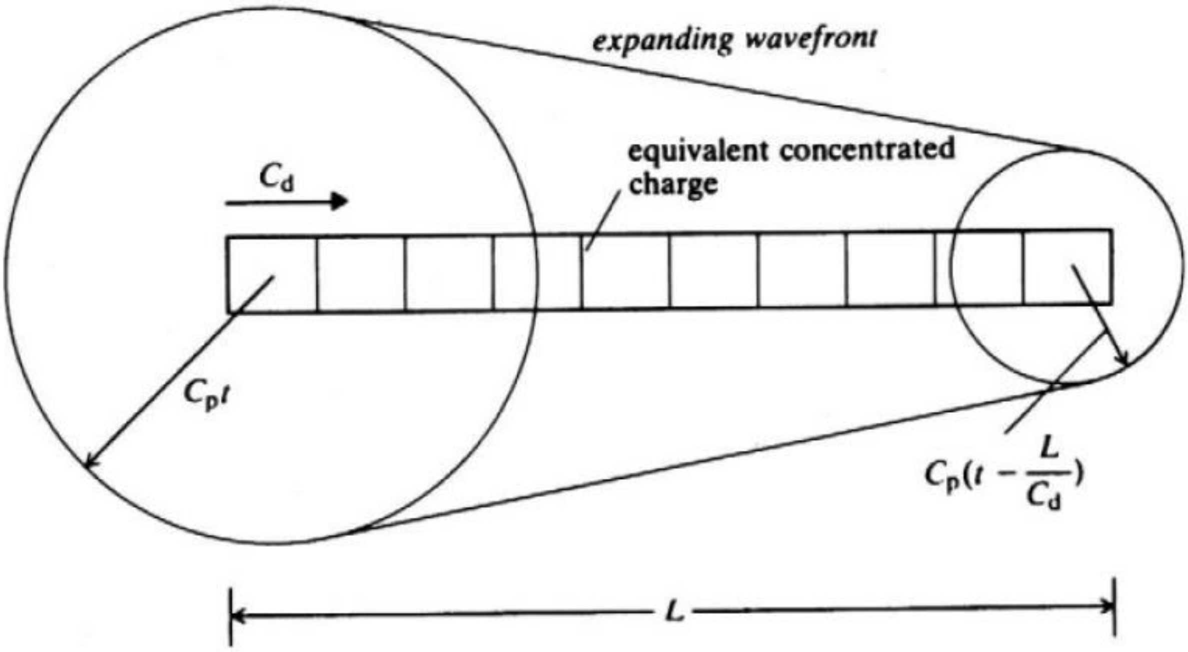


Figure 3-7 - Finite difference model of detonation and wave generation by a long, cylindrical charge. (Brady & Brown, 2004)

3.3.2 Explosive detonation in a cylindrical charge

When the bulk explosive is initiated, the shock wave produced by the chemical reaction has sufficient energy to cause the neighbouring molecules to undergo the same chemical reaction, leading to a self-sustaining chemical reaction. The shock wave produced by the reaction is a space of negligible thickness bounded by two infinite planes – on one side of the wave is the unreacted explosive and on the other side is the exploded gases as illustrated in Figure 3-8 (P.D. Sharma, 2012). The three distinct zones are thus created. The undisturbed zone is where the explosive has not yet been affected by external forces. The reaction zone is the shockwave initiates a chemical reaction (through a rapid increase in pressure) that is completed at the C-J (Chapman-Jouquet) plane. At the C-J plane, there is a self-sustaining shockwave released that maintains the pressure & temperature required to initiate the explosive in front of it. The decomposition zone is where the reaction has completed, and the products of the chemical reaction rapidly expand outwards as high pressure gasses.

The speed at which the detonation wave propagates is referred to as the Velocity of Detonation ("VOD") which can be thought of as a proxy for the energy of the shockwave. The detonation pressure that exists at the C-J plane is a function of the VOD of the explosive (rate of energy release) and the explosive product's density (latent energy). Cooper (1937) derived an equation to estimate the pressure at the C-J plane for any explosive of any density within 5% of experimental values. The equation is expressed in Equation (3).

$$P_{cj} = \rho D^2(1 - 0.7125\rho^{0.04}) \tag{3}$$

Where P_{CJ} is the pressure at the C-J plane (Mpa), ρ is the density (g/cc) of the explosive, and D is the velocity of detonation (m/s).

As the density and velocity of detonation increase, so too does the pressure at the C-J plane. For the commonly used Ammonium Nitrate Fuel Oil ("ANFO") explosive, which has a blow loaded density of approximately 0.95g/cc with corresponding VOD estimate of 4500m/s (Orica, 2015A), the pressures produced at the C-J plane would be approximately 5GPa. For comparison, the main rock type at Agnew Gold Mine to which this study pertains is Scotty Creek Sandstone, which has Unconfined Compressive Strength (UCS) measured between 135-160 MPa. The pressures generated by the reaction are more than sufficient to cause damage to the surrounding rock mass.

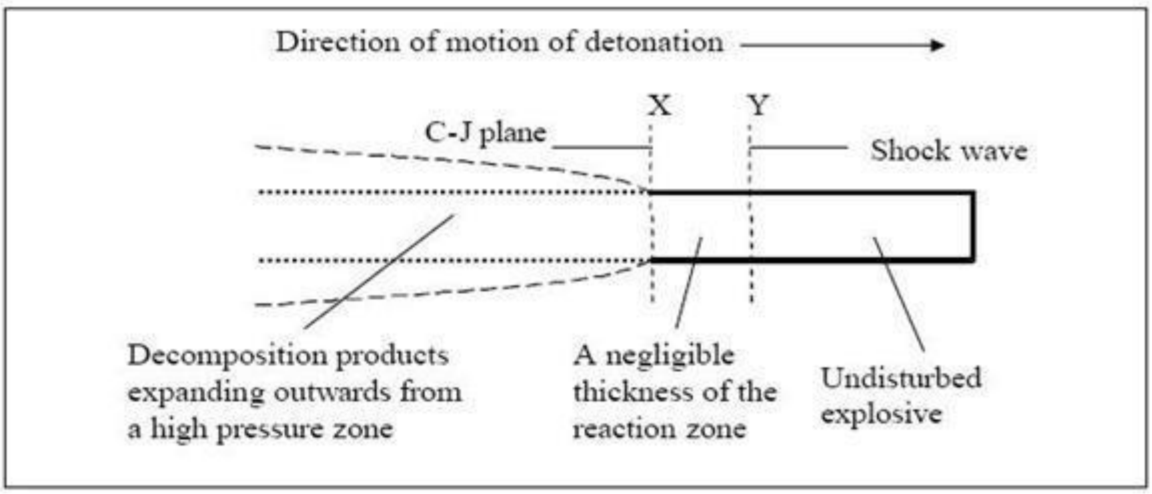


Figure 3-8 - C-J Plane in a detonation within a cylindrical charge. (P.D. Sharma, 2012)

3.3.3 Blast-induced mechanism of rock breakage

The shockwave generated from the chemical reaction not only propagates through the explosive column, but also through the surrounding rock mass as a compression wave. An & Liu et al. (2017) summarizes earlier works in rock breakage and illustrates the mechanism in Figure 3-9 - Blast induced mechanism of rock breakage. (An & Liu et al., 2017).

As discussed earlier in this chapter, the shockwave detonation pressure is orders of magnitude greater than the UCS of the surrounding rock. The shockwave presses the rock against itself as the wave travels through it. Since the initial detonation pressure is greater than the compressive strength of the rock, the initial mechanism of rock breakage is via crushing, and a crush zone immediately surrounding the blast hole is formed immediately after the initial detonation.

As the shockwave expands outwards and attenuates, the proportion of damage due to compressive stresses decreases while the proportion of damage due to tensile stresses increases in what is referred to as the transitional, non-linear zone.

The third zone, known as the elastic or fragment formation zone, appears as the explosive wave continues to attenuate to the point where the compressive stresses rarely meet the UCS of the rock. As such, no more damage is caused by compressive stresses, though the tangential stresses are still large enough to cause radial fractures due to rock's tensile strength being significantly lower than its UCS.

As the compression wave expands out, it is also reflected at nearby free-faces or internal discontinuities within the rock mass (such as cracks and joints). The wave is reflected as a tensile wave, which causes further tensile failure to the rock mass. Since rock is brittle/weak under tension, these reflected tensile waves are where most of the fracture damage occurs.

Almost immediately after the shockwave has causes the initial stress induced damage, the gasses generated by the chemical reaction begin to rapidly expand into the pre-existing and newly formed cracks within the rock mass, following the path of least resistance. The high-pressure gasses apply further tensile strain on the rock as the fragments are forced apart, further promoting the propagation of the fractures. As the gas expands the rock swells which creates additional flexural stresses at the freeface of the rock mass, leading to new tensile cracks further fragmenting the rock, effectively snapping it.

As the gas expands the fragmented rock is thrown in the direction of the blast. The final rock breakage mechanism is through collision with other rocks, through this is not directly associated with the blasting mechanism.

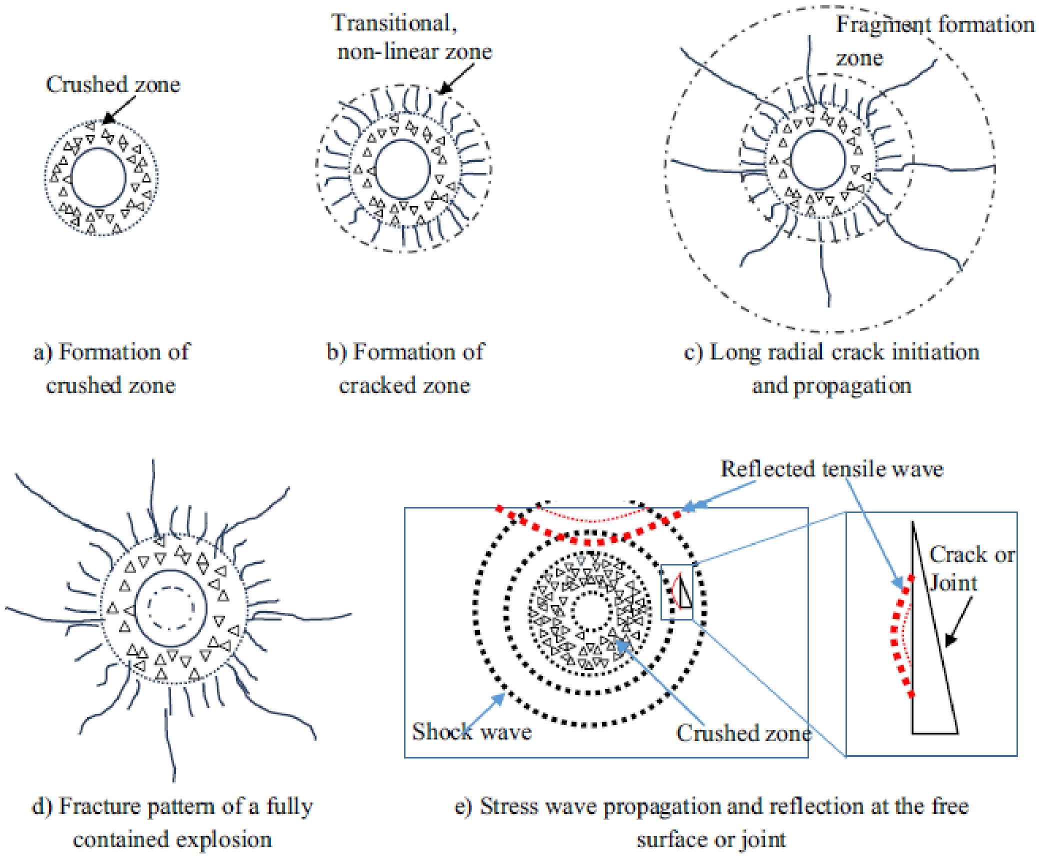


Figure 3-9 - Blast induced mechanism of rock breakage. (An & Liu et al., 2017)

3.3.4 Effect of rock jointing on fracture propagation

The geological setting of the blast has a significant impact on the behaviour of the fracture formation and overall effectiveness of the blast. Hustrulid (1999) illustrates the concept in Figure 3-10.

As the shockwave travels outward its pressure is attenuated. In heavily jointed/bedded ground, further energy is lost when some of the shockwave is reflected off the discontinuity as a tensile wave. This inhibits the expansion of the fracture zone and overall reach of the shock energy. Radial fracture formation is also inhibited as it reaches existing joint structures. An analogy can be made to a brittle fracture in a glass windscreen in Figure 3-11. The crack will extend until it hit a weakness plane at which point it will stop as pressure is relieved at existing joints.

When the gas expands, rather than further expanding the radial cracks generated by the detonation shock wave, the gas pressure is relieved through the pre-existing cracks and fissures in the rock mass. As such, the existing discontinuities of the rock mass can directly impact the fracture pattern of the blast which can impact overall fragmentation. (Abu Bakar & Hayat et al. 2013)

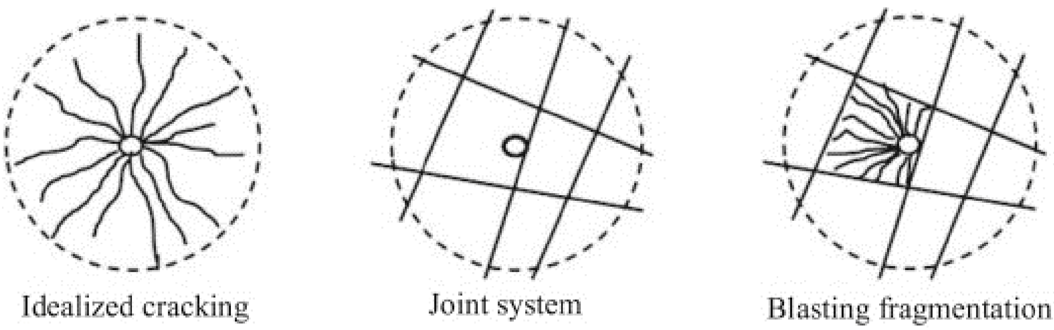
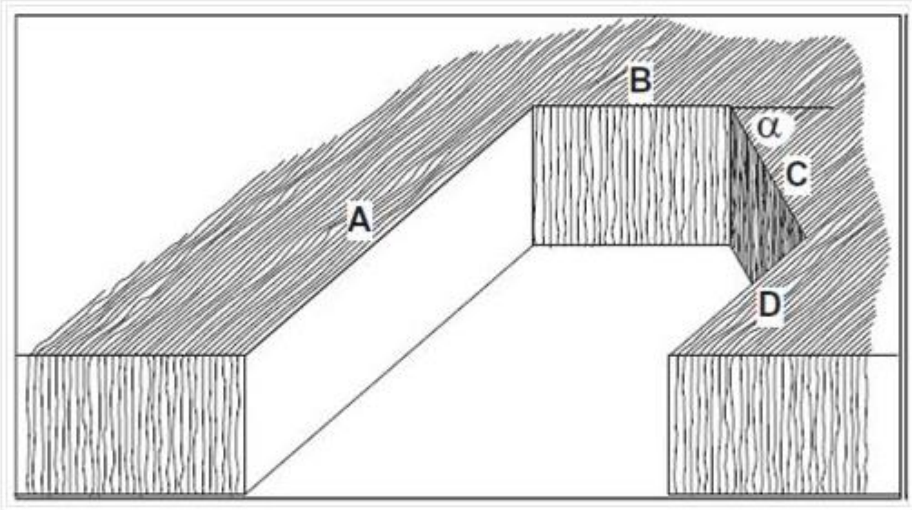


Figure 3-10 - Effect of joints on fracture propagation (Hustrulid, 1999).

Figure 3-11 - Crack propagation is arrested at pre-existing discontinuities. (Cal Auto Glass, 2016)

Partha Das Sharma (2012) illustrates the impact of anisotropic rock on fragmentation during blasting in Figure 3-12. He notes that when blasting a face in the direction of A the fragmentation and over/underbreak results will be more desirable than when blasting a face in the direction of B. Figure 3-12 is analogous to developing a tunnel in an underground environment, and specifically the ground type experienced at Agnew Gold Mine, where the ground is high laminated in a single orientation, and the drives faces are often similar to Face B, referred to as developing through "end-grain".



FACE DIRECTION	FRAGMENTATION	BACKBREAK ANDTOE PROBLEMS	BENCH FLOORUNDERBREAK
A	Medium (1)	small	medium
B	Poor (2)	large	large

Figure 3-12 - Effect of anisotropic ground conditions on blast fragmentation. (P.D. Sharma, 2012)

3.4 EXPLOSIVE PROPERTIES & TYPES

3.4.1 Energy Profile

In any explosive reaction the total energy released is finite, and is manifested in various ways such as heat, sound and mechanical energy. The useful work done by the explosive is the mechanical energy component and is further described by its ability to shock and fracture the rock (known as brisance) and its ability to throw the rock (known as heave). The brisance is of an explosive material is determined by the energy of the initial detonation shockwave, and the heave is the mechanical work done by the gas as it expands through the cracks and moves the broken rock.

3.4.2 Bulk Explosives

The two most common bulk explosives used in mining are ammonium nitrate mixed with fuel oil (ANFO) and emulsion-based derivatives of ANFO (referred to as “Emulsion”). The fundamental chemical reaction during an ANFO-based explosion is represented by the formula

$$3NH_4NO_3 + CH_2 \rightarrow 3N_2 + 7H_2O + CO_2.$$

The ammonium nitrate (NH_4NO_3) provides the oxygen for the reaction and the fuel oil (CH_2) provides the hydrocarbons used to fuel the reaction. Though both ANFO and Emulsion are a mixture of the same chemicals, their constitution is different which results in different explosive properties.

ANFO is a blend of porous ammonium nitrate (AN) prill and fuel oil. Due to the air pockets or void surrounding the ANFO prill, the explosive reaction is less then ideal and so detonates relatively inefficiently causing a slower shockwave (i.e. slower VOD) and higher percentage of gas than an emulsion explosive. As discussed earlier, the VOD of ANFO is approximately 4,500m/s.

Emulsion explosive is a grease-like matrix of AN solution and fuel oil. The emulsifier enables microscopic particles of AN to be suspended tightly against fuel particles and micro pockets of air (after the addition of a gassing nitrite component) improving conditions for an ideal chemical reaction. Due to the more efficient reaction in emulsion, the shockwave produced during detonation is more energetic (i.e. faster VOD / higher brisance) and less gaseous (i.e. less heave) than standard ANFO. The VOD of emulsion is approximately 6,000m/s (Orica, 2019).

Due to the higher density of emulsion relative to ANFO, emulsion has a greater relative weight strength and is known to have a higher energy density than ANFO. (Orica, 2019)

Emulsion explosive also has the benefit of being water resistant, whereas ANFO prill is prone to dissolving in water. ANFO/Emulsion blends are also used to gain the benefits of each explosive type (e.g. controlling density range, water resistance, cost factors, etc).

3.5 Bulk Explosive Selection

Explosive selection should be primarily based on explosive properties with respect to performance requirements and ground type, however other factors such as cost are also important considerations in a blast design. (Konya & Konya, 2019)

3.5.1 W.r.t Ground Type

A stronger rock type requires more energy to break than a weaker rock type, and so a bulk explosive with a higher energy density would be more appropriate. Alternative, rock types with low strength require less energy to break and so an explosive with lower energy density would be a more cost-effective solution.

A homogenous rock type with a low fracture frequency will require the explosive to create the fractures, and so an explosive with a higher VOD would be best suited.

3.5.2 W.r.t Performance Requirements

A highly fractured ground type with existing discontinuities will not require a high fragmentation requirement but rather enough heave energy to pry the rock apart at the existing discontinuities and so an explosive with a lower VOD will be more appropriate.

A scenario that requires the rock be moved as much as possible will require an explosive with a high heave energy (i.e. a lower VOD). A scenario that requires the blasted material remain confined to a smaller area may consider using a product with a higher VOD to minimize the throw.

Fidler (2009) illustrates the above considerations in Figure 3-13.

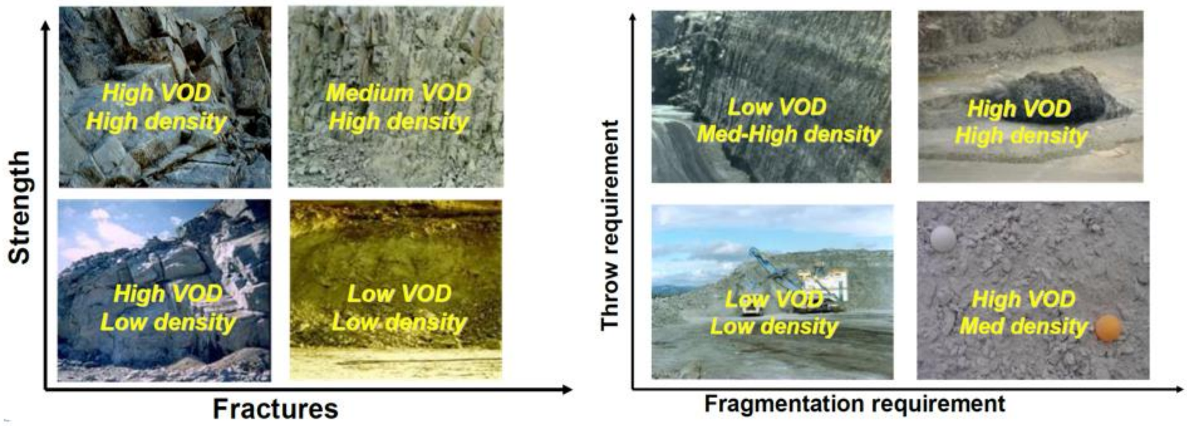


Figure 3-13 - Bulk Explosive selection based on Rock Type & Explosive Properties

3.6 Bulk Explosive Selection - case studies

Several research papers are available that investigate operational differences between ANFO and Emulsion in order to make a data-driven decision.

Chikande & Zvarivadza (2017) conducted a comparative study of Emulsion vs ANFO at an underground room & pillar mine in Zimbabwe in anisotropic stopes (6mW x 2mH) that were drilled to 3.2m deep to achieve an advance rate of 2.8-3.0m (87%-94% of ideal cut length). Using emulsion of 1.1g/cc, Results showed emulsion to be a more effective blasting agent in this mine relative to blow-loaded ANFO (0.95g/cc) with these drive dimensions with the increase in powder factor driving a slight increase in cut advance to 96% ideal cut length as well as increased fragmentation (3% increase in blasted material passed through a 400mm x 400mm grizzly aperture). The overall results showed that an increased powder factor resulted in greater advance rate at higher cost per blasted tonne using emulsion. A reduction in overbreak was also observed. Drill patterns did not appear to be altered throughout the study, nor the reamer pattern. Likewise, the study did not mention which type of boosters or timing/sequencing of blast holes – the primary parameter was bulk explosive selection.

Widodo et al (2019) conducted a comparative study of Emulsion and ANFO and their effect on overbreak and underbreak in an underground development setting at Deep Mill Level Zone, Freeport Mine in Indonesia. The key rock type was diorite and joint spacing was measured between 1.0m-3.0m with UCS 156 MPa. The results showed ANFO was more prone to overbreak while emulsion was more prone to underbreak in this ground – although there was no mention of drill design parameters or methods used to control perimeter hole placement. The study noted the costs were greater with emulsion cuts.

3.7 Design Optimisation & Practice

The Dyno Nobel Engineers Handbook (2020) provides various rules of thumb, definitions and guidelines for blast design using various explosives. It describes the basic parameters considered in a blast design such as burden, spacing, and powder factor. It describes various explosives and booster types to consider in the operation.

Explosives.org (2022) states that the blast design process can begin once the existing site conditions are understood. It then states that the goal is to distribute the explosive throughout the rock mass so that the breakage achieves the desired result and is easily removable by the excavation equipment. The article describes the role that detonation sequence between blast holes plays improving rock breakage and controlling off-site effects such as ground vibration.

Konya & Konya (2019) describes the impact over-fuelling or under-fuelling of the explosive has on the performance of the blast, whereby over-fuelling reduces the overall energy released and under-fuelling leads to an increase of toxic NOx gasses being produced as a by-product of the blast.

Konya & Konya (2019) state the decision for bulk explosive selection comes down to the environment in which the explosive will be used and what is most cost-effective in terms of both explosive cost and drilling cost. In order to determine a blast designs success, the authors recommend the simple method of viewing the blast and blast results.

Pybar Mining Services (2017) published their method for trialling ANFO vs Emulsion at Carrapateena mine for their decline development. The method included establishing a baseline (for cycle times, advance rate, and overbreak) using a fixed parameter (ANFO) and then changing the parameter to Emulsion and testing 11 development cuts. 3D scanning was used to determine impact on overbreak. The only parameter investigated was the bulk explosive type.

