Subdrill – The Underutilised Blasting Parameter

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Abstract

In metalliferous mining operations, subdrill is that portion of the blast hole that is drilled below the target grade elevation, and in most cases loaded with explosives. Its primary aim is to enable efficient excavation down to the target floor level. Subdrill is often described as being expensive and that excessive subdrill should be avoided, as it can result in damage to the subsequent floor, lead to difficult drilling conditions and higher vibration levels. As a result, many operations aim to apply the bare minimum amount of subdrill in order to excavate to the target floor.

At a large open cut copper operation, challenging geological structure resulted in slabby and blocky fragmentation in the stemming region, resulting in poor crusher throughput due to bridging. To overcome the poor fragmentation in the top of the bench, longer than conventional subdrill was used to precondition the top third of the bench. The technique significantly improved the run of mine fragmentation and resulted in feed to the in-pit crushers that is more consistent, which produced less crusher downtime due to bridging.

A new method of quantifying the energy in a blast has been created. Termed the linear powder factor, or energy factor, the calculation considers the energy contributed by the subdrill of the upper bench and its impact on the energy in the lower bench. The technique enables blast designs to be optimised to deliver finer fragmentation and more consistent blast outcomes.

To support the layer of preconditioned rock at the top of the bench, an innovative new device was developed. The device is inserted into the drill collar immediately after drilling and remains in position throughout the subsequent steps of hole depth measurement, priming, loading and removed prior to stemming. The device not only supports the broken rock at the top of the bench, but also minimises the amount of drill cuttings that fall back into the hole. Furthermore, the device helps protect the hole from damage by external factors such as weather events, drill tracks, vehicle tyres and movement around the collar during measurement and loading of the blast hole.

Introduction

In open pit blasting, the portion of the blast hole that is drilled below the target floor elevation, and in most cases loaded with explosives, is known as sublevel drilling, subgrade or subdrill. According to common literature, the primary aim of subdrill is to enable efficient excavation down to the target floor level and to pull the full face (Dick, Fletcher, & D'Andrea, 1983; Jimeno, Jimeno, & Carcedo, 1995; International Society of Explosives Engineers, 2011).

In addition to the common recommendation to use the minimal amount of subdrill, many sources caution against the use of long or excessive subdrill due to the negative effects for the mining operation (Dick et al., 1987; Jimeno et al., 1995; Hustrulid, 1999; ISEE, 2011), including:

- Wasting drilling and blasting time resources and expenditure.
- Increasing ground vibration due to over confinement of explosives in the subdrill.
- Accentuated rock movement and displacement.
- Damage to the rock mass at and below the target floor elevation.

- Difficult drilling conditions in the underlying bench, including collaring, leading to problematic drilling conditions.

As a result, many operations aim to apply the bare minimum amount of subdrill required to excavate to the target floor.

Longer than recommended subdrill has been used previously to deliver finer run-of-mine (ROM) fragmentation and improved downstream process of crushing and milling. In the Quarry Academy workshop, Mirabelli & Lislerud (2005) showed that large subdrill from the prior bench delivered a higher percentage of sub 1¼" using small diameter holes. In a large open cut mine in Mexico, Hawke & Dominguez (2015) used subdrill equal to the stemming length of the underlying bench to significantly improve run of mine (ROM) fragmentation and downstream processes.

Subdrill – The Current Paradigm

Design guidelines, or rules of thumb, for subdrill (Su) are general in nature and consist of design factors based around the blast hole diameter (D) or burden distance (B). Many authors recommend the use of no subdrill in the event that a favourable bedding plane, joint set or parting plane aligns with the target floor level (Ash R. L., 1963b; Dick, Fletcher, & D'Andrea, 1983; Jimeno, Jimeno, & Carcedo, 1995; Hustrulid, 1999; International Society of Explosives Engineers, 2011).

Langefors & Kihlström (1963) discussed the toe breaking capacity of a spherical and cylindrical charge and showed that a cylindrical charge with a length equal to 0.3B has the same breaking capacity as the equivalent spherical charge, and that the breakage decreases for charges exceeding 0.3B (Figure 1).