Orica's (2022) pocket blasting guide suggests various changes to make to the blast design in response to poor tunnel advance –

- Review cut drilling to ensure holes are correct length and parallel
- Review charging quality to ensure primers are located at the toe of the blast hole
- Increase number or size of Void Holes in the Cut Area
- Skip delay numbers to slow the cut initiation sequence
- Review initiation sequence to ensure adequate relief
- Reduce hole spacing to increase energy at the toe
- Attempt a charge utilizing low density product in the cut

Overall there is sufficient information and literature to design a blast using rules of thumb, but there is no literature in the underground mining space that describes a systematic method for multivariate optimization of development cut blast designs.

4 A PRAGMATIC APPROACH

4.1 PRAGMATISM – WHAT DOES IT MEAN TO BE PRAGMATIC?

Pragmatism is an approach that evaluates theories or beliefs in terms of the success of their practical application (Oxford, 2022a). To be pragmatic, in essence, is to be both reasonable and practical. It is to think about solving problems in a practical and sensible way rather than by having fixed ideas or theories (Oxford, 2022b). In the mining industry often people's opinions are expressed as fact, particularly the more senior the engineer or more experienced the operator, and those opinions can tend to be held quite steadfastly. While experience plays a part in rapid decision making, when the data is available to assist in making the decision it is wise to utilize the data accordingly and effectively. If the data is not available, then it is also wise to consult with existing literature. Both the data and literature should be verified and sanity checked against real world experience and general knowledge specific to the challenge at hand.

4.2 DEMONSTRATING THE PRAGMATIC APPROACH

This study will demonstrate a pragmatic methodology combining both statistical and empirical analysis of operational data against a backdrop of established blasting and rock breakage theory to determine critical design parameters for success in developing through Agnew's laminated geology. Cost analysis will be used to confirm the commercial benefit of optimizing the blast design for improved advance.

At all times, the hierarchy of considerations for optimized blast design will be employed as in Figure 1-1. After an in-depth literature review covering blast mechanics and theory, industry rules of thumb, existing explosive applications and various blasting techniques, as well as a discussion of the results of the informal site studies (internal + external) conducted previously, the methodology outlined in Table 4-1 will be used to address the optimization problem.

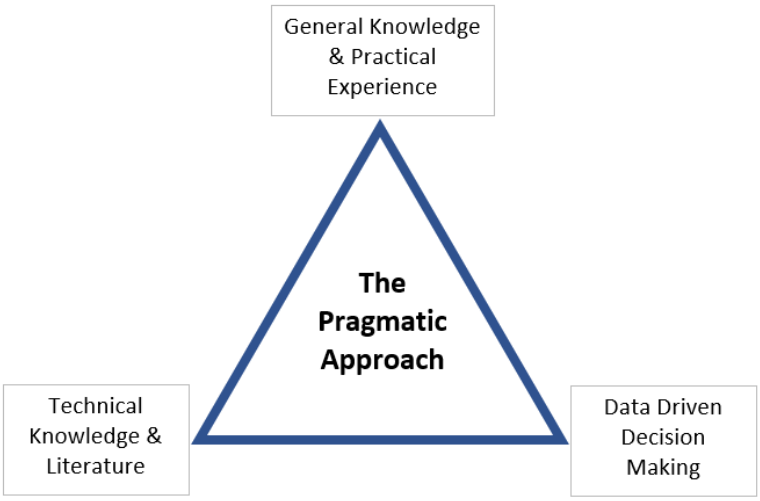


Figure 4-1 - The pragmatic approach draws from all available sources of information to drive a practical and reasonable result.

1. Understand the problem	Benchmarking current practices (Design & in the field)
	Failure Modes and Effects Analysis
	Affinity Diagram – Factors contributing to failure
	Identify and define Success Metrics
	Identify uncontrollable and controllable parameters, specifications and related success metric
2. Data Collection	Describe the data collection plan
	Face Markup Sheets (consider both controllable & uncontrollable parameters)
	Drill hole deviation sheets
	Development of Excel Database with Visual Basic input form & Automated Chart Analysis
	Establish operational performance benchmark
3. Data Analysis & Interpretation	Sensitivity Analysis & Scatter plots
	Linear regression
	Histograms
	Statistical Analysis
	Regard to be given to established rock breakage theory, factors contributing to failure, success metrics.
	Geology
	Drilling Practice (Bits, Steels, Feed & Rotation Pressures)
	Explosive consumables (Primers)
	Bulk Explosive (ANFO vs Emulsion)
	Cost Analysis of old designs vs new designs
4. Generation & Implementation of Improvement Plan	**UTILISE HIERARCHY OF CONSIDERATIONS FOR OPTIMISED BLAST DESIGN**
	Generate standardized blast design
	Rationale to be provided for the selection of all parameters with respect to results
	Collect data and compare to benchmark results
5. Discussion of new results	Effectiveness of standardized design
	Overall Cost Benefit to operation

Table 4-1 - Methodology for optimizing drill and blast designs

5 UNDERSTAND THE PROBLEM

5.1 PROBLEM STATEMENT

Agnew's development advance is historically below the required standard, particularly when mining in parallel with laminated ground. Target advance is 4.1m per cut while the actual operational average is 3.6m per cut.

5.2 REVIEW OF INTERNAL INFORMATION & INFORMAL STUDIES

Internal and external studies had been conducted by various groups in the preceding 12 months period to ascertain the cause of the under-performing advance rates.

The findings summarized in Table 5-1 highlight various potential sources of failure, depending on who was conducting the investigation or providing the summary.

While the internal studies highlighted various potential sources of failure (e.g. explosive selection, drilling technique, and ground conditions), the external explosives supplier determined that the only problem was the drilling (Table 5-1).

Despite the explosive supplier's determination that drilling was the cause of the problem, the report went on to recommend an increase in powder factor (by increasing bit diameter or by an increase in the emulsion density – both of which result in an increase in explosives costs).

Due to operational constraints in both time and manning resources, the data used by all previous studies was found to be of a small sample size, incomplete and lacking formal structure. For the same reason, the investigations did not include statistical analysis, and used only high level inferences without consideration to parameters outside of the bulk explosive selection. Once the data had been sanitized and structured, the results of the external analysis could not be replicated using the original sample data.

Overall, the informal investigations highlighted some of the types of issues that could be contributing to the failure of each cut to reach maximum advance, but there did not appear to be enough supporting evidence for the claims. Further, the studies tended towards a single primary reason for failure – all significantly different from the other.

PREVIOUS INVESTIGATIONS	KEY FINDINGS	CONTRIBUTING FACTORS DISCUSSED/SUGGESTED	RECOMMENDATIONS MADE
AGN Development Review, July 2017, Barminco.	• Consistently missing target by 100-200m per month • Most significant operating delay was excessive scaling • Average cut length 3.82m from a theoretical max of 4.25m	• Primary - The use of emulsion • Secondary - Poor ground conditions, drill hole deviation, excessive overbreak	• Train operators on drilling techniques in poor ground (e.g. perimeter standoff) • Ensure low density explosive used in perimeter holes • ANFO to be used full time in all development cuts Improve control of charge plans & emphasize their importance
Improving Development Efficiency, Nov 2017, Barminco.	• Mid-October reconciliation showed actual advance 33.2m less than claimed • EOM overbreak @ 13% (target is 8%)	• Excessive scaling & re-bogging contributed to reduced cycle efficiency	• Introduce new charge plan that streamlines capture of relevant information • Reiterate to supervisors importance of inspecting headings & ensure charge plans filled out correctly • Encourage discussion of better drill/charge practices between operators • Implement "performance discussion" instead of "what cuts have we fired" discussion
Development Review / Site Visit Report, Dec 2017, Orica.	• Analysis of historical data (27th Sep to 1st Dec) shows that the advance achieved with ANFO and Emulsion is the same, therefore the cause of the poor advance is not the explosive.	• Hole deviation is the major cause of poor face advance.	• Increase hole diameter from 45mm to 48mm is recommended to improve accuracy of drilling and advance. • Possibility of trialling 51mm diameter and 1.1g/cc emulsion will be explored.

Table 5-1 - Company investigations into the development advance performance at Agnew - Waroonga.

5.3 BENCHMARKING CURRENT PRACTICES

Part of understanding the problem is investigating the current mining practices to determine if there are any obvious opportunities for improvement. The practices worth noting are outlined in Table 5-2.

	AREA	GENERAL OBSERVATIONS
DRILLING	**General**	Guide steels used half of the time. Used more frequently when boring around corners (e.g. beginning of cross-cuts, declines, etc.) than in flat/straight drives.
		Operators drill blast patterns that they have subjectively observed to work best. No standardization between bit size, pattern size and reamer pattern.
		Operators reamer formation differs between operators but on average most operators choose a 6 Reamer Circle pattern.
		Only feedback on cut performance available to operators is the brief subjective feedback discussed between operators during cross-shift. For example, "Did it come out alright?" "Yeh, it pulled."
CHARGING	**Administrative**	Procedure does not exist for charging explicitly with emulsion (only more generalised charging procedures). Orica has provided instructions for charging with the Hypercharge unit.
		The Emulsion Quality Assurance (QA) Test procedure is an old Orica procedure for Cup Weight density tests.
	In-Practice	Both ANFO and Emulsion used in development cuts depending on machine availability. Bulk explosive selection is independent of drill pattern design.
		Most operators have been formally trained on Orica's hypercharge unit.
		Supervision is difficult due to time constraints and supervisors not having had practical experience charging with emulsion, and have not received the same formal training on the hypercharge machines as the charge up operators.
		Operators choosing not to bunch nonel lead tails after clipping to detonating cord ("det cord") to create a clear work area. Doing so assists in preventing misfires.
	Cup Weights	Operators have been trained by other operators to fill each hole with a set number of 'pumps' of the emulsion hose. Differing opinions exist amongst operators on how much this should vary depending on 4.3m/4.9m cuts, 45mm/48mm bit, 0.6/1.0 g/cc density.
		Some operators not correctly measuring cup weights (i.e. Waiting the correct time before recording the cup-weight density).
		Cup weights vary between charge rigs used (2 x emulsion rigs on site, 1 x ANFO rig)
		Cup weights range from 1550g to 1750g (0.85 to 1.0 cup density)
		Orica document provided to site engineers recommended adding "0.15" to the cup density measurement to estimate in-hole density. This is a fixed value regardless of hole size, hole length, etc. Interestingly, a copy of the same document existed on file that did not include the footnote that suggested adding 0.15 to the density measurement.
		Based on current measurements, this leads to in-hole densities ranging between 1.0 and 1.15g/cc.
		Foreman observed correct cup weight process measure 1743g/cc over 40 minutes. Engineer observed correct cup weight process measure 1550g/cc over 25 minutes. Gasser Product selected was the same for both tests.

Table 5-2 - Current general practices by operators at Waroonga.

6 DATA COLLECTION & MEASUREMENT

Poorly planned data collection runs the risk of collecting superfluous information that detracts from a clear end-goal.

To determine which data needs to be collected, both critical-to-quality (CTQ) factors and factors contributing to failure must be identified. From this analysis, appropriate success metrics can be identified and used to determine which input parameters should be analysed.

6.1 Critical to Quality Factors

A CTQ analysis was conducted and identified the primary need of a development program as the need to stick to the budgeted schedule, which requires both rapid cut turnover and successful cuts. The analysis in this study will focus more so on generating a successful cut (which also contributes to rapid cut turnover). Rapid cut turnover (often referred to as “high speed development” in the industry) is also driven by human behaviours designed to maximise utilisation the jumbo and boggers in the target work areas. These behaviours broadly include effective time and fleet management, a respect for safe practices, a sense of urgency and all the micro-behaviours (e.g. a bogger returning the pump to the heading prior to the jumbo pulling back in) that improve operational efficiency.

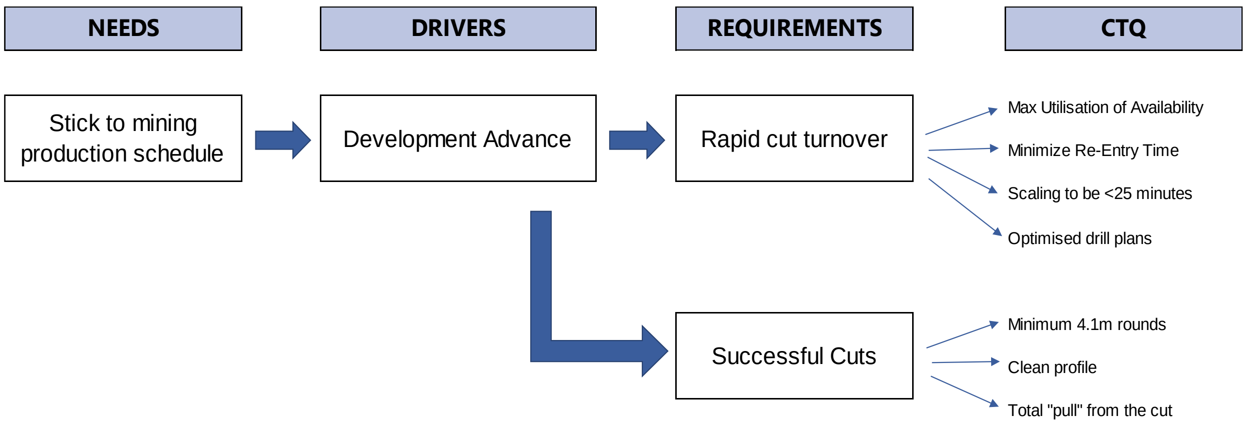


Figure 6-1 - Critical to Quality Factors for development mining at Waroonga

6.2 Failure Modes & Effects Analysis

A failure modes and effects analysis was conducted to identify potential modes of failure or factors that would lead to an unsuccessful cut (e.g. A cut not pulling to full length). Table 6-1 investigates and identifies 5 key modes of failure and the respective causes. This information can be used to drive the design of data collection models to measure appropriate drill and blast parameters.

FAILURE MODE	**FAILURE EFFECTS**	**CAUSES**
In what ways can the Key Input (the Drill & Blast Cycle) go wrong?	**What is the impact on the Key Output Variables or internal requirements?**	**What causes the Key Input to go wrong?**
Insufficient Scaling	Difficulty drilling / charging face	• Rushing • Poor Operator Judgement
Incorrect drill pattern for the circumstance	Cut chokes up and doesn't pull properly	• Not enough holes • Burden/Spacing too high • Not enough Reamers
Hole Deviation, Hole too short	Inefficient distribution of blast energy	• Feed pressure too high • Rotation not optimised to percussion • steel too flexible • Bit not optimised to rock type
Incorrect primer for circumstance, incorrect timing sequence	Ineffective fracture & throw characteristics	• Lack of correct inventory • Poor assessment of the blast plan & ground conditions
Incorrect bulk explosive for the circumstance, Incorrect charging technique	Ineffective fracture & throw characteristics	• Bulk explosive has too low a density (not enough energy) • Non-Continuous charge caused by poor charging technique

Table 6-1 - Failure Modes & Effects Analysis for Development Mining

6.3 Factors Contributing to Failure

The results from the failure modes and effects analysis as conducted in Table 6-1 can be used to inform and guide a factors contributing to failure affinity diagram. By first splitting the factors into either controllable or uncontrollable groups, we can begin to consider how these will impact design parameters. Figure 6-2 identifies 9 factors contributing to failure, grouped into one of four sub-categories which fall into either controllable or uncontrollable factors.

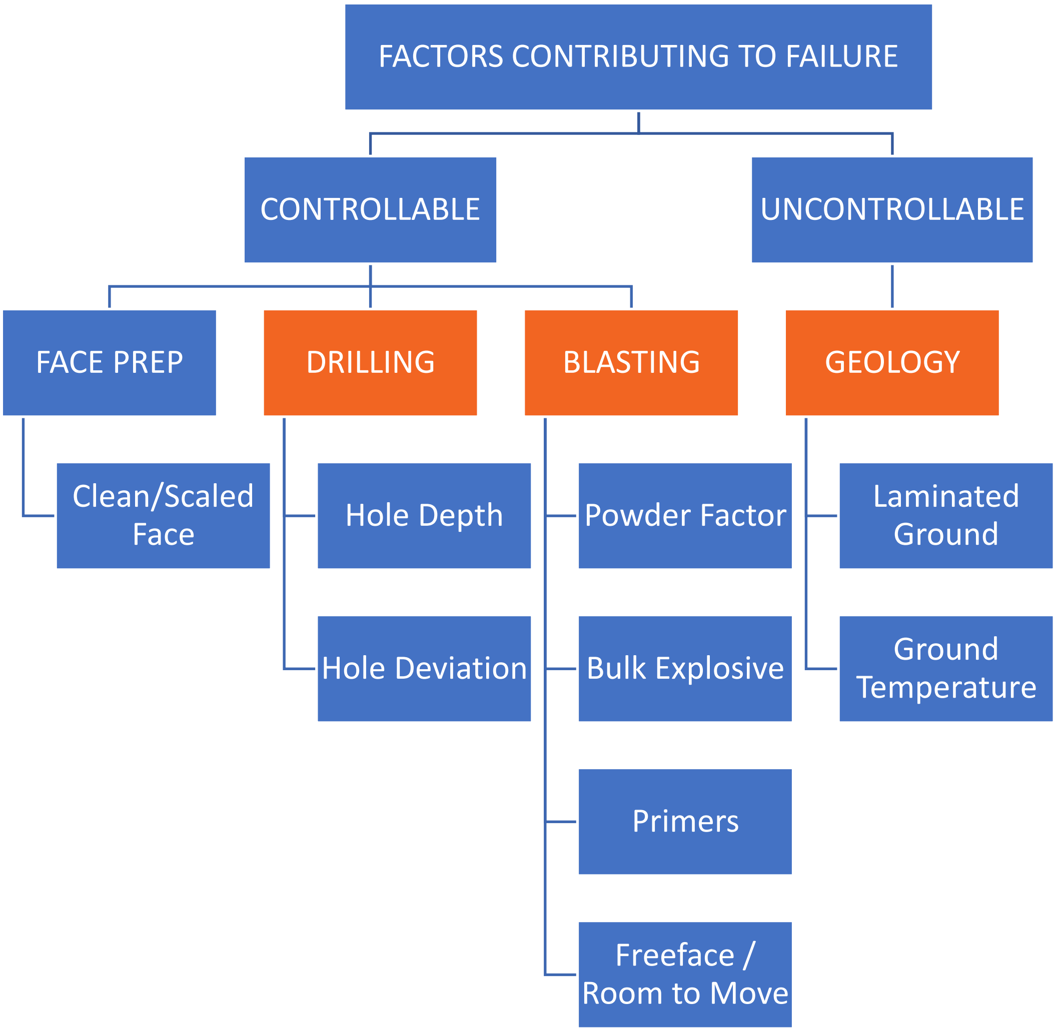


Figure 6-2 - Affinity Diagram - Factors Contributing to Failure

6.4 Identify Relevant Parameters (Uncontrollable and Controllable)

The factors determined in Figure 6-2 are investigated further in Table 6-2 and relevant real-world design parameters that drive the success of the identified design factors are listed for consideration in the data collection program.

<table>
<tr><th></th><th></th><th>IDENTIFIED FACTORS</th><th>RELEVANT PARAMETERS</th></tr>
<tr><td rowspan="7">CONTROLLABLE</td><td rowspan="2">DRILLING</td><td>Hole Depth</td><td>• Steel length</td></tr>
<tr><td>Hole Deviation</td><td>• Drill Collar
• Bit Type
• Bit Diameter
• Drill Steel Profile
• Feed Pressure
• Rotation Speed</td></tr>
<tr><td rowspan="4">BLASTING</td><td>Powder Factor</td><td>• Number of charged holes
• Spacing, Burden
• Bit Diameter
• Charge Length
• Charge Density</td></tr>
<tr><td>Bulk Explosive</td><td>• Product Type / Specifications</td></tr>
<tr><td>Primers</td><td>• Product Type / Specifications</td></tr>
<tr><td>Freeface / Room to Move</td><td>• Timing Sequence
• Direction of Blast</td></tr>
<tr><td rowspan="2">UNCONTROLLABLE</td><td rowspan="2">GEOLOGY</td><td>Laminated Ground</td><td>• Firing perpendicular to laminations if possible
• Increase number of holes across the lamination strike
• Ensure appropriate explosive selection</td></tr>
<tr><td>Ground Temperature</td><td>• Suitable bulk explosive product</td></tr>
</table>

Table 6-2 - Relevant Parameters to be considered based on controllable and uncontrollable factors

6.5 Quality/Success Metrics

The quality/success metrics are defined in Table 6-3 and will be used in the data collection program to measure the success of a cut. Once the success metrics are defined, the next step is to map the success metrics to their related design parameters as per Table 6-3.

	Description	The Metric	The Metric Defined
PRIMARY	Directly related to the scope of the study	Actual Advance Metres per Cut	The difference between the chainage measurement of one cut, and the chainage measurement of the following cut.
SECONDARY	Indirectly related to the scope of the study	Scaling Time	The time taken to effectively scale the face of a heading in order to deem it safe for human approach.
		Cost-Benefit	Do the production benefits outweigh the cost of the improvement strategies?
SUPPLEMENTARY	Offers additional understanding to the problem	In-Hole Deviation	The degree to which the toe of the drill hole deviates from it's desired end-point in space.

Table 6-3 - Quality/Success Metrics

6.6 Map related parameters to the measurable metrics

To close the loop and ensure that the data collected is designed for a specific purpose, it is important that the relevant parameters identified in Table 6-3 are mapped against the success metrics defined in Table 6-3. This work is illustrated in Table 6-4.

PARAMETER	SPECIFICATION	RELEVANT METRIC
Steel Type	4.3m - Rounded 4.9m - Rounded	Metres Advance Scaling Time Hole Deviation
Bit Type	45mm - Round (standard) 48mm - Round (Standard) 51mm - Round (Standard)	Metres Advance Scaling Time Hole Deviation
Feed Pressure	Operator Control	Hole Deviation
Rotation Speed	Operator Control	Hole Deviation

Drill Pattern (i.e. Burden/Spacing/Number of Charged Holes)	5x5 + Shoulders + Perimeter 5 x 6 + Perimeter 4x6 + Shoulders + Perimeter See Appendix F for further detail.	Metres Advance Scaling Time
Bulk Explosive Type	0.95g/cc Blow-loaded ANFO 1.1g/cc Emulsion 1.2 g/cc Emulsion	Metres Advance Scaling Time
Primer Type	Pentex D (25g) Pentex G (110g) Senatel Magnum 32x200	Metres Advance Scaling Time
Timing & Direction of Blast	Box/Diamond Stripping	Metres Advance Scaling Time

Table 6-4 - Relevant parameters for data collection

Data Collection Paperwork

Existing operator paperwork was modified and reworked to include the relevant parameters as determined in Table 6-4. The final data collection documents can be seen in APPENDIX A and are summarized below.

6.6.1 Face Markup Sheet

The face markup sheet acts as the statutory blast plan required by law for each blast. The plan collects data on drill design and burn/reamer patterns as well as explosive usage, and from this data other parameters such as Powder Factor can be calculated both locally in specific areas of the face, and generally as a measure for the entire cut. To encourage participation from operators the data collection sheet is kept as basic and easy-to-use as possible, using primarily "tick and flick" options on both the front and back of the sheet. Operators were also required to sketch their face pattern and reamer pattern on the template provided.

6.6.2 In-Hole Deviation Sheet

The In-Hole deviation was measured using a subjective nominal scale to grade deviation as measured by the disappearance of a small LED light as it was manoeuvred up into the blast hole (taped to the end of a piece of 20mm PVC pipe as illustrated in Figure 6-5). The more the light disappears as it travels up the hole, the greater the deviation score. If the light stays completely visible all the way to the toe of the hole, the hole is given a deviation score of 0. The scoring system is outlined in Table 6-5. Care was taken to record the orientation and direction of the deviation relative to the orientation of any laminations present.

This technique was developed for the purpose of this study, in lieu of readily available survey equipment. The measurements were performed by a single site engineer in order to maintain a consistent interpretation bias across the measurements. Deviation was only measured in the burn holes and reamer holes – the most critical region of the drill pattern. The In-Hole Deviation data collection sheet can be found in APPENDIX B.

Nominal Score	**Meaning**
0 – No Deviation	LED light still visible, hole appears to be straight.
1 – Slight Deviation	LED light seen to be off center somewhat
2 – Notable Deviation	LED light not visible, ambient light still remains visible
3 – Excessive Deviation	LED light not visible, ambient light not visible; OR, Broken through to another hole.

Table 6-5 – Subjective Nominal Scoring System for In-Hole Deviation

An average deviation score was calculated for the Burn holes and Reamer holes, and an Overall Deviation score was calculated for the entire Burn for that cut.

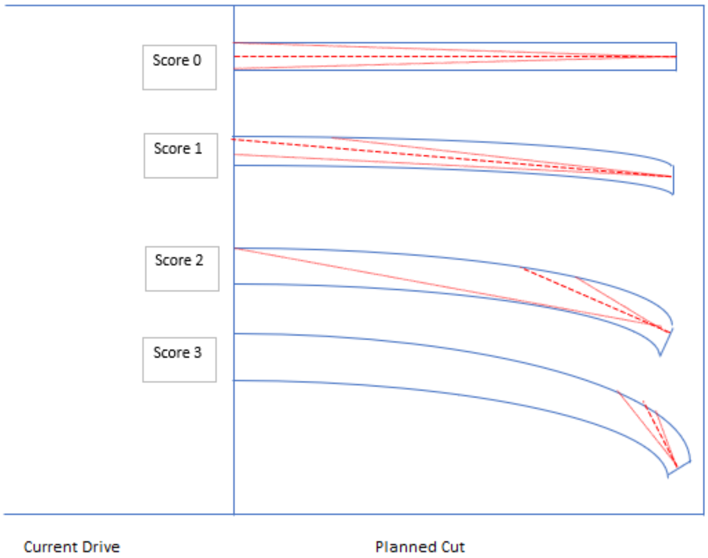


Figure 6-3 – Longsection view of deviation scoring method

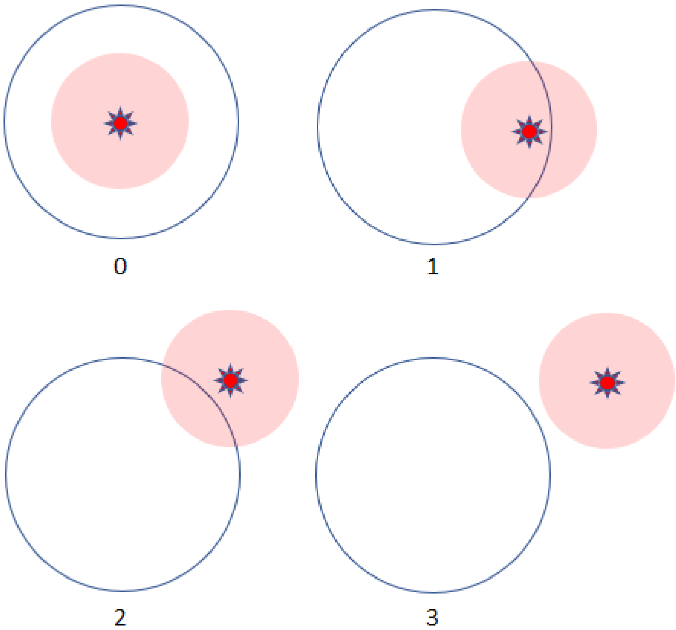


Figure 6-4 – Operator view of deviation scoring method, looking down-hole.

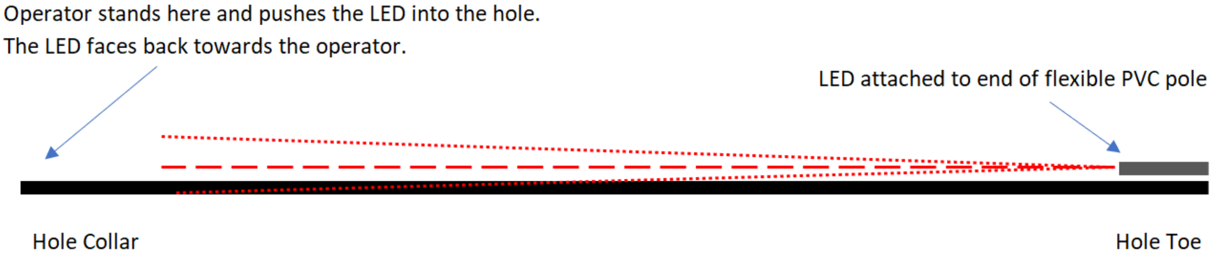


Figure 6-5- LED Deviation Measurement Tool

6.7 DATABASE

To analyse the data reliably, the data was kept in a centralized location and the data input process was standardized. A macro-capable excel database was created with a macro-driven input form to capture the information in a quick fashion after each shift. Capturing and storing the data in this way allows for detailed multi-variate analysis once the sample size was large enough. It also reduces human error as the input form is clear and user-friendly. See APPENDIX C and APPENDIX D for an example of the input forms and raw data.

7 ANALYSIS OF BASELINE DATA

The following observations are made based on one month's initial data, used to establish a baseline understanding of the blast plans created at the face by the operators and their respective results. The ideal cut length for a particular steel is defined as the steel length minus 500mm which accounts for the length of drill steel behind the jumbo boom's rubber foot, front centraliser, travelling centraliser and inside the coupling.

Due to the multi-variate nature of the problem, the pragmatic approach this study proposes is investigating which parameters are involved in the most successful cuts, in order to incorporate that particular parameter into an improved design once the data analysis has been complete. In an active mining operation, it is impractical to isolate single variables while completely ignoring others, however doing so does provide some insight into the potential for a parameters' contributing influence to a cut's success. By investigating a range of variables in such a fashion, a blast design can be built based on the best performing variables.

7.1 DATA CAVEATS

In total 136 cuts were collected in this initial period, with some data caveats outlined below.

7.1.1 Data Reliability

The reliability of the data collection was subject to human error and the quality of the collection depended on the jumbo and charge-up operators filling in their paperwork correctly (attention to detail, missing information, etc). Complete information was not obtained for all 136 cuts, so some of the charts and figures may interpret data for less than 136 cuts.

7.1.2 Outliers

Cuts that pulled greater than the ideal cut length are likely due to dishing or over-scaling. A "scaling and dishing" tolerance of 300mm has been used as a cut-off for sanitizing the data. For example, for rounds taken with a 4.9m steel (with an ideal advance rate of 4.4m), any cuts that were measured to advance more than 4.7m were considered to be erroneous and likely due to either incomplete data or reflective of ground conditions more so than blast design performance.

7.1.3 Missing Data – Deviation

The in-Hole deviation was not measured for every cut due to the engineer not having a counterpart on the opposite roster. Furthermore, the time of day the drilling of the cuts was complete varied and often clashed with other duties of the site engineer. During this period, deviation data was collected by the site engineer for 20 cuts.

7.1.4 Missing Data – Operator Error

Not all face markup sheets were completed 100% - some information was occasionally missing from the data sheet. For example, not all sheets had scaling time recorded. This was random and due to operator oversight. The frequency of missing information was not significant.

7.1.5 Measurement Error – Scaling time

The scaling time was a jumbo operator estimate – some operators referred to digital watches to assist in estimation.

7.1.6 Measurement Error – Scaled Butt

Using survey stations to measure the advance of every cut is both costly and impractical. Instead the jumbo operators were instructed to ensure that each face markup sheet included the chainage measurement used to determine their drilling offsets. The chainage measurement of the initial cut was subtracted from the chainage measurement of the following cut, and this number was defined to be the actual advance of the initial cut. Inaccuracies inherent in this method include operator error, under-scaled faces and over-scaled (or dishing) faces, as well as a laser being on a sharp angle relative to the direction of the drive.

Since Survey Mining Instructions ("Memo") were based on a specific survey laser installation, the advance metres for the last cut of each memo could not be calculated due to the change in laser position for the next memo impacting the chainage reading for the next cut.

7.2 Using Standard Deviation as a proxy for sensitivity to determine key parameters

Performing a sensitivity analysis enables a user to investigate a dependent variable's sensitivity to a change in a given input parameter while the other variables remain fixed. Due to the complex and multi-variate nature of the drill and blast problem, investigating a single variable while freezing others is impractical and unrealistic in an active mining operation. In order to determine the parameter of the blast design to which cut advance was most sensitive, a quasi-sensitivity analysis was conducted between the various subsets of data within each parameter used. Due to the non-continuous and nominal datatypes of the majority of the drill and blast parameters, average cut advance was compared between the various subsets of data available for a given parameter (with 7 data points or more), and the standard deviation of these averages was used as a proxy for cut advance's sensitivity to a change in that parameter. The calculations can be seen in Table 7-1, and parameters are ranked in order of the standard deviation for that parameter as seen in Table 7-2. The higher the standard deviation, the more cut advance was deemed to be sensitive to that parameter.

Intuitively, cut advance was most sensitive to steel length selection (which defines the target cut geometry). Cut advance was next most sensitive to orientation of the laminations in the geology with respect to the drive direction. The next most sensitive controllable parameter was the bulk explosive, and the primers (which also act as a proxy for bulk explosive selection to some extent) followed shortly after. While the result was less sensitive to reamer formation and bit diameter, there is still enough empirical data to suggest there is an impact. The drill pattern showed one of the lowest sensitivities, however it is important to acknowledge that there must also be sufficient holes in the pattern such that the blast energy is distributed sufficiently and effectively throughout the cut. There was insufficient data collected on Drill hole deviation to be included in Table 7-1, however it's impact is discussed later in this chapter.

One of the key weaknesses of this method is that it is an analysis of single variables without regard for the multi-variate nature of the problem. At best, what can be inferred, is that there was more variability in the results where changes in these particular parameters were observed.

PARAMETER	OPTION	Sample Size	Ave Cut Advance (m)	Std Dev of Average Cut Advance.
STEEL LENGTH	4.3m	31	3.35	0.412
	4.9m	87	3.93	
BULK EXPLOSIVE	ANE	91	3.70	0.232
	ANFO	27	4.03	
DRILL PATTERN	4x5	16	3.82	0.035
	5x5	50	3.76	
	5x6	29	3.76	
BURN PRIMERS	D	8	3.84	0.169
	G	82	3.69	
	PG	26	4.03	
FACE PRIMERS	D	54	3.69	0.187
	G	37	3.71	
	PG	26	4.03	
REAMER FORMATION	CIRCLE SIX	89	3.73	0.153
	FLAT INDO	14	3.96	
	TALL SIX	7	4.03	
LAMINATION ORIENTATION	NON-PARALLEL	56	4.05	0.376
	PARALLEL	62	3.52	
BIT DIAMETRE - BURN	45mm	17	3.81	0.025
	48mm	96	3.77	
BIT DIAMETER - FACE	45mm	14	3.98	0.164
	48mm	102	3.74	

Table 7-1 - Standard deviation of average cut advance as a measure of parameter sensitivity

Sensitivity Rank	PARAMETER	Std Dev (m)
1	Steel Length	0.412
2	Lamination Orientation	0.376
3	Bulk Explosive	0.232
4	Face Primers	0.187
5	Burn Primers	0.169
6	Reamer Formation	0.153
7	Bit Diameter - Face	0.164
8	Drill Pattern	0.035
9	Bit Diameter - Burn	0.025

Table 7-2 - Parameters ranked in order of their impact on Cut Advance (based on Std Dev)

7.3 Key Parameters

Some basic initial observations have been made about the data collected and are summarized below.

7.3.1 Drill Steel Length

Figure 7-1 shows the frequency of actual advance for the cuts in the initial baseline data set using 4.9m steels only (sample size of 57). The cut performance has a left-skewed distribution because cut performance has an upper bound limit (the ideal cut length) based on steel length and any measurement above 0 is possible depending on the blast. The green column represents the ideal cut length for a 4.9m steel. Ideally the distribution would be more dense around this area of the chart, rather than trailing to the left. The red upper and lower tails are included in the analysis because they are realistic on rare occasions (e.g. failed cuts freezing and only pulling 2.7m, over-scaling a cut so that it appears to have pulled 4.7m, dishing, etc).

Figure 7-1 represents a high level snapshot of cut performance without consideration of the finer variables such as geological laminations, explosive type, etc. It also illustrates the high frequency with which cuts are under-performing relative to the ideal cut length.

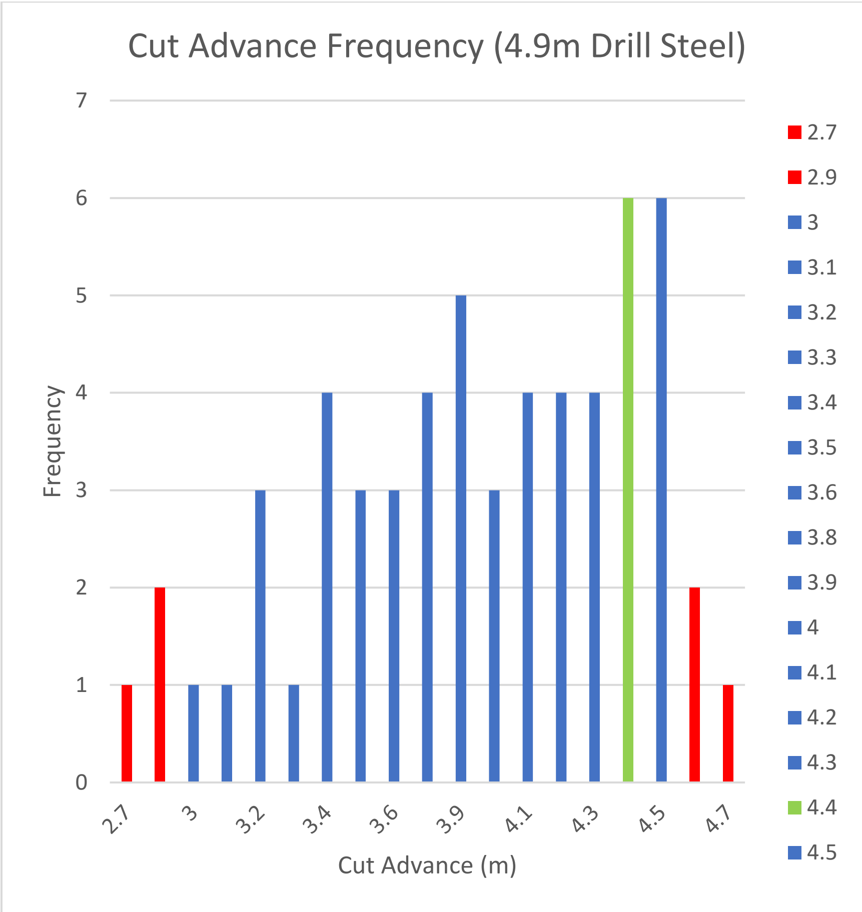


Figure 7-1 - Cut performance using a 4.9m drill steel. Green – Ideal cut length, Red – Realistic on rare occasions.

7.3.2 Bulk Explosive Performance

A more detailed breakdown of the data that includes the explosive type used is shown in Table 7-3, where it becomes evident that cuts bored with 4.9m steels are achieving 84% of ideal cut length while the shorter 4.3m steels are achieving 87% of the ideal cut length. It also shows that the ANFO cuts appear to be performing better than the ANE (Emulsion) cuts. These observations align with initial feedback during discussions with operators, and initial intuitions discussed in the internal documents. In particular, at first glance it supports the notion of moving to shorter drill steels and using ANFO to charge the cuts. These notions will be tested as hypotheses later in this study. It is important to note that other important variables have been ignored in this observation (e.g. bit diameter, primer type, number of holes, etc), so inferences from this data cannot be taken as deterministic, though it does offer some insight.

Table 7-4 illustrates the performance (both cut advance and scaling time) of the bulk explosive types in both short and long rounds. Since drive dimension directly impacts the amount of rock being scaled, it has been fixed in the below instances in order to evaluate impact on scaling time. It is observed that the scaling time in the ANFO cuts is less than that of emulsion cuts in these instances.

Steel Length	Explosive Type	Cuts Taken	Average Advance (m)	% Ideal Cut Length
4.3m (Ideal advance 3.8m)	ANE	19	3.3	87%
	ANFO	6	3.4	89%
	Total	**25**	**3.3**	**87%**
4.9m (Ideal advance 4.4m)	ANE	47	3.8	86%
	ANFO	12	4.2	95%
	Total	**58**	**3.9**	**84%**
All Cuts	**Total**	**84**	**3.7**	

Table 7-3 - Cut Advance comparing drill steel length and bulk explosive selection.

Steel Length	Drive Dimensions	Explosive Type	Cuts Taken	Average Advance (m)	% Ideal Cut Length	Average Scaling Time (mins)
4.3m (Ideal advance 3.8m)	**5.5mW x 6.0mH**	ANE	19	3.3	88%	42
		ANFO	6	3.4	90%	30
		Total	**25**	**3.3**	**87%**	38
4.9m (Ideal advance 4.4m)	**5.0mW x 5.0mH**	ANE	25	3.8	86%	41
		ANFO	5	4.3	98%	27
		Total	**30**	**3.9**	**89%**	34

Table 7-4 - Cut performance in drives with specific dimensions.

7.3.3 Performance in laminated ground

The data in Table 7-5 illustrates the cut advance performing worse in drives that are travelling in parallel with the orientation of the laminations (‘with the grain’). Cuts are observed to be performing better when traversing or travelling in a non-parallel direction (‘across the grain’) to the laminations. This observation is consistent with Figure 3-12 that suggests blasting in this type of ground will result in backbreak and toe problems. No data for “non-parallel” cuts was collected using 4.3m steels.

Steel Length	Drive Orientation vs Laminations	Cuts Taken	Average Advance (m)	% Ideal Cut Length	Average Scaling Time (min)
4.9m **(Ideal advance 4.4m)**	NON-PARALLEL	29	3.9	89%	35
	PARALLEL	18	3.6	82%	55
	Total	**47**	**3.8**	**86%**	

Table 7-5 - Cut Performance in Laminated Ground

7.4 Hole Deviation

The further a hole deviates during the drilling process the shorter the effective length of the drill hole becomes, and the greater the degree of deviation of the overall drilling, the more likely the resultant face of the cut will be uneven after blasting. An uneven face can lead to increased scaling time and contribute to sub-optimal advance. Measurements were taken for deviation to determine whether the level of deviation inherent in the current drilling practices was significant enough to negatively impact performance of the cut. Deviation was measured for burn and reamer holes reachable from the floor by hand using the light pole.

The overall deviation scores for both long rounds (4.9m) and short rounds (4.3m) have been compiled and compared against performance in both advance and scaling time, as shown in Figure 7-2 and Figure 7-3. The data represents 10 cuts using 4.9m steels and 5 cuts using 4.3m steels. The R^2 shows no effective correlation between the existing level of deviation found in the current drilling practices and the performance of the rounds that were measured. This is not consistent with the suggestion in the explosive supplier's report that suggested sub-optimal cut performance was due to issues with drilling accuracy.

However, there is a slight correlation between deviation score and resultant scaling time, as seen in Figure 7-4. It is however a weak correlation, with the most significant impact on scaling time ($R^2 = 0.43$) being due to significant burn hole deviation. The burn hole with a score of 4 was a single shot hole that had broken through into one of the reamers without being re-drilled. It was measured after the cut had been charged, but since it had broken through into the reamer (as evidenced by explosive leaking into the reamer from the shot hole) it was given the worst possible score of 4.

Figure 7-5 illustrates that in all drive sizes, scaling time weakly trends upward with a higher level of deviation. Though the density of the data means that this observation cannot be considered substantive and is only slightly indicative. The two red data points (0,50) and (1.3,45) are both from 486 Drive which has notoriously poor ground conditions. The bottom-right red data point (1.25, 20) is in the 795 PFC which has solid, competent ground with presenting with consistent half-barrels in the perimeter and so does not require much scaling at all.

This information is based on a low amount of data – only 17 Deviation QA sheets were able to be reliably captured due to the engineer's roster. Of the 17 sheets, 10 had corresponding scaling time data from the operators – mostly due to the operators forgetting or neglecting to include the information on the face markup sheet.

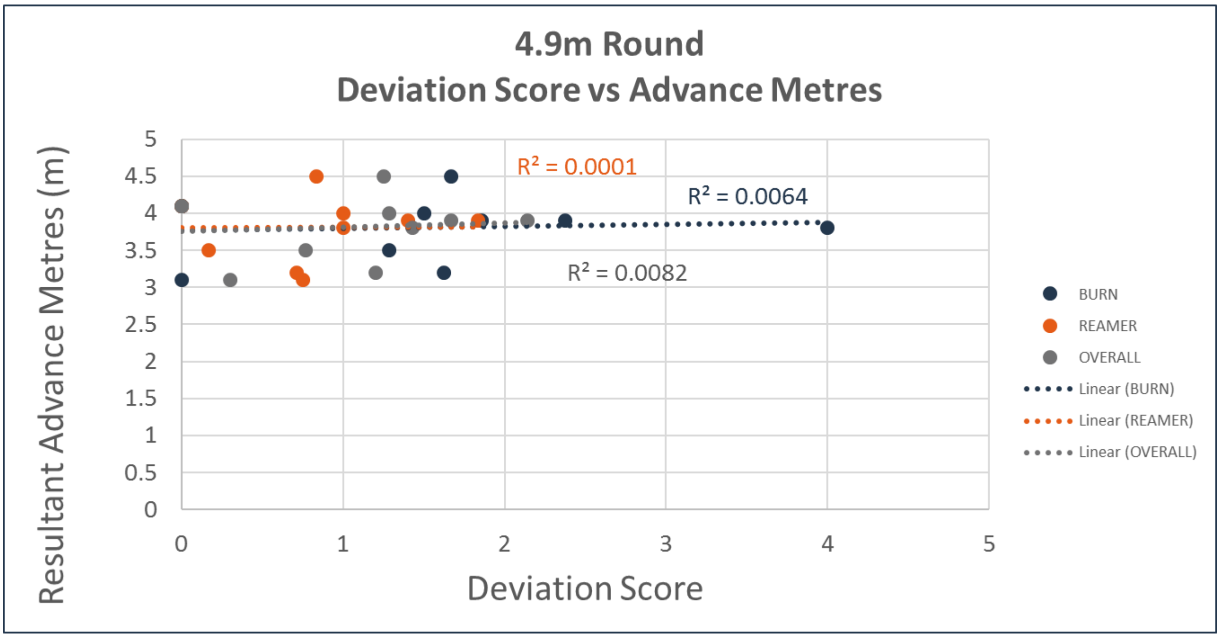


Figure 7-2 - Deviation Score vs Advance Meters for a 4.9m Round.

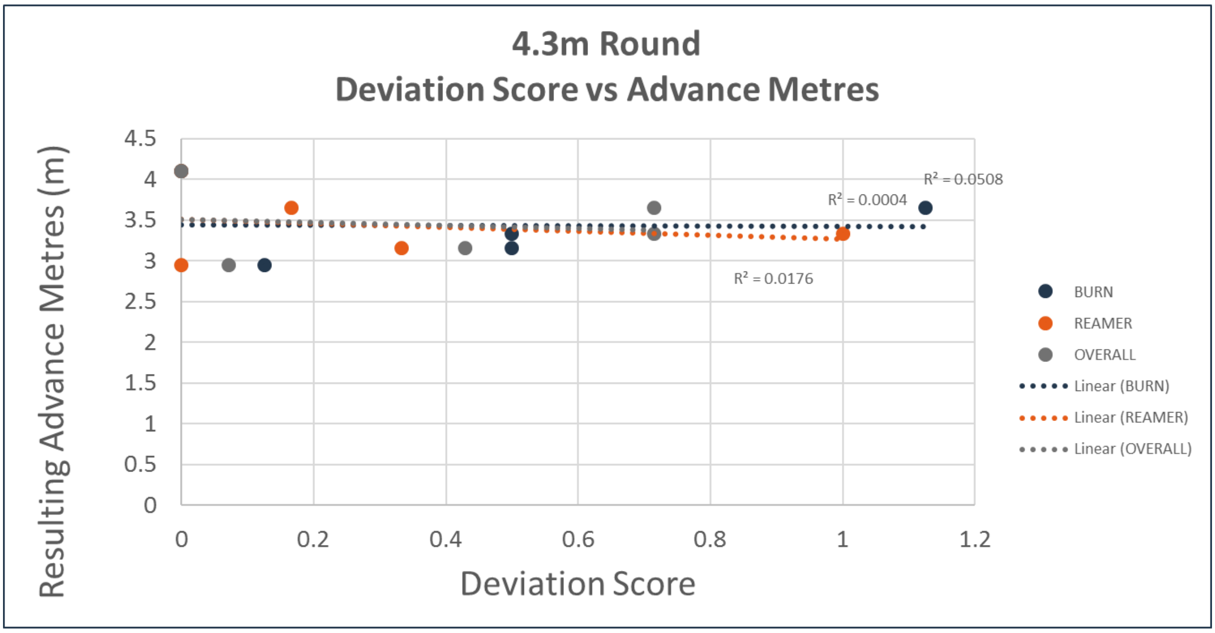


Figure 7-3 - Deviation Score vs Advance Metres for a 4.3m Round.