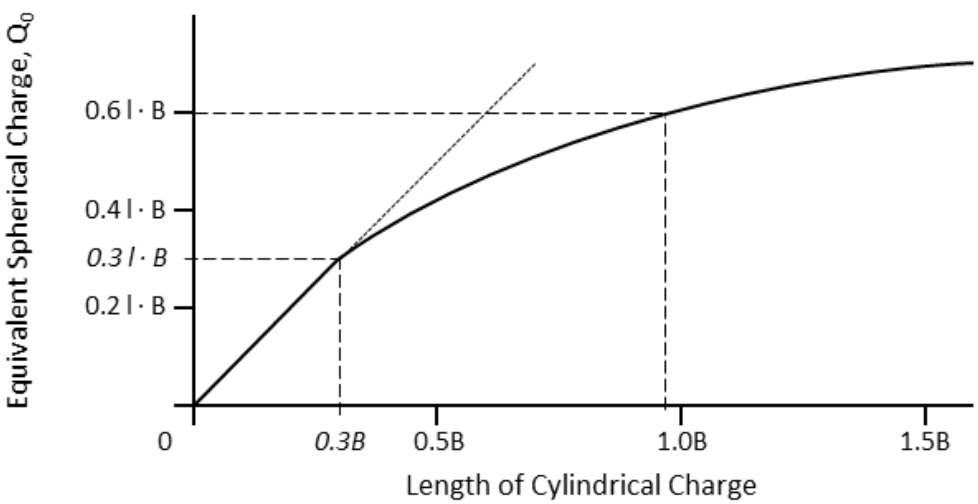

Figure 1: Toe breaking capacity of spherical and cylindrical charges (Langefors & Kihlström, 1963).

In 1963, Ash (1963a) described five blast design standards with which to evaluate a blast. Ash gathered data from a wide range of operations and calculated the ratios in use for each of his design standards. These standards form the basis for the rules of thumb for basic blast design still used today. Using data from 125 open cut operations, excluding coal stripping, Ash recommended that the subdrill ratio should never be less than 0.2B, and was typically around 0.3B (Ash, 1963b).

Jimeno, Jimeno, & Carcedo (1995) relate subdrill effectiveness to cratering theory and compare breakage at the toe of blast holes to inverted cones, with angles to the horizontal of between 10 and 30 degrees, depending on the rock mass properties, and that subdrill is usually equal to 0.3B. The International Society of Explosives Engineers (2011) recommends the use of a site specific cratering study to determine the most effective subdrill, or to multiply the burden by 0.2 to 0.5 as a starting point. The main point of difference between a crater blast and a length of subdrill is that in a blast, there is no bench level present for the crater to break to.

Hustrulid (1999) discusses two main methods for calculating subdrill length. The first relates to the run up distance of the explosive, assumed to be in the order of 6D and the normal offset of the primer in the blast hole of 2D, which yields a recommended subdrill length of 8D. The second method relates to the burden distance and is derived from the first method using the presumption that the burden is proportional to the hole diameter, which yields a subdrill value of ≈0.3B. Hustrulid (1999) investigated the validity of the 0.3B subdrill value on breakage of the bench toe using a series of computer simulations and concluded that the rule of thumb is valid.

In quarry blasting subdrill is often kept to a minimum, and in the past horizontal holes have even been used to ensure breakage at the desired floor level (McAdam & Westwater, 1958; Sandvik Tamrock, 1999). In open cast coal mines, subdrill is known to damage coal seams, therefore blast holes are either drilled to terminate above the coal seam or are backfilled, resulting in negative subdrill (Jimeno, Jimeno, & Carcedo, 1995). In metalliferous mining operations, most production blasting uses positive subdrill, and zero or negative subdrill is used in some wall control holes to protect the integrity of the rock at the pit wall position.

Challenging Geology vs. Standard Subdrill Parameters

The Sentinel operation is an open cut copper mine, located approximately 150 kilometres west of the town of Solwezi in the North Western Province of Zambia. The Sentinel deposit is a structurally modified, sediment hosted copper deposit. The mineralisation is primarily sulphide copper, with sheet like horizons of ore that dip from south to north at between 20 and 30 degrees (Gray, Lawlor, & Stone, 2015). In the current mining stage, the northern and southern ore zones. The southern ore zone is characterised by more closely spaced joint sets with long persistence and the northern ore zone by more widely spaced joint sets. One joint set in particular has a spacing of many metres and lies almost parallel to the bench level.

The operation is a traditional drill, blast, load and haul operation, but instead of hauling ore to a run of mine (ROM) stockpile, the ore is tipped directly into three in pit crushers, before being crushed and conveyed overland to the process plant. The in pit crushing and conveying strategy is designed to reduce haulage costs and remove the cost of rehandling ore between a traditional ROM stockpile and stationary crusher. Due to the absence of the ROM stockpile, it is necessary to fragment the ore by drilling and blasting to maximise crusher throughput and minimise crusher blockages.

To enable the annual material movement target of 150 million tonne (165 million ton), large rotary drills drill large diameter holes. Historically the production blast designs used subdrill and stemming lengths with standard design factors relative to the hole diameter. As a result, the stemming length was equal to almost 50% of the bench height.

The uneven vertical distribution of the explosives combined with the structural orientation and spacing produced different ROM fragmentation in each of the ore zones. In the southern ore zone, the relatively close joint spacing and long persistence produced large slabs of material that often bridged the in pit

crushers. In the northern ore zone, the much wider joint spacing resulted in oversized material, which required secondary breakage.