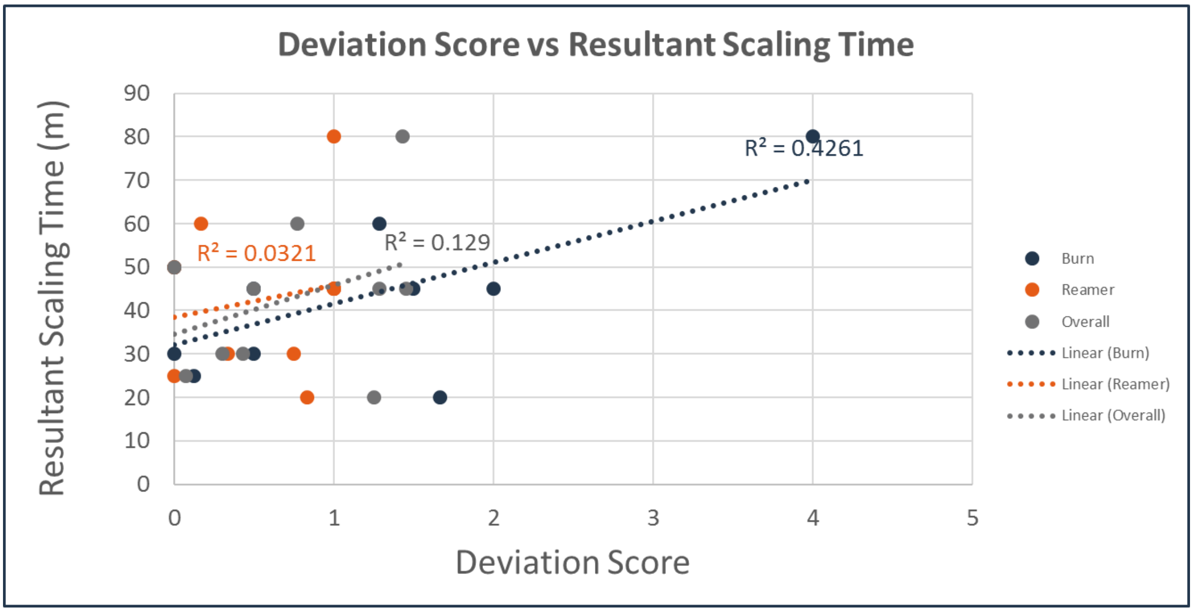


Figure 7-4 - Deviation Score vs Scaling Time

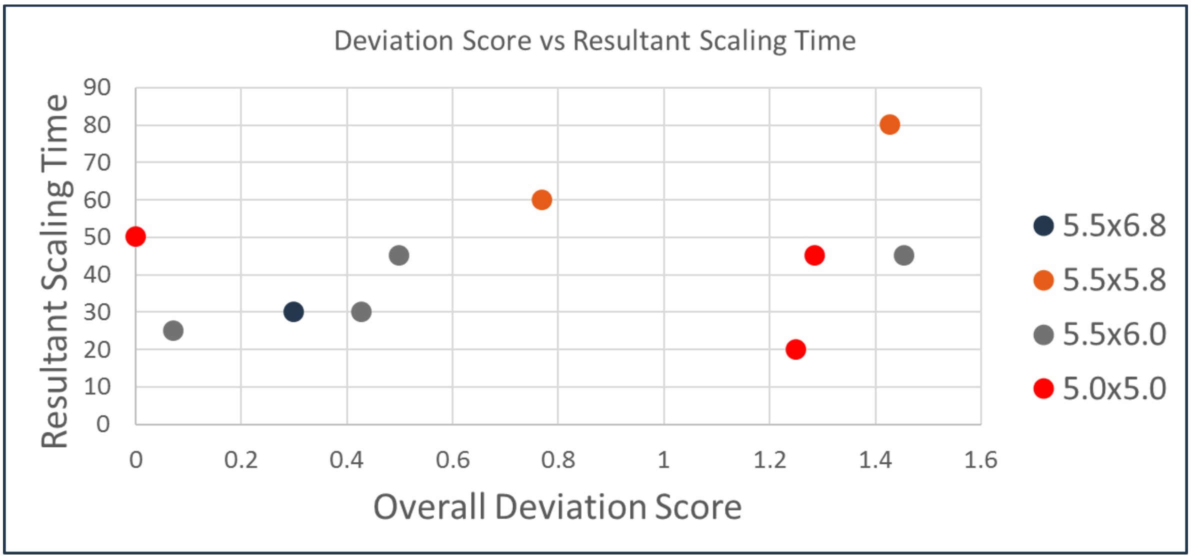


Figure 7-5 - Deviation Score vs Resultant Scaling time in Various Drive Sizes

7.5 Reamer Formations

The various reamer formations used by operators are illustrated in Figure 7-6 where the naming convention used for this study is also provided. Figure 7-7 shows that the operators are using the Circle Six formation approximately 90% of the time. There is not enough data on cuts that use a reamer formation other than Circle Six to draw reliable inferences from. However, in drives that run parallel with ground laminations, the "tall" formations have been involved in the better performing cuts. Specifically, the two cuts that used the Tall Twin Six formation pulled above average in the 520 EXP drive. This drive is a straight drive that consistently runs parallel with the ground laminations.

This is potentially due to the tall formations being "open" at the top and bottom of the formation, allowing the initial fracture of the blast holes to propagate as much as possible vertically along the lamination bedding layers without being interrupted by reamer holes. Since the laminated bedding layers are vertical, the initial fracture propagation is limited in a horizontal direction as observed in Figure 3-10.

There does not appear to be any difference in performance between the different reamer formations in drives that are not parallel with the ground laminations.

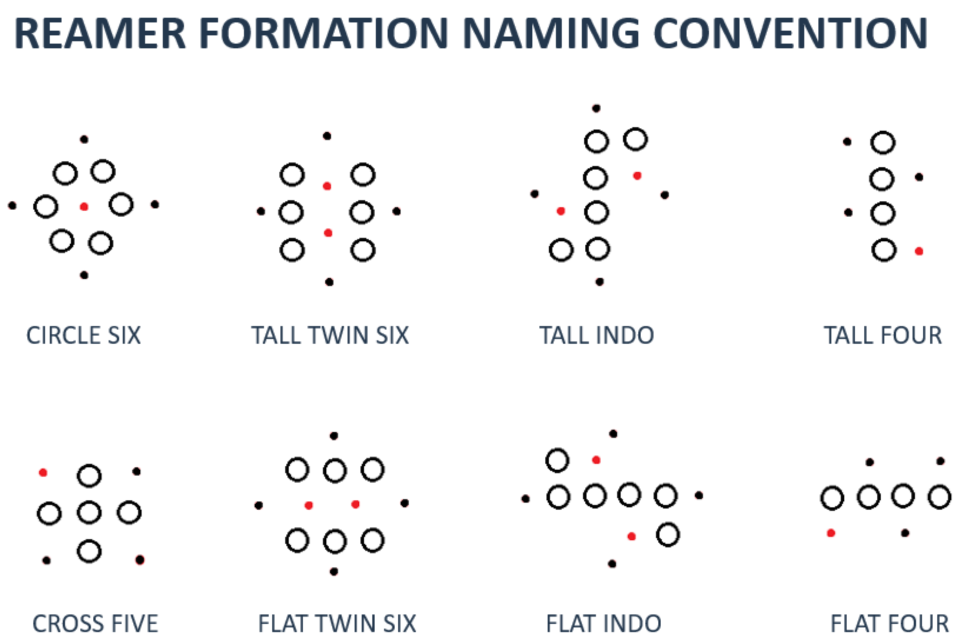


Figure 7-6 - Reamer Formation Naming Convention

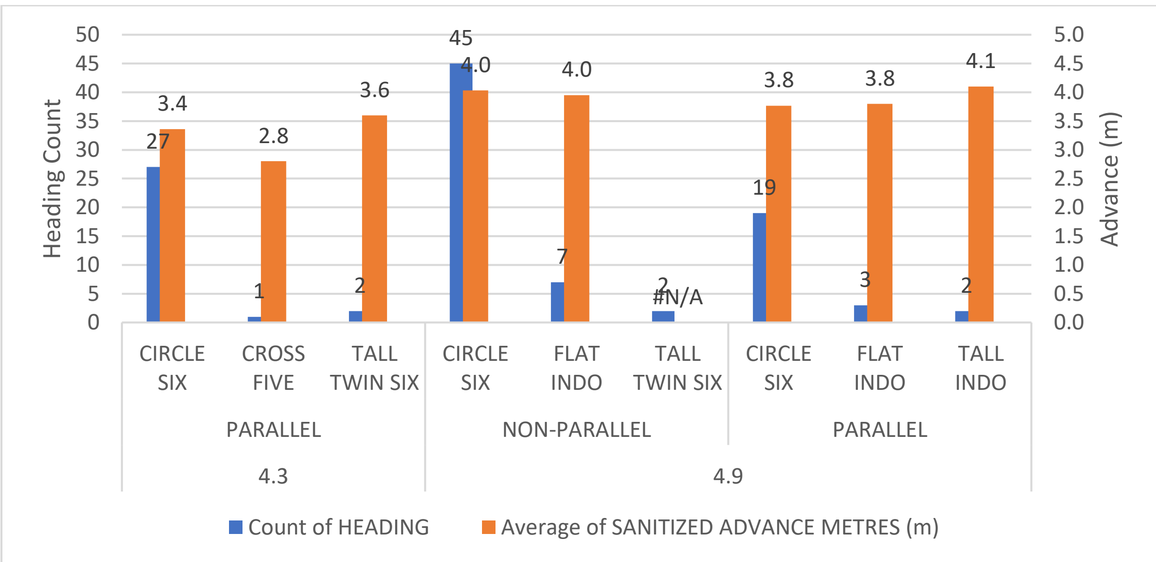


Figure 7-7 - Reamer Formations used with 4.3m and 4.9m steels in varying ground types

7.6 Drill Patterns & Powder Factor

In operations, rather than refer to burden and spacing measurements, Operators will often refer to their drill patterns by the number of rows they drill – though most hole spacings are approximately 1.0m. The number of rows drilled by the operator is dependent on the dimension of the drive as well as the operator's intuition as to the type of ground being drilled and what has been observed to work best (or alternatively, to have 'not failed') in the past. For the purpose of this study, the naming convention for the different drill patterns utilized by the operators is described in Appendix F. Importantly it is worth noting that the naming convention refers to the rows and columns drilled in the inner grid and does not include reference to the perimeter rows. The drill pattern used determines both the powder factor of the blast and the distribution of the blast energy produced.

It is observed in Figure 7-8 that there does not appear to be any correlation between cut performance and the patterns used, for either Emulsion or ANFO cuts. The data shows that most operators are using a 5x5 pattern, with the second most common pattern being a 5x6 pattern. It does however highlight that the additional row in the 5x6 pattern appears to be an opportunity for cost reduction and saving in cycle-time, as it does not appear to be improving the performance of the blasts by being included. Figure 7-9 does not appear to show any discernible difference in scaling time between less holes (5x5) and more holes (5x6 + Extra Row).

Though there is not enough data to be statistically significant, Figure 7-9 shows a slight increase in scaling time when adding additional rows of holes in the emulsion cuts. In lieu of no other data, one can speculate that the additional fracture energy of an emulsion explosive may cause additional backbreak at the face. There are also 5 data points that suggest the ANFO cuts result in an overall lesser scaling time than those that use emulsion cuts despite having the 6th row of drill holes, which may again be due to the reduced fracture energy of an ANFO explosive compared to an emulsion cut resulting in less backbreak at the face. It is also important to note that scaling time was not a controlled measurement and highly subjective based on operator practice, and so confidence in the reliability of this information for the purpose of quantitative analysis is low.

The drill pattern (and number of holes drilled) directly drives the powder factor of the cut, with additional rows of drill holes resulting in an increased powder factor. Both Figure 7-10 and Figure 7-11 show no discernible relationship between powder factor and cut advance and scaling time, though both sets of data show the most consistent performance with powder factors in the range of 2.5kg/m^3 and 2.7kg/m^3.

Where an increase in powder factor does not result in an increase in cut advance, it represents a wastage of explosive, consumables, and time. This region is highlighted in Figure 7-10. Likewise, where an increase in powder factor does not result in a reduced scaling time at the face, there is once again a waste of explosive, consumables, and time. This is highlighted in Figure 7-11.

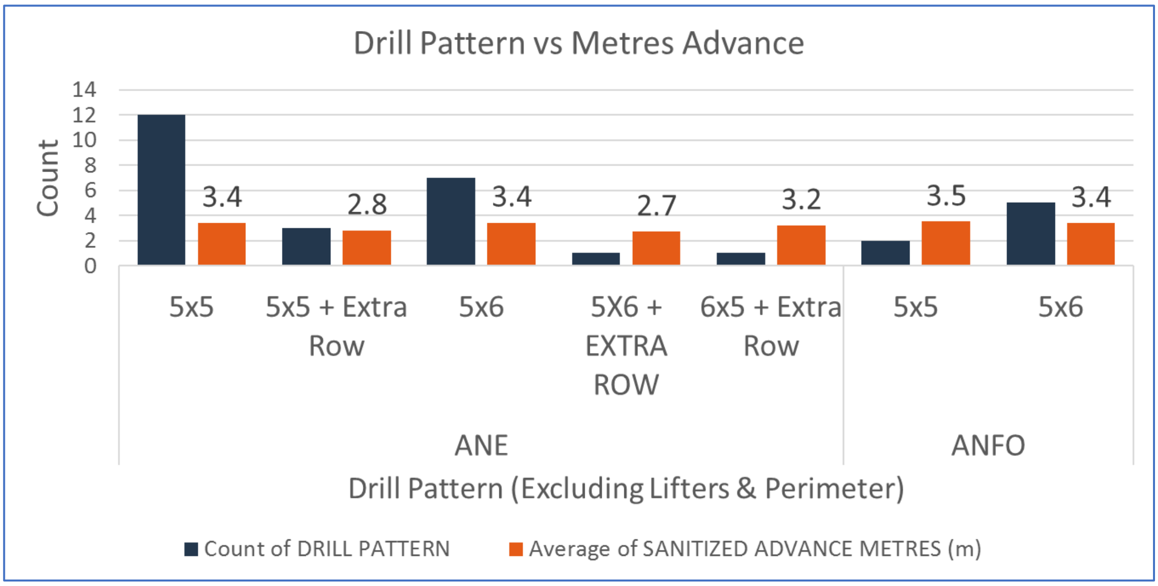


Figure 7-8 - Drill Patterns used vs Metres Advance in the 520 EXP Drive

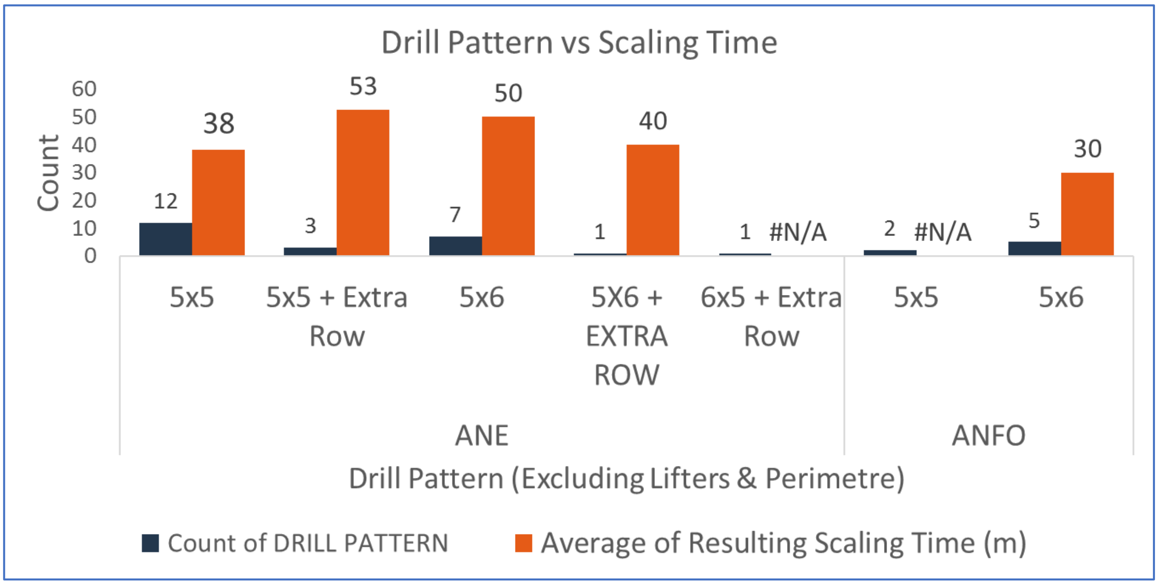


Figure 7-9 - Drill Patterns used vs Scaling Time in the 520 EXP Drive

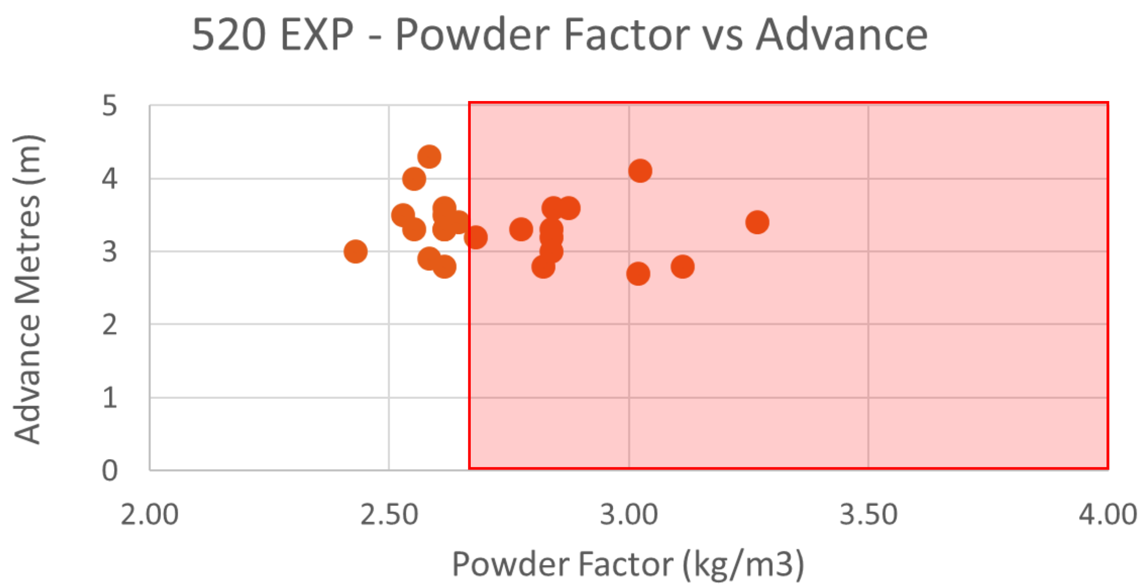


Figure 7-10 - Powder Factors used in cuts taken in the 520 EXP (with wastage highlighted)

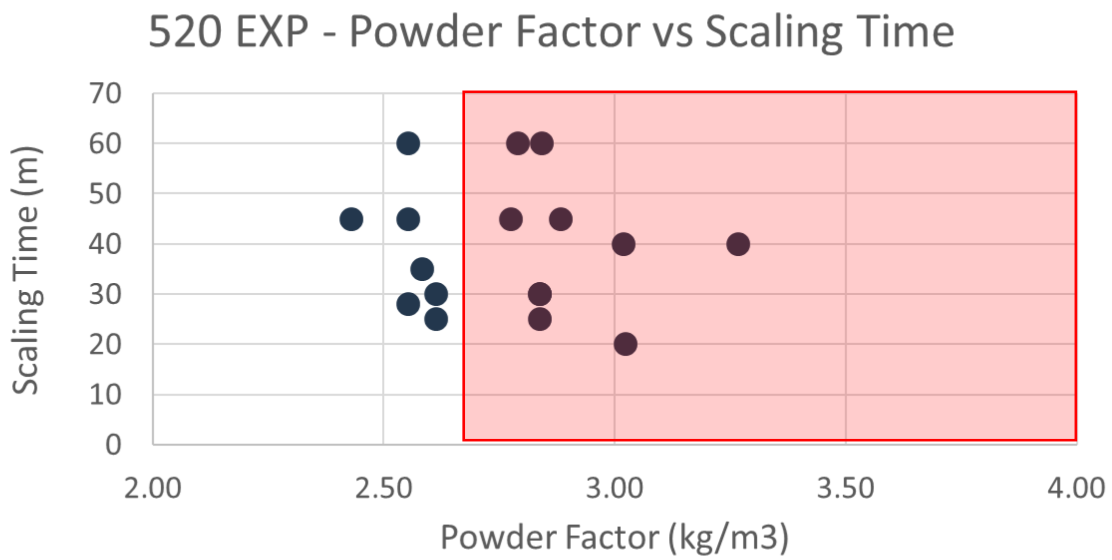


Figure 7-11 - Powder Factor and resultant scaling time in the 520 EXP (with wastage highlighted)

7.7 PRIMER SELECTION

At Agnew if ANFO is used as the bulk explosive, Orica's Senatel Magnum 32x200mm powergel ("pg") is selected as the primer. If emulsion is used as the bulk explosive, the operators select either Orica's D Primers (25g Pentolite) or Orica's G Primer (110g Pentolite). Typically G Primers were used by chargeup at the request of jumbo operators who believed a larger cast booster would result in improved performance of the cut. Sometimes the instruction was to use G Primers in the burn, but not the rest of the face.

Figure 7-12 shows the average advance of the cuts that used various primer types, for different steel sizes. Powergel primers were involved in the best performing cuts for both steel sizes, though since all powergel cuts were charged with ANFO, it is difficult to determine its contribution to the performance of the cut. Interestingly, the cuts that used D Primers outperformed the cuts that used G Primers. On the surface, this is counter intuitive due to the lesser explosive charge weights in each primer. However, this could be due to the D Primers initiating the charge column with a lower VOD than the G Primers (due to the lower level of Pentolite). A lower VOD can lead to a higher heave characteristic (Rock, Maurer & Pereira 2005) which may be more suitable to the ground's existing laminated discontinuities.

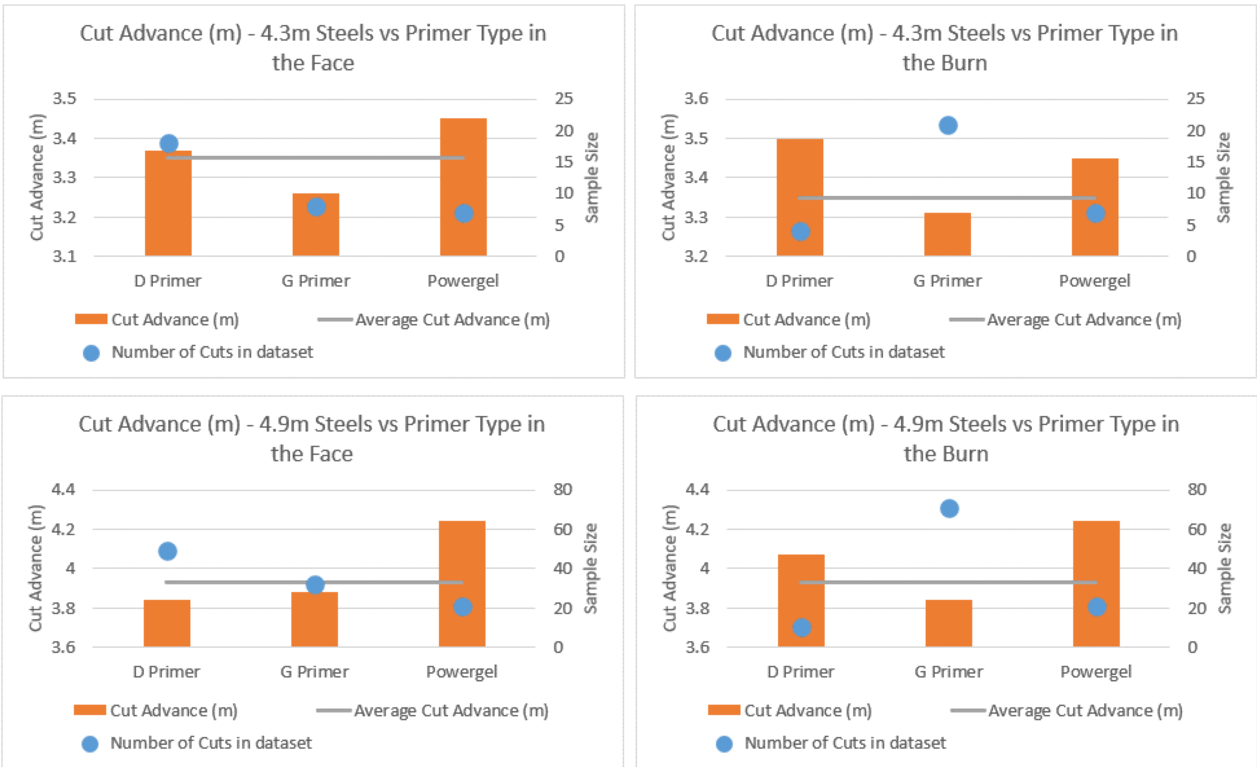


Figure 7-12 - Cut Advance using various Steels and Primers in the Burn or Face Easer holes

7.8 Statistical Analysis of Key Parameters

Outside of accurate drilling, the key parameters identified in Table 7-2 are considered the most likely to impact cut performance. Data can be interpreted subjectively, so classical statistical analysis can be utilised to remove any bias or subjectivity from the data interpretation. In this section various hypothesis are tested using classical analytical methods. Hypothesis tests were conducted using the entire data set, with 4.3m and 4.9m cuts being considered separately, and the key test parameters considered were Bulk Explosive and Geology.

The analysis used a combination of hypothesis testing tools –

1. 2 Sample F-Test for variance
2. 2 Sample T-Test assuming equal variance
3. 2 Sample T-Test assuming unequal variance

The details of the hypothesis tests are provided in Appendix G, and the results are shown below in Table 7-6 and Table 7-7.

ROUND LENGTH	FOCUS AREA	There is enough evidence to suggest….	Confidence Interval
4.9m	GEOLOGY	Emulsion advance metres will be greater in non-parallel ground than in parallel ground.	95%
	GEOLOGY	ANFO advance metres will be greater in non-parallel ground than in parallel ground.	85%
	GEOLOGY	Scaling time will be longer in parallel ground than in non-parallel ground.	95%
	BULK EXPLOSIVE	ANFO will result in greater cut lengths than emulsion cuts.	95%
	BULK EXPLOSIVE	ANFO will result in less variance in advance metres than emulsion cuts.	95%
	BULK EXPLOSIVE	ANFO will average 0.2m advance more than emulsion.	90%
	BULK EXPLOSIVE	ANFO will result in lower scaling time than emulsion cuts.	90%

Table 7-6 - Hypothesis Test Results for cuts taken with a 4.9m Steel

ROUND LENGTH	FOCUS AREA	There is enough evidence to suggest….	Confidence Interval
4.3m	BULK EXPLOSIVE	ANFO will result in lower scaling time than emulsion cuts.	95%
	BULK EXPLOSIVE	In 4.3m Rounds, ***there is NOT ENOUGH evidence at 90%*** confidence that using ANFO will result in greater cut lengths than emulsion cuts.	90%

Table 7-7 - Hypothesis Test Results for cuts taken with a 4.3m steel

Notes:

1. There was only 85% confidence in the second geological hypothesis due to the low number of samples.
2. The only drive that used 4.3m steels was the 520 EXP.
3. It is possible that both ANFO and emulsion are approximately equally effective at moving 4.3m. The reason ANFO performs better than emulsion in a 4.9m round could be due to emulsion not being effective in performing mechanical work ('heave') on an additional 20m3 (~60t) of rock. It may also require more data to improve the confidence interval.

7.9 Results Discussion with regard to established rock breakage theory

It is important to review the data and relate the results back to established rock breakage theory to strengthen the understanding of the results.

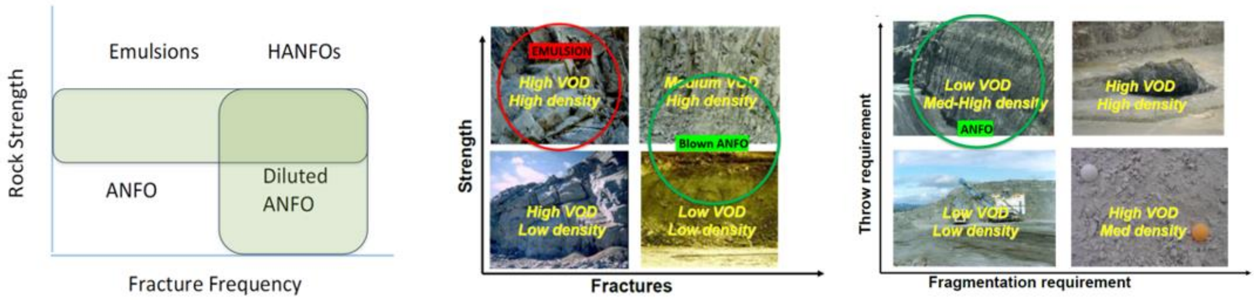
The data has provided some insights as to opportunities for cost-reduction, and that in general ANFO is the better performing bulk explosive product to use in the Waroonga ground type shown in Figure 7-14, both in terms of cut advance, scaling time and consistency in results. These results align with established explosive selection theory discussed below.

7.9.1 Bulk Explosive Selection with respect to ground conditions

The 520 EXP drive is within the Scotty Creek Sandstone formation, which has an average Unconfined Compressive Strength (UCS) of 140-220 Mpa (Moulding et al, 2017), which is a moderate-to-strong rock type. The fracture frequency of the ground at Waroonga is considered high due to the high level of laminations evident in the strata. According to Fidler's (2009) work shown in Figure 7-13, an appropriate explosive for this ground type would be a denser, medium-VOD ANFO-based product, like a Heavy ANFO mix or blow-loaded ANFO at 0.9g/cc density.

Furthermore, the rock type described as being best suited to medium VOD and high density in Figure 7-13(b) is similar to the rock type observed throughout the Waroonga mine (Figure 7-14), particularly in the 520 EXP. To prevent the cut choking and hanging up on the face, it is ideal to have the rock be thrown as far as possible to provide sufficient void for the rest of the rock to break into.

Figure 7-13 describes a similar requirement as well as providing an image of similar looking rock to the laminated scotty screek sandstone. Since a higher density is achieved by blow-loading the ANFO during development charging (which achieves a 0.9g/cc density), the results in Figure 7-12 align with Fidler's graphs in Figure 7-13.



(a) (b) (c)

Figure 7-13 – Bulk Explosive Selection considerations. (Fidler, 2009)

Figure 7-14 – Near-Vertical Laminations in the Waroonga Kim Lode

8 DESIGN RECOMMENDATIONS

Once the initial data has been collected and analysed alongside existing rock breakage theory, decisions can be made on potential blast design improvements. In order to minimize sources of error and reduce variability of the cuts, it is a good practice to ensure that the design changes are tested in at least one long, consistent and continuous drive in order to ensure the ground conditions are as similar as possible.

At Waroonga, the 520 EXP drive was selected to monitor any performance improvements due to its long, straight nature through the vertically laminated ground. The following blast design recommendations were made and built into a standardized pattern, specific to Agnew's drive sizes (5.0mWx5.0mH to 6.0mWx6.0mH) that was rolled out to all jumbo and charge-up operators (Appendices H-K).

As part of the hierarchy of considerations for blast optimization, each parameter was considered for its specific role in the blast design, and a decision made based on operational data, quantitative and empirical analysis, as well as cost considerations.

8.1 BULK EXPLOSIVE SELECTION

Based on the results of the hypothesis testing with regards to scaling time and advance metres, positive operator feedback and consideration for established rock blasting theory, the recommended bulk explosive to implement in the standardised blast design is a low VOD, medium density product. The available product that has these characteristics is ANFO.

Perimeter control should remain a priority, so the lower density products designed for perimeter blasting is recommended. Orica's LD50/50 is available in this instance.

8.2 CONSIDER PREFERENTIAL CRACK PROPAGATION

As discussed earlier, the crack formation caused during the initial fracture phase of the blast-induced rock breakage process is hindered by pre-existing cracks and discontinuities. The crack formation reduces once it hits a discontinuity, and in laminated ground this means that the fractures will extend further in the orientation of the laminations rather than perpendicular to the laminations (illustrated in Figure 8-1). That is to say that blasting in laminated ground will cause the initial fractures to propagate along existing planes of weakness.

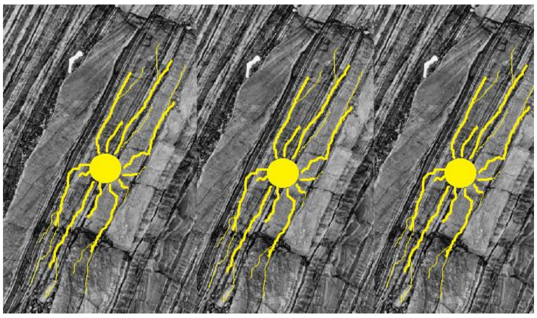


Figure 8-1 - Laminated ground encourages preferential fracturing in the orientation of the laminations.

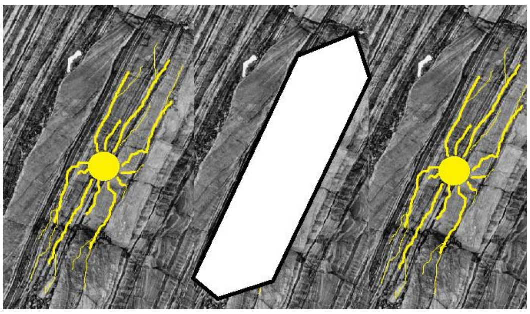


Figure 8-2 - A blasted hole profile in laminated ground will not be circular but rather tend towards being longer in the direction of the joints.

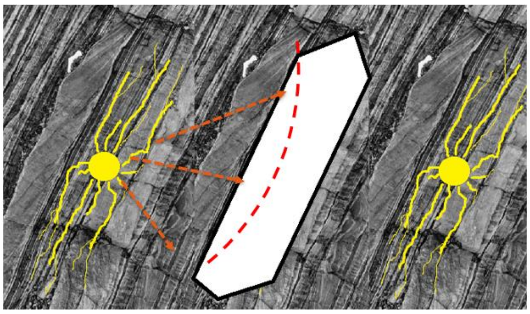


Figure 8-3 - This vertical hole shape provides an opportunity for the adjacent holes to break into the largest freeface taking advantage of flexural tensile stresses.

8.2.1 Perimeter Holes

As is illustrated in Figure 8-4, not drilling enough back holes will increase the probability of a blast resulting in a scalloped profile. This effect is more pronounced in ground where the joints are aligned vertically. In vertically laminated ground, the back holes are firing downwards with the grain (in the direction of the laminations) and the wall holes are firing across the grain (perpendicular to the laminations). Therefore, it is important in this ground to consider that more holes are required in the backs and a larger spacing can be considered for the wall holes.

Some jumbo operators have used 6 holes in the walls and 7 in the backs successfully in the 520 EXP, so this practice will be included in the standard blast design.

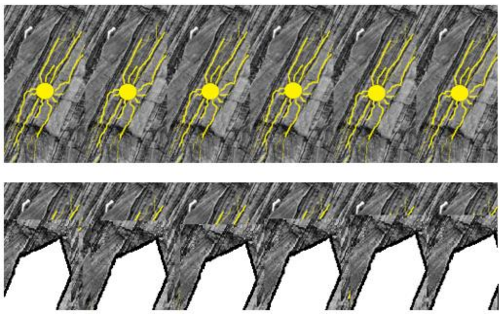


Figure 8-4 - Perimeter (Back) holes in vertically laminated ground have a higher chance of producing a scalloped back profile if the holes are not close enough together (i.e. spacing is too large).

8.2.2 Blast Hole Firing Sequence

A typical firing sequence would include a standard box-diamond pattern that expands outward evenly from the burn, however this may not be the best practice for anisotropic ground. In vertically laminated ground such as the 520 EXP, as discussed earlier, the initial fracture propagation is likely to tend in the direction of the laminations. It is also seen in the data that the better blast results are when the blast is firing perpendicular to the laminations where the freeface allows the greatest flexural stresses.

The dual-shot holes are both fired using Orica's #0 LP detonators (25ms), and the #1 LP is skipped to allow additional time for the initial rock to clear. The next hole to be initiated is primed with a #2 detonator (400ms).

The blast sequence should work to create the largest freeface as soon as possible for the other holes to begin firing across the grain to achieve maximum breakage and throw. To achieve this, the recommended blast sequence will first work to create a vertical opening, and then begin stripping the sides into the void. Stripping is useful to ensure the vector is as perpendicular to the grain as possible, whereas a box-diamond style sequence will cause the throw vector to have some vertical component which is not optimal for vertically laminated ground.

In short, sequence the cut to fire across the grain of the laminations. Since the lamination planes are approximately vertical, maximum fragmentation will occur by blasting in a lateral direction. Avoid stripping rows downward as this is the least efficient use of the heave energy. Only the backs should be stripped downwards as they are the last holes fired as part of the perimeter control. Firing the back holes together simultaneously will also maximise tensile forces along the line of the perimeter holes.

Two blast sequences are illustrated in Figure 8-5 (with the sequence steps provided in Appendix L and Appendix M) that illustrate the opening up of the void in the manner described. Two sequences are provided to account for the safe practice of positioning the burn in a difference area of the face to the previous cut.

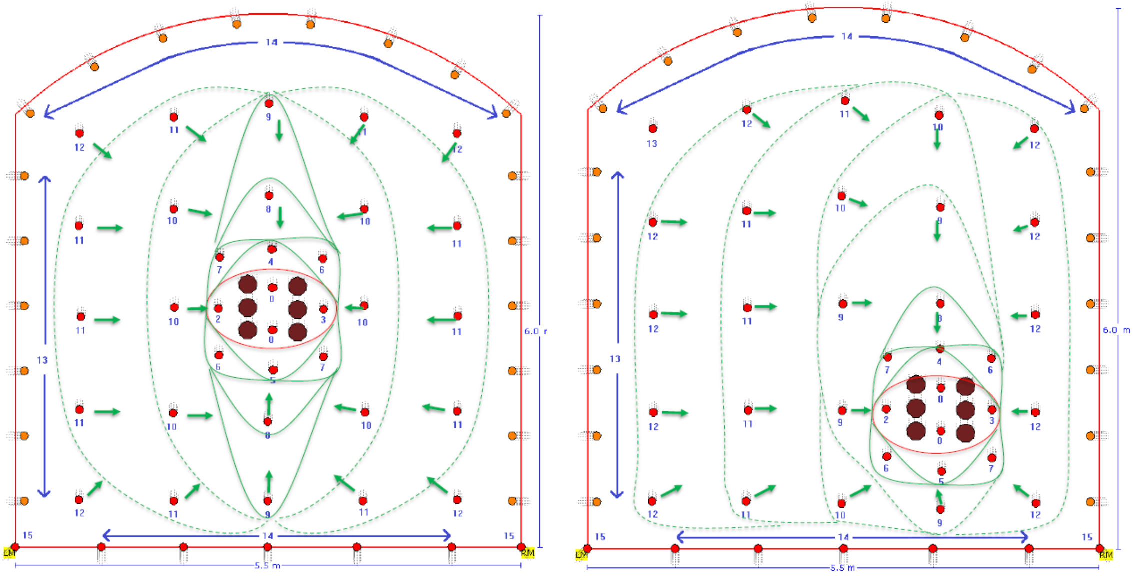


Figure 8-5 - Blast sequences opening up a vertical void (acknowledging the initial preferential fracture mechanism prior to initial void generation) and then then stripping the majority of the cut across the grain of the vertical laminations.

8.3 Steel Length

Since the recommended bulk explosive is ANFO, and the results have shown ANFO to be most effective at taking 4.9m rounds, a 4.9m steel is recommended for the standardised pattern at Agnew.

8.4 Powder Factor

The most effective overall powder factor range was shown to be within 2.3kg/m3 to 2.6kg/m3. This can be achieved using a variety of drill patterns and bit diameters, of which the recommendations are made below.

8.4.1 Drill Pattern

The results show that all of the drill patterns used are more or less getting similar results. Drill patterns that tended towards using more holes (and thus a higher-than-average powder factor) did not result in improved performance. Therefore, based on the most successful and cost-effective operator designs, a 5x5 pattern is recommended using 7 lifters, 2 x 6 wall holes, and 7 back holes.

As mentioned previously, since the ground is vertically laminated, the walls will require less blast holes than the backs.

8.4.2 Bit Diameter

In order to maintain a powder factor within the effective range described above while using the 5x5 pattern, bit diameter of 51mm is to be used in the burn and face holes. 45mm bits are to be used in the perimeter. The smaller bits are selected for use in the perimeter because the role of these holes is to create a clean profile rather than being concerned with advance rate.

8.5 Reamer Configuration

A reamer configuration that is designed to create the largest initial vertical opening is recommended, as it will work best with vertically laminated ground. There is some data to suggest that the "Tall Twin Six" reamer configuration has been involved in the stronger performing cuts. 27 cuts used a basic

“Circle Six” configuration for an average advance of 3.4m, while 2 cuts used a “Tall Twin Six” configuration for an average advance of 3.6m).

This configuration has dual-shot holes (initiating holes) that effectively doubles the local powder factor and explosive energy of the first blast in the sequence, designed to create as much initial void as possible for the rest of the blast to move into. The formation of the reamers also allows the initial shock and heave energy to freely propagate vertically along the lamination planes, increasing the vertical length of the burn opening. The vertical burn opening will provide a greater freeface for the rest of the cut to be fired laterally across the grain of the laminations, as described earlier. The double shot-hole provides a boost in local powder factor to ensure the rock is cleared from the area. A second shot hole also acts as a backup initiation point in case the shot hole misfires during blasting.

8.6 Steel Type

Although the level at which deviation is currently measured is not having a significant impact on advance, it does appear to impact scaling time, and mitigation strategies are always worth considering. Increasing the bit diameter to 51mm while continuing to bore with a 4.9m hex steel will increase the steel-to-borehole annulus and risk additional flex down the steel, increasing the likelihood of in-hole deviation. Therefore a round steel profile is recommended to use for its greater stiffness.

8.7 Economic Considerations & Cost Comparison

A range of ‘typical’ cuts were costed and compared to the cost of the proposed design recommendations. Cost per drill meter was determined for drill consumables (bits and steels), and explosive costs were calculated using existing vendor prices per unit (or per kg) and converted into a cost per metre advance. The optimisation showed a cost saving against a range of exiting ‘operator designs’, despite moving to a 51mm bit and away from emulsion as a default parameter. The proposed design parameters show a 14% cost reduction compared to existing practices. While not specific to advance rate, if the advance rate is shown to improve using the proposed configuration, then the benefits will be two-fold. The costs in Figure 8-6 only considers consumables & explosives, and does not include other fixed cost contributors to the cost of development such as operator wages, fuel consumption, asset depreciation, etc.

BLAST PLAN	"Standard" Cut	520 EXP - UPPER	520 EXP - LOWER	Proposed
Steel	4.9m Hex	4.3 Round	4.3 Round	4.9m Round
Face Diametre	48mm Round	48mm Round	48mm Round	51mm Round
Perimetre Diametre	45mm Round	45mm Round	45mm Round	48mm Round
Bulk Explosive	Subtek Emulsion	Subtek Emulsion	Subtek Emulsion	ANFO
Primer Type	Pentex G	Pentex G	Pentex D	Senatel Magnum (32x200)
Grid Pattern	5x5	5x6	5x5	5x5
Drive Dimensions	5.5 mW x 6 mH	5.5 mW x 6 mH	5.5 mW x 6 mH	5.5 mW x 6 mH
Powder Factor	2.65	2.89	2.63	2.68
Total Consumables Cost	$ 1,270	$ 1,233	$ 988	$ 1,094
Cost per Blasted BCM	$ 8.74	$ 9.83	$ 7.88	$ 7.53
Cost per Metre Advance	**$ 288.57**	**$ 324.40**	**$ 260.00**	**$ 248.57**

Figure 8-6 - Cost Comparison of current practices with the proposed standardised design rules.

9 STANDARDISED PATTERN RESULTS

The jumbo and charge up operators had the initial data, results and analysis presented to them prior to passing on the standardised designs. This encouraged operator “buy in” which assisted in maintaining consistency in the execution of the plans across various crews.

Once the standardised design had been determined and rolled out, data was collected for a further 2 months to measure the success of the design recommendations. Table 9-1 shows results from the standardised pattern design within the test drive 520 EXP. The 520 EXP was selected as the test site for the standardized pattern because the drive was a long straight drive that was expected to be aligned in the direction of the laminations for the duration of its development. This meant a consistent representation of what was considered challenging ground and was a control against unknown geological variables. Though the recommended standard design required the use of ANFO, there were 9 instances where the ANFO charge rig was unavailable (service day / breakdown, etc) and so emulsion was used resulting in poorer performance. However, the same drill parameters were used, and the results observed in Table 9-1 mirror the earlier findings that argument that ANFO is the preferred bulk explosive in this particular ground type.

The average advance for an ANFO cut using a 4.9m steel is slightly less than the initial results, however this is likely due to the larger sample size reflecting a more representative sample of data. The sample size in the same drive in the initial results was 12, whereas the sample size is 37.

Steel Length	Explosive Type	Cuts Taken	Average Advance (m)	% Ideal Cut Length
4.9m (Ideal advance 4.4m)	ANE	9	3.6	82%
	ANFO	33	4	91%
	Total	**42**	**3.9**	**89%**

Table 9-1 – Bulk Explosive performance in the 520 EXP with standardised drill pattern.

9.1 Discussion

Early results in Table 7-3 establish that the operation was achieving an average of 3.7m advance per cut by allowing jumbo operators to drill according to their own design & preferred consumables (e.g. different steels, bit sizes, patterns, reamer formations, etc), and the charge up operators to select which bulk explosive was used based on convenience (e.g. proximity of charge rig). By standardizing the drill and blast practices, the operation was able to increase average advance by 0.3m advance per cut and reduce the standard deviation of the blast results by approximately 25% (shown in Table 9-2).

		Cuts	Ave Adv (m)	Std Dev (m)
Cuts taken using all steel lengths	All Cuts Prior to Standard Design	84	3.7	0.564
Cuts taken using 4.9m rounds	Prior to using Standardised Design	58	3.9	0.526
	Using Standardised Design	**33**	**4.0**	**0.394**

Table 9-2 - Initial Results vs Standardised Pattern

In an operation with a target of 160 cuts per month, this represents 48m additional advance per month. To an owner-operated mine, this means significant improvement in the mine schedule. To a contract-miner, this means a significant step change in monthly revenue, reduction in rework and more efficient use of the fleet due to longer steel lengths utilised in the design. Using an arbitrary revenue figure of $2000/m advance (based on industry standard and without consideration for ground support regimes and drive sizes), this represents a potential gain of $96,000 revenue per month.

While the average advance of the standardised pattern (Table 9-1) was not shown to be higher than the initial results for 4.9m Steel + ANFO cuts (Table 7-3), there is a far greater sample size with the standardized cuts (37 samples) vs the initial results with this blast design configuration (5 cuts). Further, there is confirmation that the ANFO continues to perform better in highly laminated ground than emulsion. By optimising the blast pattern and implementing a standard, an estimated cost saving of $40/mAdv was achieved (Figure 8-6). Across the 37 cuts the total development was 158m total development. This includes 5 cuts that did not have chainage data due to repositioning of the laser and were estimated at 4.0m advance.

The consumables cost saving over 158m represents a saving of $6,320 just for the cuts taken in the 520 EXP. In an operation that achieves aims for 160 cuts per month (approx. 640m) this results in an approximate saving of $25,600 in development drill and blast consumables per month. For a net benefit to the contractor of $121,600/month if the standards can be maintained through appropriate education, supervision, and accountability.

10 CONCLUSION

In the mining industry, often decisions need to be made rapidly based upon the experience and general knowledge of the management team making the decision. Whilst it is often appropriate to make a quick decision due to operational requirements and constraints, when it comes to solving problems of a grander scale, often a more considered approach is more appropriate. A holistic, pragmatic approach considers general knowledge but also leans into empirical data, technical knowledge and data driven decision making.

The proposed Hierarchy of Considerations for Optimised Blast Design has shown to be effective in producing an improved development blast design at Agnew.

By first reviewing the literature and undertaking a systematic approach to understanding the challenge presented at Agnew, undertaking both quantitative and empirical data analysis, and investigating cost implications of each decision, an optimised blast design was generated. The design was able to deliver an average cut length 0.3m greater than the overall operational average whilst simultaneously reducing the standard deviation by 25%, and driving a cost reduction of drill and blast consumables of approximately 15% per metre advance.

Whilst data driven decision making was used to arrive at some design decisions, established blast theory was also used to verify the data and strengthen the decision. For example, choosing ANFO over Emulsion in the laminated geology due to their respective explosive properties and relevance to the geology. This contrasted with the explosive supplier's insistence that emulsion was the preferred bulk explosive and that increasing the powder factor could solve the problem.

Further, where data was unable to provide a tangible result or measurement, blast theory provided a base with which to make an informed, reasonable decision on parameter selection. For example, with the sequencing of the blast holes, consideration was given to the anisotropic properties of the laminated ground. Where both the data and blast theory were unable to provide an insight into optimised selection of parameters, cost analysis was used to drive the decision – for example where additional holes blast holes did not appear to lead to improved performance, they were removed to conserve costs and improve cycle time.

Learnings from this study implicate the importance of a robust data capture system designed to measure the appropriate design parameters, a sound understanding of established literature, and a holistic approach to combining the various sources of knowledge and information available to make an informed, reasonable, and practical decision.

The underground industry could benefit by applying this pragmatic approach to other mines that exist in different geological and geotechnical settings.

11 FURTHER STUDY

Unfortunately, the volume of research targeting underground drill and blast optimisation that directly relates to development advance is rather thin, in that there is not a significant amount of work that has been done to address the challenges in ensuring cuts pull to full length. Some publicly available case studies talk to adjusting single parameters to optimise a blast design (e.g. changing the Bulk Explosive type), or talk to reducing overbreak, but it was difficult to find sources that talk to holistic optimisation techniques.

Blasting rules of thumb are available in supplier handbooks and SME handbooks however they are primarily for initial blast design and do not assist in refining the blast design based on available data.

The hierarchy of considerations for optimised blast design is not just suitable for underground development advance, but also in underground production blasts, as well as open pit drill and blast optimisation. The industry would benefit from further case studies implementing the hierarchy of considerations for optimised blast design in these environments.

In Chapter 6 of this paper, a CTQ analysis was conducted which identified that adhering to the mine's budgeted development schedule required both Successful Cuts and Rapid Cut Turnover. While this research focused on generating a successful cut based on the hierarchy of considerations for optimised blast design, there is still room for research into the organisational and team behaviours that drive rapid cut turnover, colloquially known as "high speed development." Researching the nuances of the underground development cycle to determine which behaviours are conducive to a reduction in operational delays and reduction in cycle time whilst simultaneously increasing the utilisation of the jumbo and haulage fleet would be of significant value to an underground company, in particular a contract miner. From an organisational perspective, ensuring the machines availabilities are maximised and ensuring systems are developed to allow maximum utilisation of the machines is crucial. For example – in a mine that is capable of sustaining sufficient broken production stocks such that it is not critical that production blasts occur at the end of every shift, restricting blasting to the end of nightshift (and no longer blasting at the end of dayshift) opens up an additional 2 hour work window between dayshift and nightshift which allows a hot-seating arrangement to capture an additional 2 hours of machine utilisation every 24 hours. Team based behaviours required to facilitate an effective hot seating setup require the appropriate man-run vehicles being made available, and pre-start meeting times adjusted slightly such that the incoming crew can attend the pre-shift meeting and get to their counterparts in time without delay. Independent mid-shift firing can also be implemented to bring forward a development cycle so that the development headings that sit below the return air rises do not need to wait for the 24 hour "nightshift only" blasting cycle. The mechanics of these effective systems do not exist in literature and the 'best practice' is commonly transferred as a 'word of mouth' knowledge. Without a structured system or literature to support it, it is difficult for garner team buy-in and overcome any initial scepticism relating to these systems of work and behaviours. Micro-behaviours (such as a bogger operator returning the pump to the decline face so that it is already in position for the jumbo to arrive at the face and get started immediately) are another area of team-based behaviours that would be worth capturing as a list of organisational and team-based behaviours critical to high speed development.

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APPENDIX A

FACE MARKUP SHEET - FRONT

Barminco

HEADING ____________________

DRILLER - See Back of Sheet Also

Date	
SHIFT	
MEMO #	

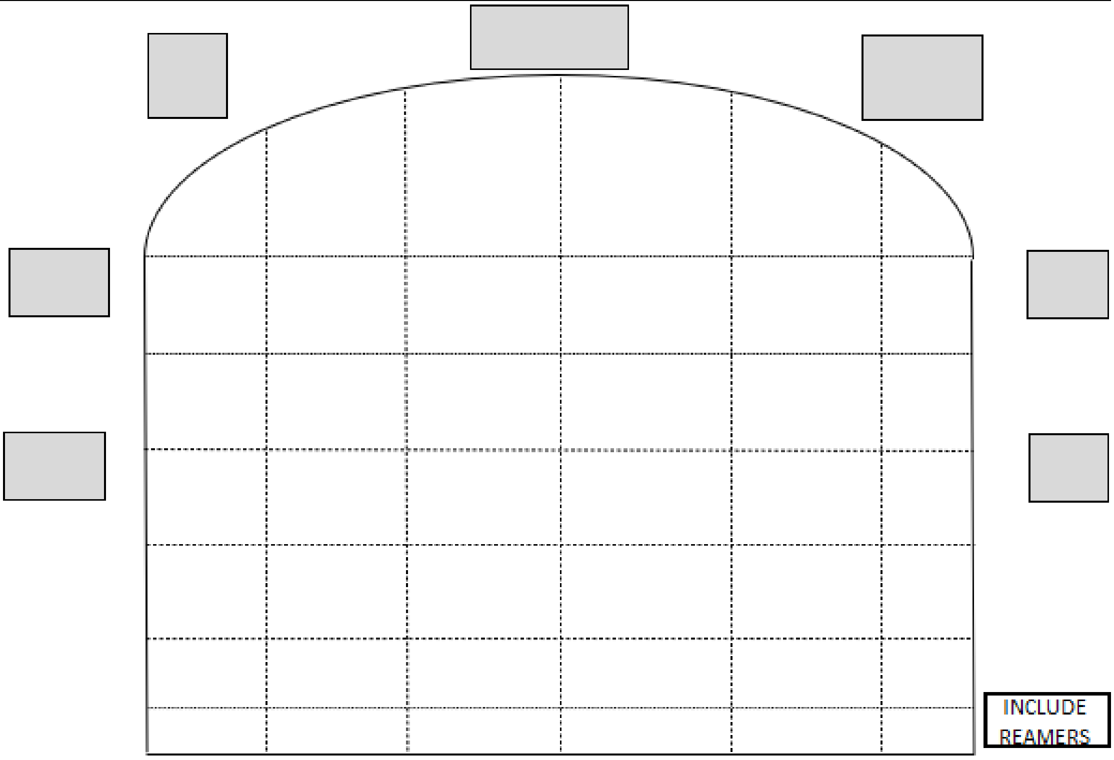
DRILLERS NAME #1	
DRILLERS NAME #2	
DIMENSIONS	X

GSS	
# HOLES DRILLED	
CHAINAGE	

DISTO MEASURE FROM CAB	
BEFORE SCALING	
AFTER SCALING	
SCALING TIME	
SCALING TIME	Minutes

	ACTUAL	COMMENTS
HEIGHT		
LIFTERS		
LEFT		
RIGHT		

FACE MARK UP



INCLUDE REAMERS

SHOTFIRERS

Date		Shift	DS / NS

SHOTFIRER'S NAME	
ASSISTANT'S NAME	
HOTFIRER SIGNATU	

PRIMERS			
BURN	D	G	Powergel
FACE	D	G	Powergel
PERIMETE	D	G	Powergel

DENSITY					
GASSER	0.6	0.8	1.0	1.1	1.2
CUP					g / L
TEMP					°C

SHIFTBOSS NAME	
SIGNATURE	

EXPLOSIVE USAGE

DETS	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	Elec.
QUANTITY																	

ANE (kg)	ANFO (kg)	SONOL (kg)	PRIMER	G PRIMERS	X700 POWERG	X200 POWERG	DET CORD (m)	HOLES FIRE

FACE MARKUP SHEET – BACK

Barminco

DRILLER

STEEL - LENGTH	4.3m	4.9m		
STEEL - TYPE	HEX	ROUNDED		
BIT TYPE	BALLISTIC	ROUNDED	CROSS BIT	
DIAMETRE - BURN	43mm	45mm	48mm	51mm
DIAMETRE - FACE	43mm	45mm	48mm	51mm
DIAMETRE - PERIMETER	43mm	45mm	48mm	51mm

Average FEED PRESSURE	bar
verage ROTATION PRESSUR	bar

DRIVE DIRECTION **vs** **LAMINATIONS**	RUNNING WITH RUNNING ACROSS OTHER	NOT LAMINATED

APPENDIX B

Drill Hole Deviation Sheet

Drill Hole QA Sheet

HEADING ____________ **DATE DRILLED** ____________ **SHIFT DRILLED** D / N

	BURN					
	HOLE	**DEPTH**	**TILT**	**BEARING**	**DEVIATION OBSERVED**	**DEV. DIRECTION**
		(metres)	(vertical degrees)	(horizontal degrees)		(Relative to laminationss)
BURN CUT	SHOT HOLE				0 1 2 3	Follows Across Both
	2				0 1 2 3	Follows Across Both
	3				0 1 2 3	Follows Across Both
	4				0 1 2 3	Follows Across Both
	5				0 1 2 3	Follows Across Both
	6				0 1 2 3	Follows Across Both
	7				0 1 2 3	Follows Across Both
	8				0 1 2 3	Follows Across Both
	9				0 1 2 3	Follows Across Both
REAMERS	1				0 1 2 3	Follows Across Both
	2				0 1 2 3	Follows Across Both
	3				0 1 2 3	Follows Across Both
	4				0 1 2 3	Follows Across Both
	5				0 1 2 3	Follows Across Both
	6				0 1 2 3	Follows Across Both
	7				0 1 2 3	Follows Across Both
	8				0 1 2 3	Follows Across Both

DEVIATION SCORES	MEANING	OBSERVATION
0	No Deviation	LED Light still visible, Hole appears to be straight
1	Slight deviation	LED Light seen to be off center somewhat
2	Notable Deviation	LED Light not visible, ambient light is still highly visible
3	Excessive Deviation	LED Light not visible, ambient light not readily visible / Broken through to another hole

Follows	The deviation follows the joints of the laminations
Across	The deviation cuts across the laminations
Both	A mix of the above

APPENDIX C

Face Markup Sheet Data Input Form

Barminco

CUT ID	520 EXP-43201-D
HEADING	520 EXP

That Charge Plan has not been entered yet and you are free to input the data.

CHARGE PLAN

DRILL PATTERN	5x6	<-- Determine from operator sketch
REAMER FORMATION	CIRCLE SIX	<-- Determine from operator sketch
LEVEL	520	
HEADING SUFFIX	EXP	
DATE DRILLED	11/04/2018	
SHIFT DRILLED	D	
DRILLERS NAME 1	MICK	
DRIVE DIMENSIONS	5.5X5.8	
CHAINAGE	33.7	
ACTUAL HEIGHT	5.8	
LIFTER HEGHT	1.5	
LEFT	4	
RIGHT	0.4	
AVE. BUTT DEPTH	0.2	
DATE FIRED	11/04/2018	
SHIFT FIRED	N	
BURN PRIMERS	G	
FACE PRIMERS		
PERIMETER PRIMERS		
BULK EXPLOSIVE		
PRODUCT DENSITY		
ANE		
ANFO		
ISONOL		
# PENTEX D		
# PENTEX G		
# 32x700 POWERGEL		
# 32X200 POWERGEL		
# FIRED HOLES		
STEEL LENGTH		
STEEL TYPE		
BIT TYPE		
DIAMETRE - BURN		
DIAMETRE - FACE		
DIAMETRE - PERIMETER		
AVE FEED PRESSURE		
AVE ROTATION SPEED		
DRIVE/GEO ORIENTATION		

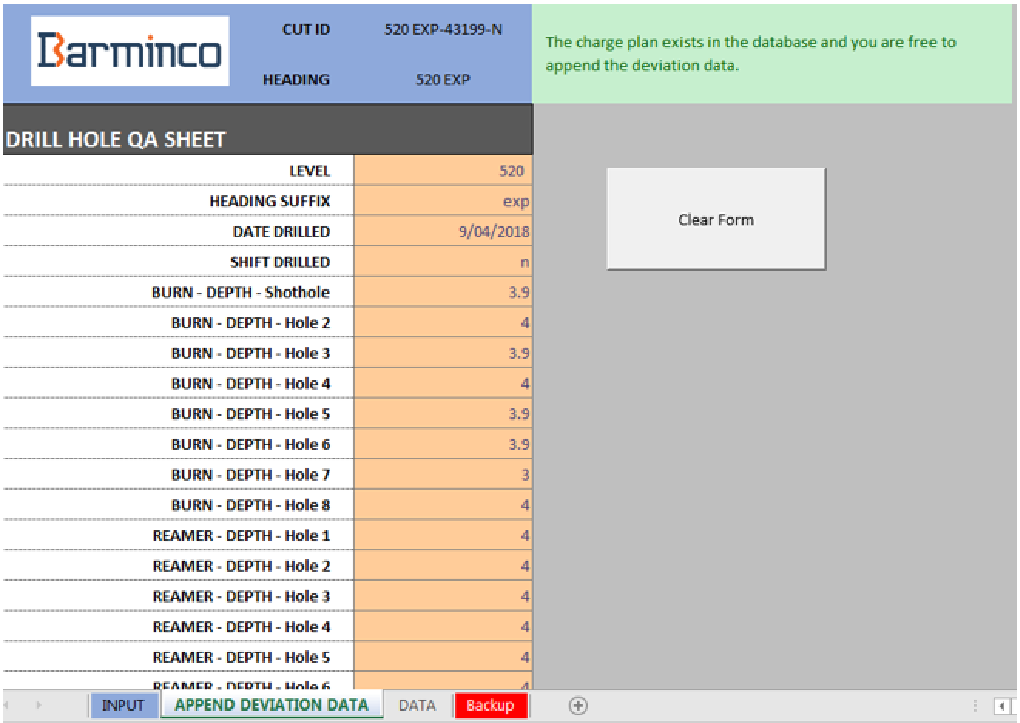
Clear Form

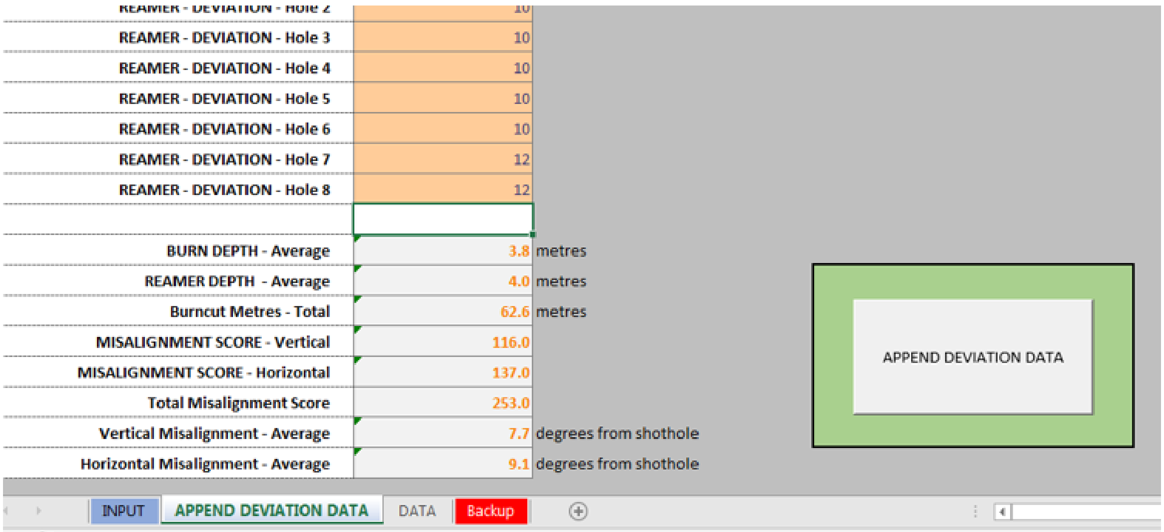
If there is no Drill Hole QA data...

STORE CUT DATA

Otherwise, complete the entire form...

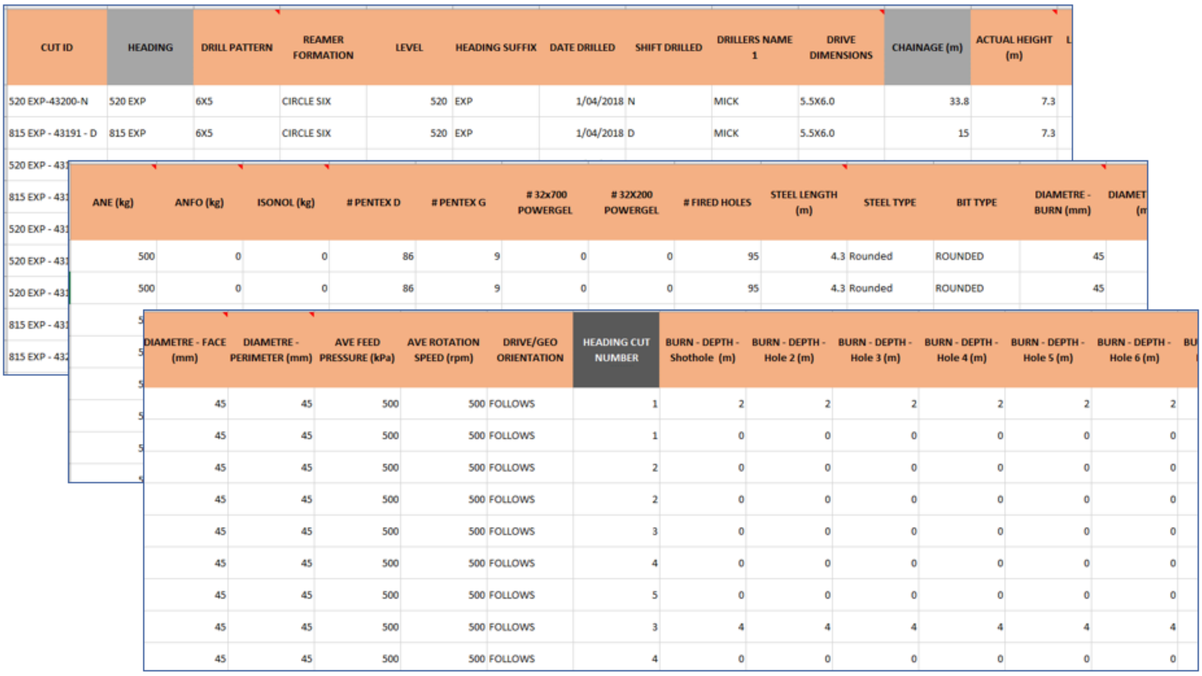
In-Hole Deviation Data Input Form





APPENDIX D

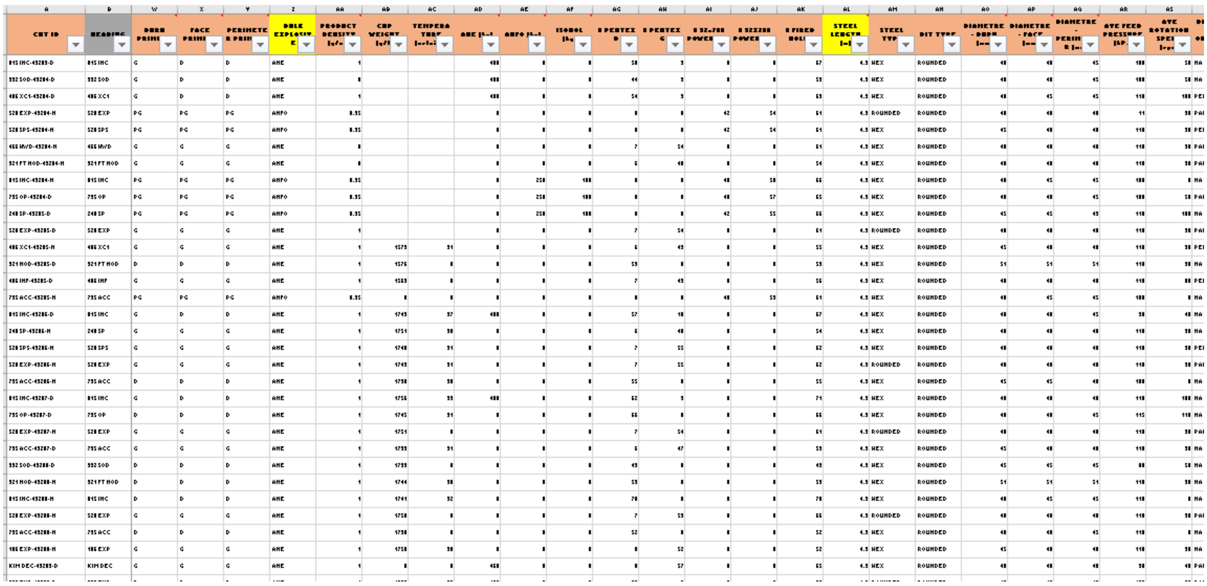
Snapshot of the Database



CUT ID	HEADING	DRILL PATTERN	REAMER FORMATION	LEVEL	HEADING SUFFIX	DATE DRILLED	SHIFT DRILLED	DRILLERS NAME 1	DRIVE DIMENSIONS	CHAINAGE (m)	ACTUAL HEIGHT (m)	L
520 EXP-43200-N	520 EXP	6X5	CIRCLE SIX	520	EXP	1/04/2018	N	MICK	5.5X6.0	33.8	7.3	
815 EXP - 43191 - D	815 EXP	6X5	CIRCLE SIX	520	EXP	1/04/2018	D	MICK	5.5X6.0	15	7.3	
520 EXP - 431												
815 EXP - 431												
520 EXP - 431												
520 EXP - 431												
520 EXP - 431												
815 EXP - 431												
815 EXP - 432												

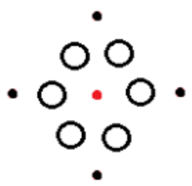
ANE (kg)	ANFO (kg)	ISONOL (kg)	# PENTEX D	# PENTEX G	# 32x700 POWERGEL	# 32X200 POWERGEL	# FIRED HOLES	STEEL LENGTH (m)	STEEL TYPE	BIT TYPE	DIAMETRE - BURN (mm)	DIAMET (m
500	0	0	86	9	0	0	95	4.3	Rounded	ROUNDED	45	
500	0	0	86	9	0	0	95	4.3	Rounded	ROUNDED	45	
5												
5												
5												
5												
5												

DIAMETRE - FACE (mm)	DIAMETRE - PERIMETER (mm)	AVE FEED PRESSURE (kPa)	AVE ROTATION SPEED (rpm)	DRIVE/GEO ORIENTATION	HEADING CUT NUMBER	BURN - DEPTH - Shothole (m)	BURN - DEPTH - Hole 2 (m)	BURN - DEPTH - Hole 3 (m)	BURN - DEPTH - Hole 4 (m)	BURN - DEPTH - Hole 5 (m)	BURN - DEPTH - Hole 6 (m)	BU
45	45	500	500	FOLLOWS	1	2	2	2	2	2	2	
45	45	500	500	FOLLOWS	1	0	0	0	0	0	0	
45	45	500	500	FOLLOWS	2	0	0	0	0	0	0	
45	45	500	500	FOLLOWS	2	0	0	0	0	0	0	
45	45	500	500	FOLLOWS	3	0	0	0	0	0	0	
45	45	500	500	FOLLOWS	4	0	0	0	0	0	0	
45	45	500	500	FOLLOWS	5	0	0	0	0	0	0	
45	45	500	500	FOLLOWS	3	4	4	4	4	4	4	
45	45	500	500	FOLLOWS	4	0	0	0	0	0	0	

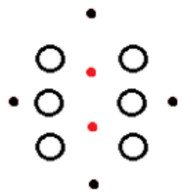


APPENDIX E

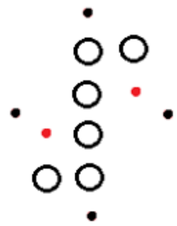
REAMER FORMATION NAMING CONVENTION



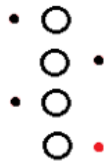
CIRCLE SIX



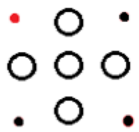
TALL TWIN SIX



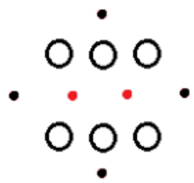
TALL INDO



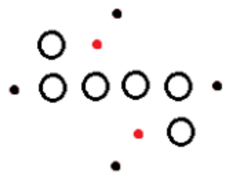
TALL FOUR



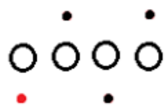
CROSS FIVE



FLAT TWIN SIX



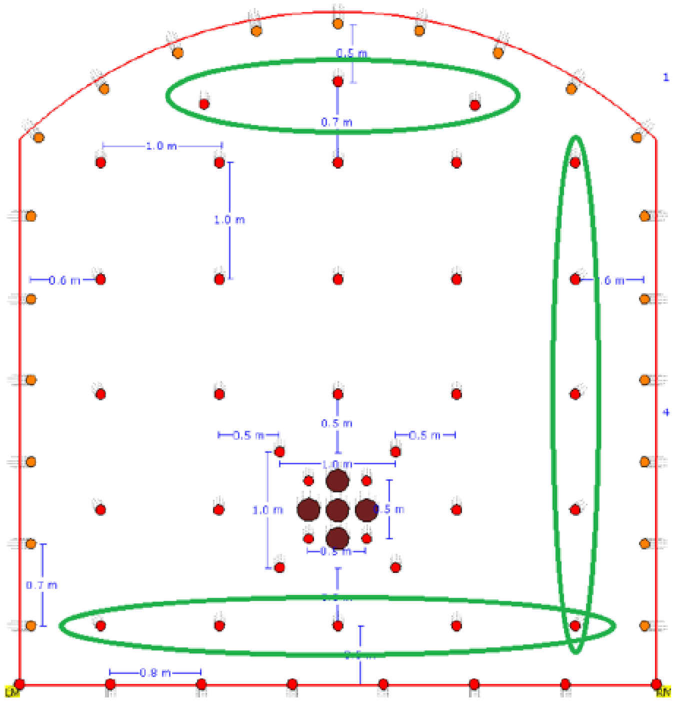
FLAT INDO



FLAT FOUR

APPENDIX F

DRILL PATTERN NAMING CONVENTION



For the purpose of this study, drill patterns were defined by characterizing the hole-grid inside the perimeter holes (ignoring both perimeter and lifter holes).

HOLES ACROSS by HOLES UP

The example shown is referred to as

5 x 5 + Extra Row

Since the inside grid is 5 holes across and 5 holes up, with an extra small row above the main grid (just below the backs).

APPENDIX G

Cut Advance - ANFO vs Emulsion - 4.9m Steels

ANFO	ANE
4.5	3.9
4.4	3.6
4.6	4.2
3.9	4.4
4.1	4.2
3.9	4.1
4.1	3.4
4	2.9
4.7	4.5
3.5	4.6
4.5	4.4
4.4	3.1
	4.4
	4.9
	3.5
	4
	3.3
	3.8
	2.7
	4.2
	4
	3.4
	3.2
	4.3
	3.9
	4.3
	4.1
	4.4
	3.9
	3.36
	4.5

Fixed Variables: 4.9m steels

F-Test Two-Sample for Variances

H0: The cut length variance between ANFO and ANE cuts is the same

Ha: The cut length variance between ANFO and ANE cuts is unequal

Alpha = 0.05

	ANE	*ANFO*
Mean	3.811086957	4.216666667
Variance	0.360454348	0.126969697
Observations	46	12
df	45	11
F	2.838900591	
P(F<=f) one-tail	0.032417368	
F Critical one-tail	2.517449882	
Reject H0?	YES	YES

Interpretation: There is enough evidence to suggest (at 95% confidence) that using ANFO will result in less variance than ANE cuts.

t-Test: Two-Sample Assuming Unequal Variances

H0: The average advance metres between ANFO and ANE cuts is the same

Ha: The average advance metres is greater in ANFO cuts than ANE cuts

Alpha = 0.05

	ANFO	*ANE*
Mean	4.216666667	3.811086957
Variance	0.126969697	0.360454348
Observations	12	46
Hypothesized Mean Difference	0	
df	29	
t Stat	2.988611497	
P(T<=t) one-tail	0.002828794	
t Critical one-tail	1.699127027	

3.2	P(T<=t) two-tail	0.005657589	
3.5	t Critical two-tail	2.045229642	
4.25	**Reject H0?**	YES	YES
3.2	**Interpretation:**	There is enough evidence to suggest (at 95% confidence) that using ANFO will result in greater cut lengths than ANE cuts.	
3.8			
2			
4.5			
2.9			
3			
4.2			
3.6			
3.6			
3.8			
3.8			
4.5			

Cut Advance - ANFO vs Emulsion - 4.3m Steels

ANFO	ANE
3.5	3.2
3.325	3.6
3.325	3.3
3.15	3.4
2.95	3.3
4.1	3.6
	2.9
	4
	3.6
	2.7
	2.8
	3.4
	3
	2.8
	3.65
	2.8
	3.45
	4.3
	3.3

Fixed Variables: 4.3m steels

F-Test Two-Sample for Variances		
H0: The cut length variance between ANFO and ANE cuts is the same		
Ha: The cut length variance between ANFO and ANE cuts is unequal		
Alpha =	0.05	
	ANE	*ANFO*
Mean	3.321052632	3.391666667
Variance	0.182587719	0.155166667
Observations	19	6
df	18	5
F	1.176719995	
P(F<=f) one-tail	0.467373708	
F Critical one-tail	4.578534157	
Reject H0?	NO	NO
Interpretation:	There is not enough evidence to suggest (at 95% confidence) that the variances are unequal.	

F-Test Two-Sample for Variances		
H0: The cut length variance between ANFO and ANE cuts is the same		
Ha: The cut length variance between ANFO and ANE cuts is unequal		
Alpha =	0.1	
	ANE	*ANFO*
Mean	3.321052632	3.391666667
Variance	0.182587719	0.155166667
Observations	19	6
df	18	5
F	1.176719995	
P(F<=f) one-tail	0.467373708	
F Critical one-tail	3.217233991	

Reject H0?	NO	NO
Interpretation:	There is not enough evidence to suggest (at 90% confidence) that the cut length variances between ANFO and ANE cuts are unequal.	

t-Test: Two-Sample Assuming Equal Variances		
H0: The average advance metres between ANFO and ANE cuts is the same		
Ha: The average advance metres is greater in ANFO cuts than ANE cuts		
Alpha =	0.1	
	ANFO	*ANE*
Mean	3.391666667	3.321052632
Variance	0.155166667	0.182587719
Observations	6	19
Pooled Variance	0.176626621	
Hypothesized Mean Difference	0	
df	23	
t Stat	0.358794164	
P(T<=t) one-tail	0.361510852	
t Critical one-tail	1.31946024	
P(T<=t) two-tail	0.723021704	
t Critical two-tail	1.713871528	
Reject H0?	NO	NO
Interpretation:	There is not enough evidence to suggest (at 90% confidence) that using ANFO will result in greater cut lengths than ANE cuts.	

Scaling Time - ANFO vs Emulsion - 5.0 x 6.0 Drive - 4.3m Steels

ANFO	ANE
45	45
30	60
30	60
25	60
20	40
	40
	45
	25
	30
	28
	35
	45

Fixed Variables: 4.3m steels

5.5 x 6.0 Drive

F-Test Two-Sample for Variances

H0: The scaling time variance between ANFO and ANE cuts is the same

Ha: The scaling time variance between ANFO and ANE cuts is unequal

Alpha = 0.05

	ANE	*ANFO*
Mean	42.75	30
Variance	152.5681818	87.5
Observations	12	5
df	11	4
F	1.743636364	
P(F<=f) one-tail	0.312336083	
F Critical one-tail	5.935812699	
Reject H0?	NO	

Interpretation: There is not enough evidence to suggest (at 95% confidence) that using ANFO will result in less variance than ANE cuts.

F-Test Two-Sample for Variances

H0: The scaling time variance between ANFO and ANE cuts is the same

Ha: The scaling time variance between ANFO and ANE cuts is unequal

Alpha = 0.1

	ANE	*ANFO*
Mean	42.75	30
Variance	152.5681818	87.5
Observations	12	5
df	11	4

F	1.743636364
P(F<=f) one-tail	0.312336083
F Critical one-tail	3.906693742
Reject H0?	NO
Interpretation:	There is not enough evidence to suggest (at 90% confidence) that using ANFO will result in less variance than ANE cuts.

t-Test: Two-Sample Assuming Equal Variances		
H0: The average scaling time between ANFO and ANE cuts is the same		
Ha: The average scaling time is greater in ANFO cuts than ANE cuts		
Alpha =	0.05	
	ANE	*ANFO*
Mean	42.75	30
Variance	152.5681818	87.5
Observations	12	5
Pooled Variance	135.2166667	
Hypothesized Mean Difference	0	
df	15	
t Stat	2.059900469	
P(T<=t) one-tail	0.028599911	
t Critical one-tail	1.753050356	
P(T<=t) two-tail	0.057199822	
t Critical two-tail	2.131449546	
Reject H0?	YES	
Interpretation:	There is enough evidence to suggest (at 95% confidence) that using ANE will result in greater scaling time than ANFO cuts.	

Scaling Time - ANFO vs Emulsion - 5.0 x 5.0 Drive - 4.9m Steels

ANFO	ANE
30	45
40	50
10	25
	50
	60
	30
	50
	30
	20

Fixed Variables: 4.9m steels

5.0 x 5.0 Drive

F-Test Two-Sample for Variances

H0: The scaling time variance between ANFO and ANE cuts is the same

Ha: The scaling time variance between ANFO and ANE cuts is unequal

Alpha = 0.05

	ANFO	*ANE*
Mean	26.66666667	40
Variance	233.3333333	193.75
Observations	3	9
df	2	8
F	1.204301075	
P(F<=f) one-tail	0.348971783	
F Critical one-tail	4.458970108	
Reject H0?	NO	

Interpretation: There is not enough evidence to suggest (at 95% confidence) that using ANFO will result in less variance than ANE cuts.

F-Test Two-Sample for Variances

H0: The scaling time variance between ANFO and ANE cuts is the same

Ha: The scaling time variance between ANFO and ANE cuts is unequal

Alpha = 0.1

	ANE	*ANFO*
Mean	26.66666667	40
Variance	233.3333333	193.75
Observations	3	9
df	2	8

F	1.204301075
P(F<=f) one-tail	0.348971783
F Critical one-tail	3.11311764
Reject H0?	NO
Interpretation:	There is not enough evidence to suggest (at 90% confidence) that using ANFO will result in less variance than ANE cuts.

t-Test: Two-Sample Assuming Equal Variances		
H0: The average scaling time between ANFO and ANE cuts is the same		
Ha: The average scaling time is greater in ANFO cuts than ANE cuts		
Alpha =	0.1	
	ANE	*ANFO*
Mean	40	26.66666667
Variance	193.75	233.3333333
Observations	9	3
Pooled Variance	201.6666667	
Hypothesized Mean Difference	0	
df	10	
t Stat	1.40835758	
P(T<=t) one-tail	0.09467405	
t Critical one-tail	1.372183641	
P(T<=t) two-tail	0.189348101	
t Critical two-tail	1.812461123	
Reject H0?	YES	YES
Interpretation:	There is enough evidence to suggest (at 90% confidence) that using ANE will result in greater scaling time than ANFO cuts.	

Advance in Anisotropic Ground - 4.9m Steel - Emulsion

PARALLEL	NON-PARALLEL
3.6	4.4
4.2	4.2
3.4	4.1
3.5	2.9
3.8	4.5
4	4.6
3.2	4.4
3.9	3.1
4.3	4.4
4.1	4.9
3.9	4
3.2	3.3
3.5	2.7
3.2	4.2
2	3.4
3	4.3
3.6	4.4
3.8	3.36
	4.5
	4.25
	3.8
	4.5
	2.9
	4.2
	3.6

Fixed Variables: 4.9m steels

ANE

F-Test Two-Sample for Variances

H0: The cut length variance between Parallel and Non-Parallel cuts is the same

Ha: The cut length variance between Parallel and Non-Parallel cuts is unequal

Alpha = 0.05

	NON-PARALLEL	*PARALLEL*
Mean	3.9564	3.566666667
Variance	0.382274	0.291764706
Observations	25	18
df	24	17
F	1.310213306	
P(F<=f) one-tail	0.286294221	
F Critical one-tail	2.189766456	
Reject H0?	NO	NO

Interpretation: There is not enough evidence to suggest (at 95% confidence) that the variances are unequal.

F-Test Two-Sample for Variances

H0: The cut length variance between Parallel and Non-Parallel cuts is the same

Ha: The cut length variance between Parallel and Non-Parallel cuts is unequal

Alpha = 0.1

	NON-PARALLEL	*PARALLEL*
Mean	3.9564	3.566666667
Variance	0.382274	0.291764706
Observations	25	18

df	24	17
F	1.310213306	
P(F<=f) one-tail	0.286294221	
F Critical one-tail	1.836241786	
Reject H0?	NO	NO
Interpretation:	There is not enough evidence to suggest (at 90% confidence) that the variances are unequal.	

t-Test: Two-Sample Assuming Equal Variances

H0: The average advance metres between cuts in Parallel and Non-Parallel ground is the same

Ha: The average advance metres is greater in Non-Parallel ground than in Parallel ground

Alpha = 0.05

	NON-PARALLEL	*PARALLEL*
Mean	3.9564	3.566666667
Variance	0.382274	0.291764706
Observations	25	18
Pooled Variance	0.344745756	
Hypothesized Mean Difference	0	
df	41	
t Stat	2.147287108	
P(T<=t) one-tail	0.018866431	
t Critical one-tail	1.682878002	
P(T<=t) two-tail	0.037732862	
t Critical two-tail	2.01954097	
Reject H0?	YES	YES

Interpretation: There is enough evidence to suggest (at 95% confidence) that average advance metres will be greater in non-Parallel ground than in Parallel Ground

Advance in Anisotropic Ground - 4.9m Steel - ANFO

PARALLEL	NON-PARALLEL
4	4.5
3.5	4.4
4.5	4.6
	3.9
	4.1
	3.9
	4.1
	4.7
	4.4

Fixed Variables: 4.9m steels

ANFO

F-Test Two-Sample for Variances

H0: The cut length variance between Parallel and Non-Parallel cuts is the same

Ha: The cut length variance between Parallel and Non-Parallel cuts is unequal

Alpha = 0.05

	PARALLEL	*NON-PARALLEL*
Mean	4	4.288888889
Variance	0.25	0.088611111
Observations	3	9
df	2	8
F	2.821316614	
P(F<=f) one-tail	0.118240738	
F Critical one-tail	4.458970108	
Reject H0?	NO	NO

Interpretation: There is not enough evidence to suggest (at 95% confidence) that the variances are unequal.

F-Test Two-Sample for Variances

H0: The cut length variance between Parallel and Non-Parallel cuts is the same

Ha: The cut length variance between Parallel and Non-Parallel cuts is unequal

Alpha = 0.1

	PARALLEL	*NON-PARALLEL*
Mean	4	4.288888889
Variance	0.25	0.088611111
Observations	3	9

df	2	8
F	2.821316614	
P(F<=f) one-tail	0.118240738	
F Critical one-tail	3.11311764	
Reject H0?	NO	NO
Interpretation:	There is not enough evidence to suggest (at 90% confidence) that the variances are unequal.	

t-Test: Two-Sample Assuming Equal Variances

H0: The average advance metres between cuts in Parallel and Non-Parallel ground is the same

Ha: The average advance metres is greater in Non-Parallel ground than in Parallel ground

Alpha = 0.1

	NON-PARALLEL	*PARALLEL*
Mean	4.288888889	4
Variance	0.088611111	0.25
Observations	9	3
Pooled Variance	0.120888889	
Hypothesized Mean Difference	0	
df	10	
t Stat	1.246318107	
P(T<=t) one-tail	0.120526417	
t Critical one-tail	1.812461123	
P(T<=t) two-tail	0.241052834	
t Critical two-tail	2.228138852	
Reject H0?	NO	NO

Interpretation: There is not enough evidence to suggest (at 90% confidence) that average advance metres will be greater in non-Parallel ground than in Parallel Ground

t-Test: Two-Sample Assuming Equal Variances

H0: The average advance metres between cuts in Parallel and Non-Parallel ground is the same

Ha: The average advance metres is greater in Non-Parallel ground than in Parallel ground

Alpha = 0.15

	NON-PARALLEL	*PARALLEL*
Mean	4.288888889	4

Variance	0.088611111	0.25
Observations	9	3
Pooled Variance	0.120888889	
Hypothesized Mean Difference	0	
df	10	
t Stat	1.246318107	
P(T<=t) one-tail	0.120526417	
t Critical one-tail	1.093058074	
P(T<=t) two-tail	0.241052834	
t Critical two-tail	1.559235933	
Reject H0?	YES	YES
Interpretation:	There is enough evidence to suggest (at 85% confidence) that average advance metres will be greater in non-Parallel ground than in Parallel Ground	

Scaling time in Anisotropic Ground - 4.9m Steels - 5.0 x 5.0 Drive

PARALLEL	NON-PARALLEL
50	45
50	25
60	30
50	30
30	20
	40
	10

Fixed Variables: 4.9m steels

5.0 x 5.0 Drive

F-Test Two-Sample for Variances

H0: The scaling time variance between Parallel and Non-Parallel cuts is the same

Ha: The scaling time variance between Parallel and Non-Parallel cuts is unequal

Alpha = 0.05

	NON-PARALLEL	*PARALLEL*
Mean	28.57142857	48
Variance	139.2857143	120
Observations	7	5
df	6	4
F	1.160714286	
P(F<=f) one-tail	0.463263283	
F Critical one-tail	6.163132283	
Reject H0?	NO	NO

Interpretation: There is not enough evidence to suggest (at 95% confidence) that the variances are unequal.

F-Test Two-Sample for Variances

H0: The scaling time variance between Parallel and Non-Parallel cuts is the same

Ha: The scaling time variance between Parallel and Non-Parallel cuts is unequal

Alpha = 0.1

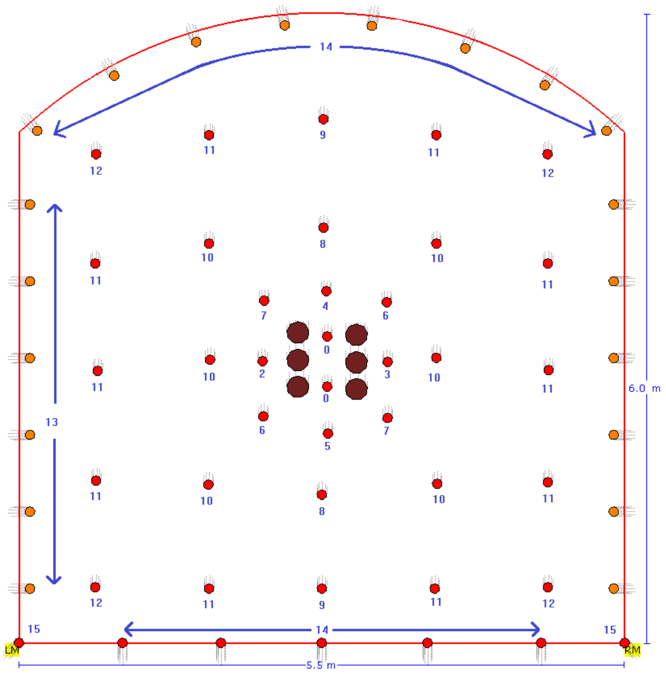
	NON-PARALLEL	*PARALLEL*
Mean	28.57142857	48
Variance	139.2857143	120
Observations	7	5
df	6	4

F	1.160714286	
P(F<=f) one-tail	0.463263283	
F Critical one-tail	4.009749313	
Reject H0?	NO	NO
Interpretation:	There is not enough evidence to suggest (at 90% confidence) that the variances are unequal.	

t-Test: Two-Sample Assuming Equal Variances		
H0: The average scaling time between cuts in Parallel and Non-Parallel ground is the same		
Ha: The average scaling time is greater in Parallel ground than in Non-Parallel ground		
Alpha =	0.05	
	PARALLEL	*NON-PARALLEL*
Mean	48	28.57142857
Variance	120	139.2857143
Observations	5	7
Pooled Variance	131.5714286	
Hypothesized Mean Difference	0	
df	10	
t Stat	2.892700512	
P(T<=t) one-tail	0.008016413	
t Critical one-tail	1.812461123	
P(T<=t) two-tail	0.016032825	
t Critical two-tail	2.228138852	
Reject H0?	YES	YES
Interpretation:	There is enough evidence to suggest (at 95% confidence) that scaling time will be longer in parallel cuts than in non-parallel cuts.	

APPENDIX H

CHARGE PLAN - KIM 520 EXP - Central Burn



DRILLING

DRILLL SETUP	SPECIFICATION
Steel - Length	4.9m
Steel - Type	Round
Bit Type	Round
Bit Diameter - Burn	51 mm
Bit Diameter - Face	51 mm
Bit Diameter - Perimeter	48 mm
Bit Diameter - Lifters	48 mm

DRILL PATTERN	QTY	Notes
Wall Holes	6	per wall
Lifters	7	
Back Holes	8	
Inner Grid Holes	24	5 across, 5 up.
Burn Holes	10	Incl. 2 shot holes
Reamers	6	Tall Twin Six

Note: If you need to, stretch the spacing to fit the holes evenly across the face.

TIMING & BLAST DIRECTION

- Both shot holes fired with 0
- The next two holes to fire are to the left and right of the reamers (fired with 2 & 3)
- Once the burn is fired, create a vertical opening (with the 8s & 9s)
- Strip the hole columns inward (to achieve lateral movement across the grain)
- Avoid stripping downward except for the backs
- Tie off & bunch lead tails to achieve clean & neat work area and prevent misfires

CHARGING

KETTLE PRESSURE	Approx. 500 kpa (72 psi)

DRILL PATTERN	EXPLOSIVE	QTY
BURN	ANFO	81 kg
FACE	ANFO	196 kg
PERIMETERS	LD 50/50	91 kg
LIFTERS	Longs	Approx. 49

TOTALS	Units	Kg
ANFO		277 kg
LD 50/50		91 kg
Shorts	54	
Longs	Approx 49	29 kg

CUT SUMMARY

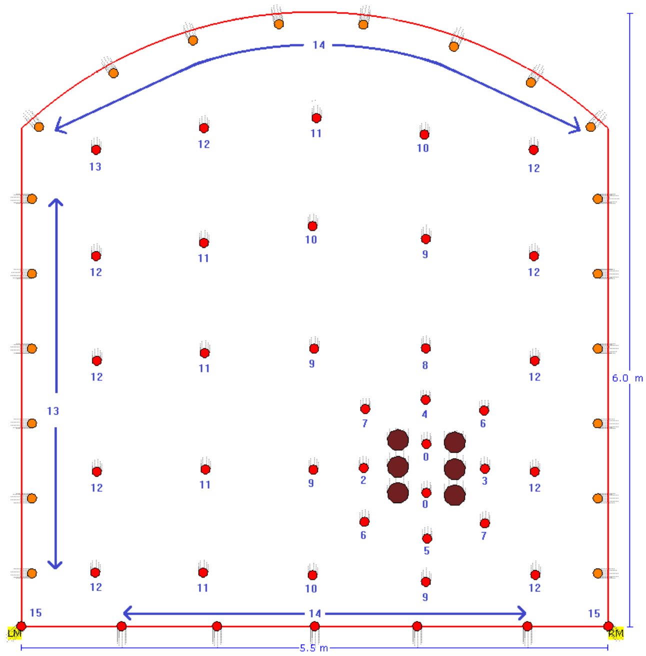
Approx Cut Volume	145.2 m3
Total Drill Metres	268 m
Total Drilled Holes	67

Total Charged Holes	61
Total Explosive	397 kg
Powder Factor	2.73 kg / m3

	Detonators															
LP #	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
QTY	2	0	1	1	1	1	2	2	2	2	6	10	4	12	13	2

APPENDIX I

CHARGE PLAN - KIM 520 EXP - Right Burn



DRILLING

DRILLL SETUP	SPECIFICATION
Steel - Length	4.9m
Steel - Type	Round
Bit Type	Round
Bit Diameter - Burn	51 mm
Bit Diameter - Face	51 mm
Bit Diameter - Perimeter	48 mm
Bit Diameter - Lifters	48 mm

DRILL PATTERN	QTY	Notes
Wall Holes	6	per wall
Lifters	7	
Back Holes	8	
Inner Grid Holes	24	5 across, 5 up.
Burn Holes	10	Incl. 2 shot holes
Reamers	6	Tall Twin Six

Note: If you need to, stretch the spacing to fit the holes evenly across the face.

TIMING & BLAST DIRECTION

- Both shot holes fired with 0
- The next two holes to fire are to the left and right of the reamers (fired with 2 & 3)
- Once the burn is fired, create a vertical opening (with the 8 & 9s)
- Strip the hole columns inward (to achieve lateral movement across the grain)
- Avoid stripping downward except for the backs
- Tie off & bunch lead tails to achieve clean & neat work area and prevent misfires

CHARGING

KETTLE PRESSURE	Approx. 500 kpa (72 psi)

DRILL PATTERN	EXPLOSIVE	QTY
BURN	ANFO	81 kg
FACE	ANFO	196 kg
PERIMETERS	LD 50/50	91 kg
LIFTERS	Longs	Approx. 49

TOTALS	Units	Kg
ANFO		277 kg
LD 50/50		91 kg
Shorts	54	
Longs	Approx 49	29 kg

CUT SUMMARY

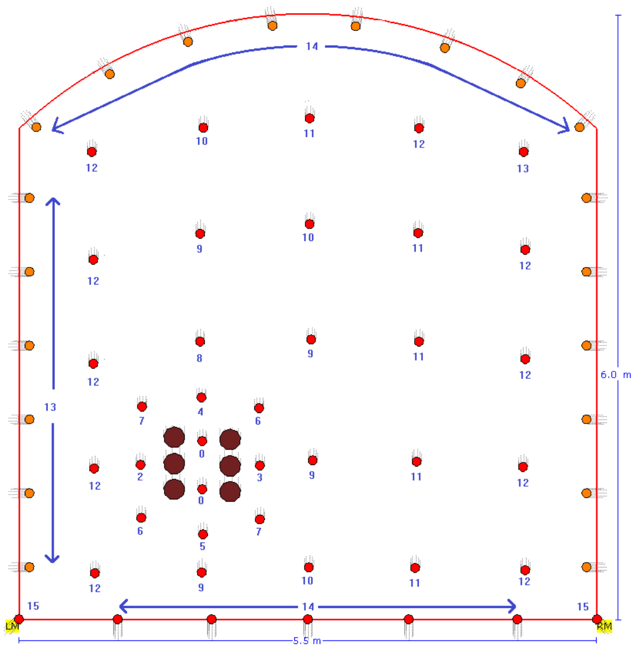
Approx Cut Volume	145.2 m3
Total Drill Metres	268 m
Total Drilled Holes	67

Total Charged Holes	61
Total Explosive	397 kg
Powder Factor	2.73 kg / m3

	Detonators															
LP #	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
QTY	2	0	1	1	1	1	2	2	1	4	3	5	10	13	13	2

APPENDIX J

CHARGE PLAN - KIM 520 EXP - Left Burn



DRILLING

DRILLL SETUP	SPECIFICATION
Steel - Length	4.9m
Steel - Type	Round
Bit Type	Round
Bit Diameter - Burn	51 mm
Bit Diameter - Face	51 mm
Bit Diameter - Perimeter	48 mm
Bit Diameter - Lifters	48 mm

DRILL PATTERN	QTY	Notes
Wall Holes	6	per wall
Lifters	7	
Back Holes	8	
Inner Grid Holes	24	5 across, 5 up.
Burn Holes	10	Incl. 2 shot holes
Reamers	6	Tall Twin Six

Note: If you need to, stretch the spacing to fit the holes evenly across the face.

TIMING & BLAST DIRECTION

- Both shot holes fired with 0
- The next two holes to fire are to the left and right of the reamers (fired with 2 & 3)
- Once the burn is fired, create a vertical opening (with the 8 & 9s)
- Strip the hole columns inward (to achieve lateral movement across the grain)
- Avoid stripping downward except for the backs
- Tie off & bunch lead tails to achieve clean & neat work area and prevent misfires

CHARGING

KETTLE PRESSURE	Approx. 500 kpa (72 psi)

DRILL PATTERN	EXPLOSIVE	QTY
BURN	ANFO	81 kg
FACE	ANFO	196 kg
PERIMETERS	LD 50/50	91 kg
LIFTERS	Longs	Approx. 49

TOTALS	Units	Kg
ANFO		277 kg
LD 50/50		91 kg
Shorts	54	
Longs	Approx 49	29 kg

CUT SUMMARY

Approx Cut Volume	145.2 m3
Total Drill Metres	268 m
Total Drilled Holes	67

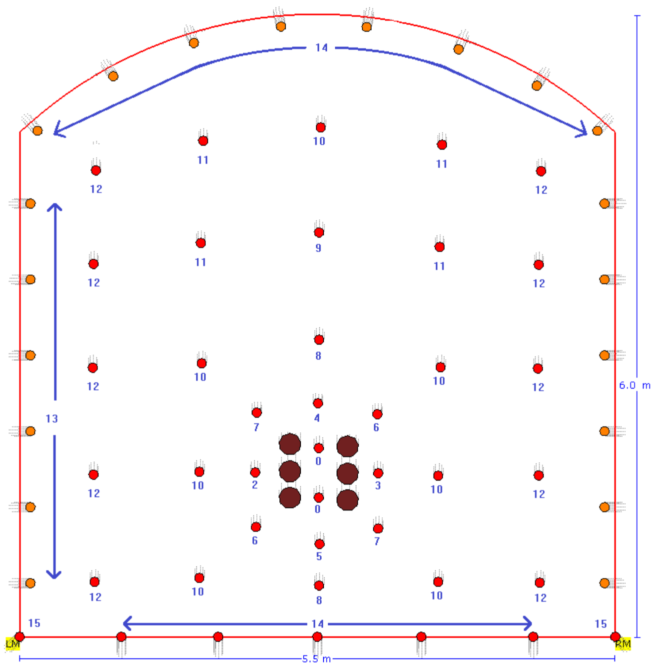
Total Charged Holes	61
Total Explosive	397 kg
Powder Factor	2.73 kg / m3

Detonators

LP #	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
QTY	2	0	1	1	1	1	2	2	1	4	3	5	10	13	13	2

APPENDIX K

CHARGE PLAN - KIM 520 EXP - Low Central Burn



DRILLING

DRILLL SETUP	SPECIFICATION
Steel - Length	4.9m
Steel - Type	Round
Bit Type	Round
Bit Diameter - Burn	51 mm
Bit Diameter - Face	51 mm
Bit Diameter - Perimeter	48 mm
Bit Diameter - Lifters	48 mm

DRILL PATTERN	QTY	Notes
Wall Holes	6	per wall
Lifters	7	
Back Holes	8	
Inner Grid Holes	24	5 across, 5 up.
Burn Holes	10	Incl. 2 shot holes
Reamers	6	Tall Twin Six

Note: If you need to, stretch the spacing to fit the holes evenly across the face.

TIMING & BLAST DIRECTION

- Both shot holes fired with 0
- The next two holes to fire are to the left and right of the reamers (fired with 2 & 3)
- Once the burn is fired, create a vertical opening (with the 8s & 9s)
- Strip the hole columns inward (to achieve lateral movement across the grain)
- Avoid stripping downward except for the backs
- Tie off & bunch lead tails to achieve clean & neat work area and prevent misfires

CHARGING

KETTLE PRESSURE	Approx. 500 kpa (72 psi)

DRILL PATTERN	EXPLOSIVE	QTY
BURN	ANFO	81 kg
FACE	ANFO	196 kg
PERIMETERS	LD 50/50	91 kg
LIFTERS	Longs	Approx. 49

TOTALS	Units	Kg
ANFO		277 kg
LD 50/50		91 kg
Shorts	54	
Longs	Approx 49	29 kg

CUT SUMMARY

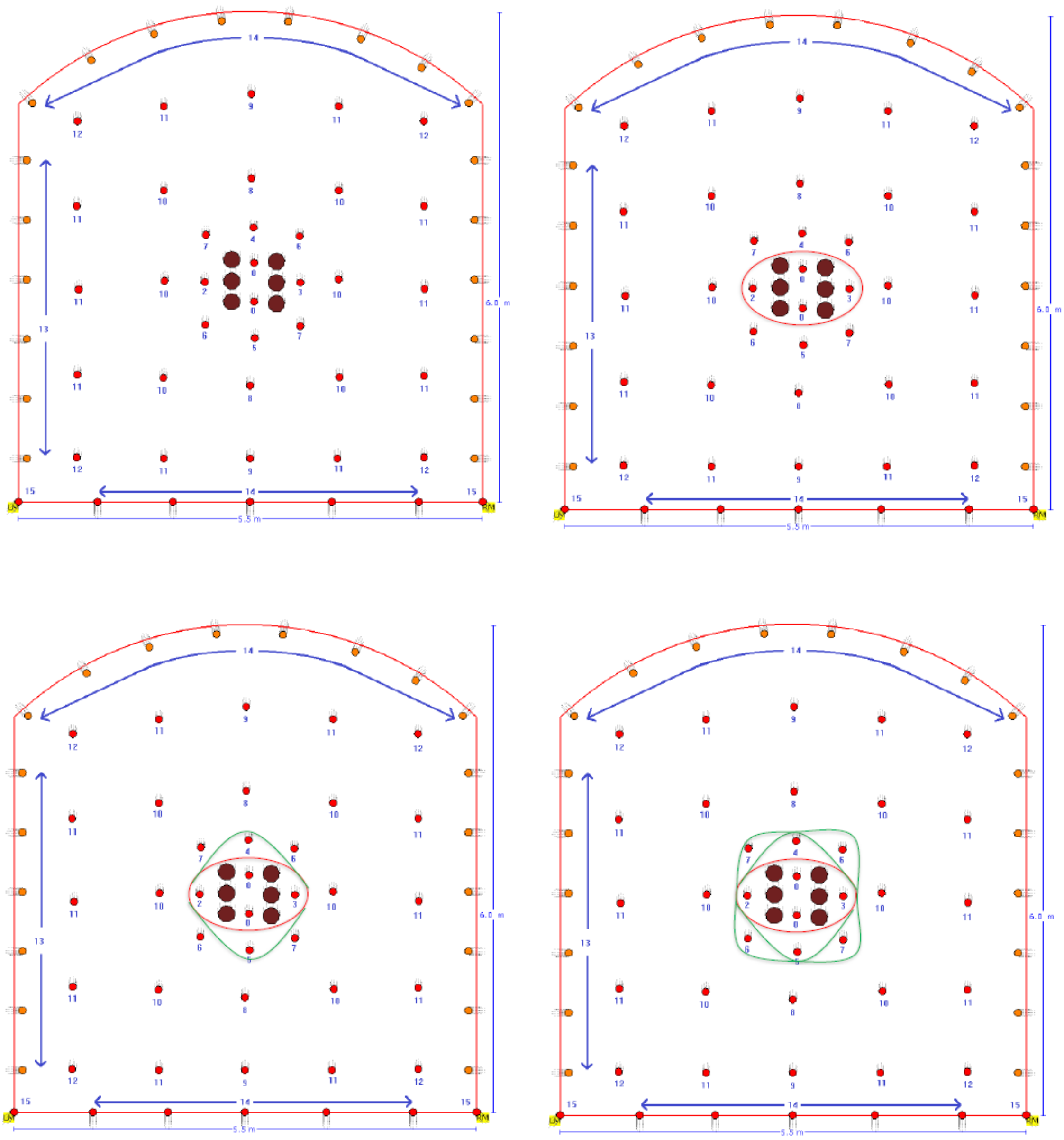
Approx Cut Volume	145.2 m3
Total Drill Metres	268 m
Total Drilled Holes	67

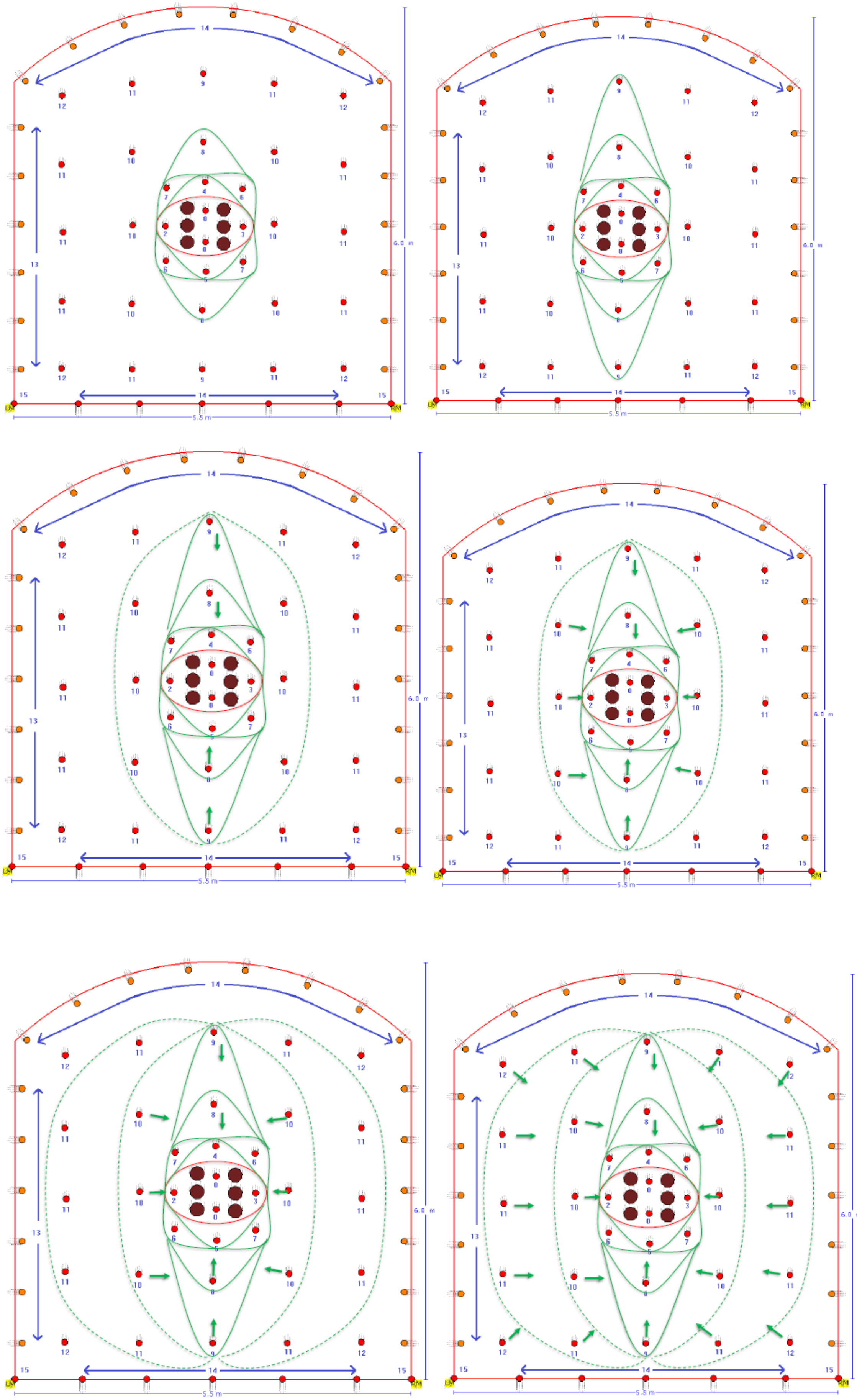
Total Charged Holes	61
Total Explosive	397 kg
Powder Factor	2.73 kg / m3

	Detonators															
LP #	0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
QTY	2	0	1	1	1	1	2	2	2	2	6	10	4	12	13	2

APPENDIX L

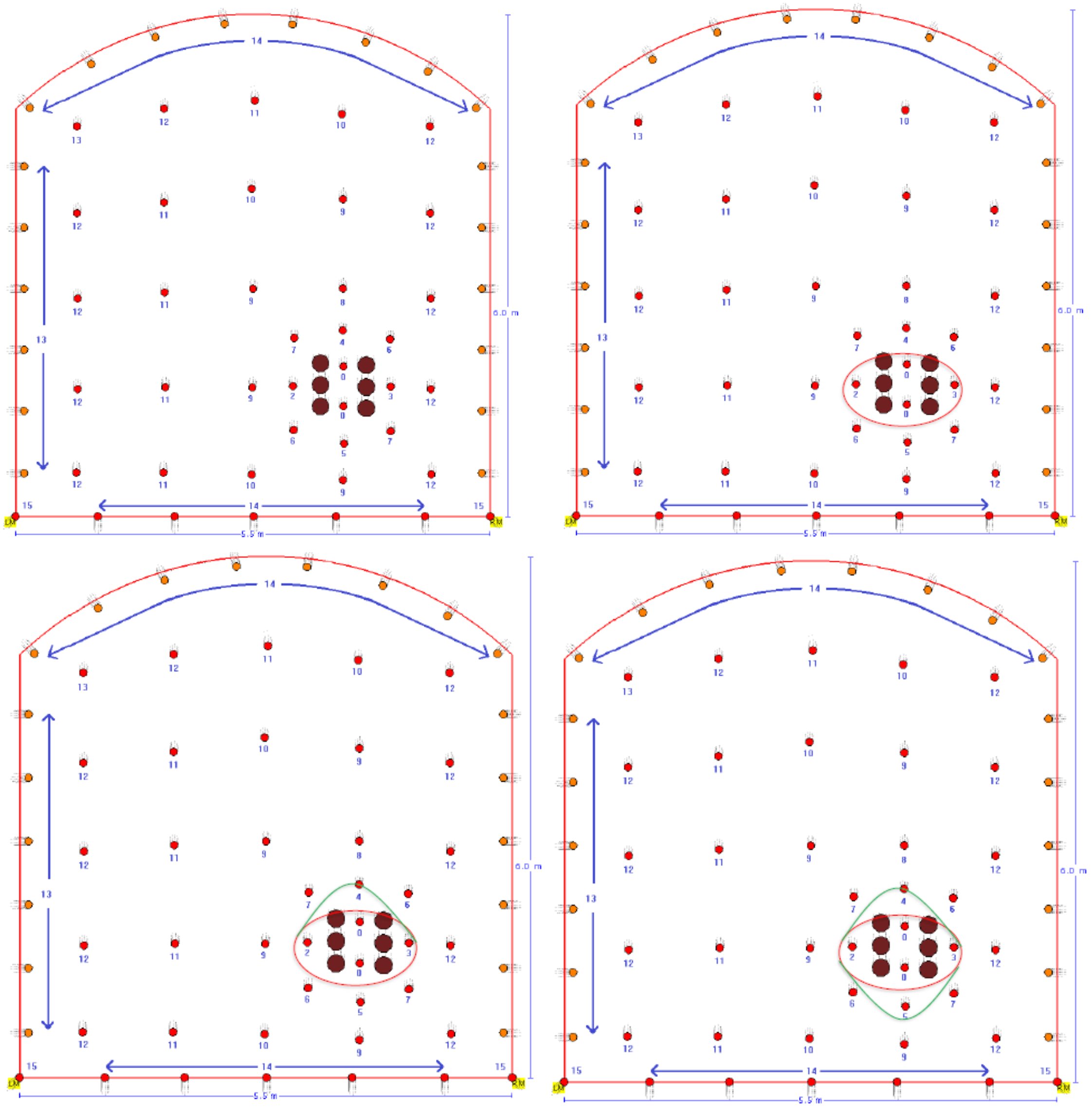
Recommended blast sequence for 520 EXP (vertically laminated ground)

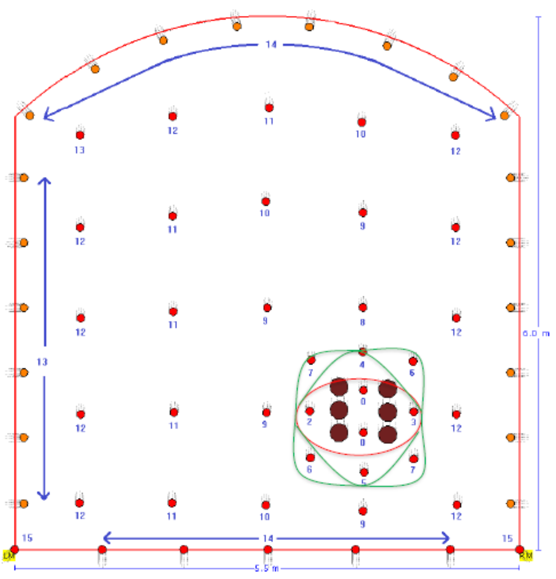


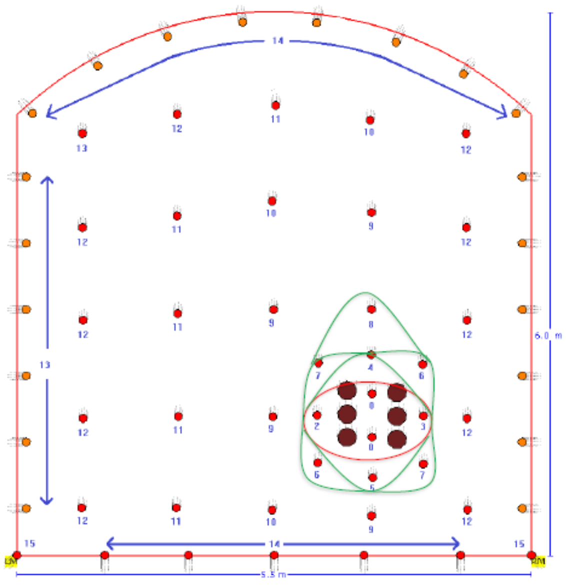

14
13
15
6.0 m
5.5 m

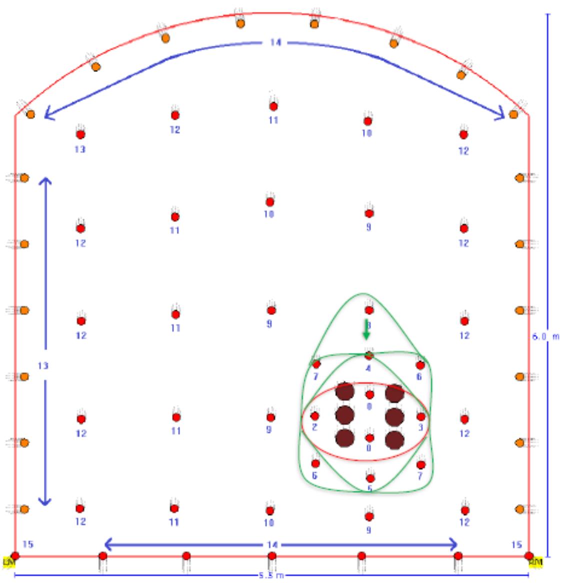
APPENDIX M

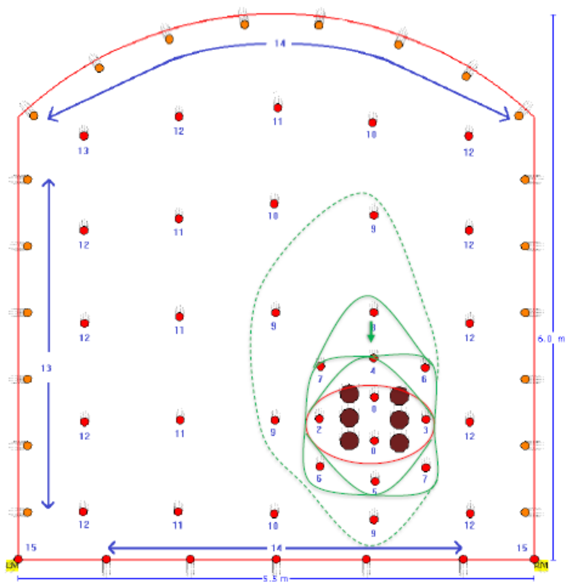
Recommended blasting sequence for 520 EXP with different burn location.

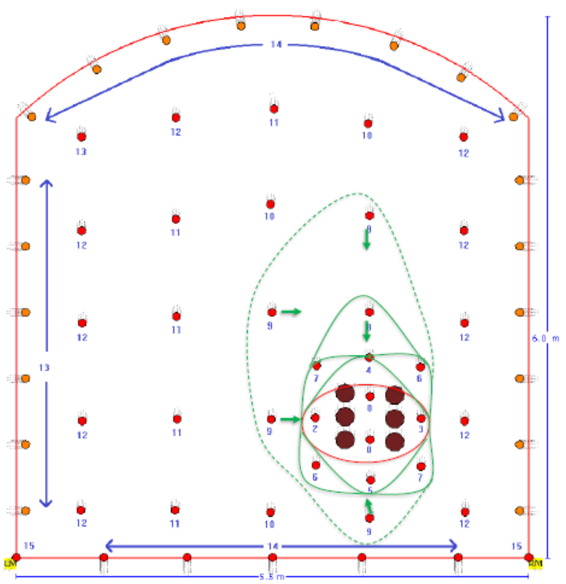


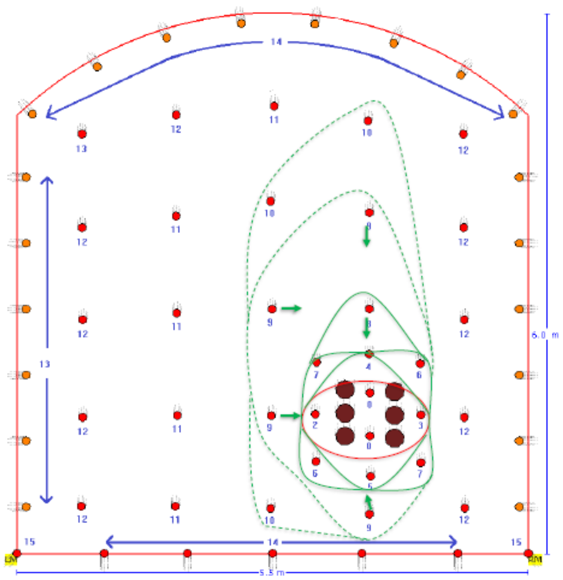

14
13
15
6.0 m
5.5 m


14
13
15
6.0 m
5.5 m


14
13
15
6.0 m
5.5 m


14
13
15
6.0 m
5.5 m


14
13
15
6.0 m
5.5 m


14
13
15
6.0 m
5.5 m