Figure 2: Historical slabby and blocky fragmentation in the stemming region.

Design techniques used to combat the poor fragmentation in the stemming region included the use of small diameter stab holes, drilled in two rows between the burdens of the production pattern, and small explosive decks within the stemming column, known as stem decks, as shown in Figure 3. The stab hole technique proved effective in fragmenting the rock in the upper part of the bench, however it required almost three times the number of holes and twice the number of metres and was more than 20% more expensive per unit of rock. From an operational perspective, it took significantly longer to drill, load and to tie in, and the metres drilled by the small diameter drills did not deliver any additional broken material.

The stem deck technique proved effective in reducing the fragmentation in the stemming zone and required no extra holes or drill metres. It did however require more precise stemming and loading practices due to the small decks required, and increased the time to load a blast due to the additional loading and stemming steps required. Since the operation already used electronic initiating systems, allocating the correct delay between the main charge and stemming deck to maximise energy containment was not a challenge. The main consequence of the stem deck technique was the increased equipment exclusion zone that was required due to the increased risk of flyrock from the close proximity of the stem deck charge to the bench surface.

Even though both techniques improved the fragmentation in the stemming region, which reduced crusher bridging delays and secondary breakage, the blast results were not consistent.

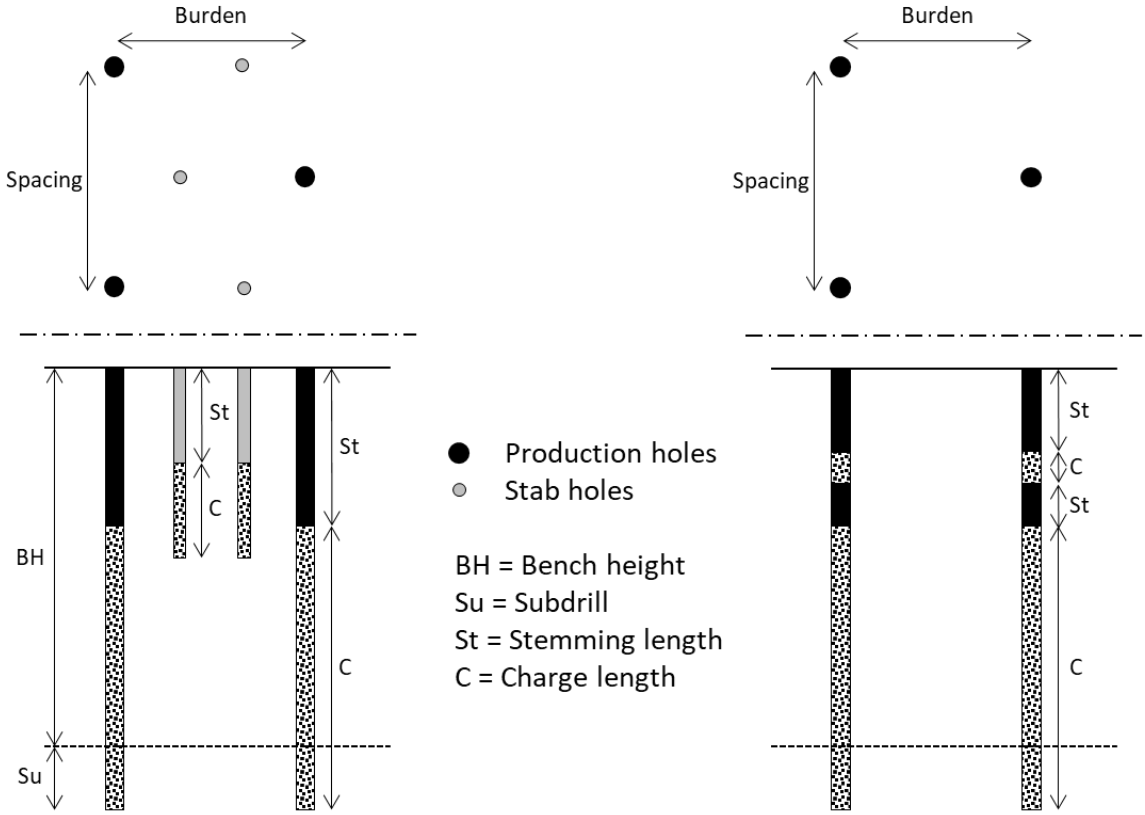

Figure 3: Stab hole technique on the left and stem deck technique on the right.

Preconditioning Technique and Operational Challenges

In an attempt to solve the stemming region fragmentation challenges, as well as simplify the drill and blast process, the subdrill was increased to equal one third of the bench height, or 0.7B, to precondition the top of the lower bench (Figure 4). All holes are initiated just above floor level to safeguard against any misfired primers being below floor level and to maximize the

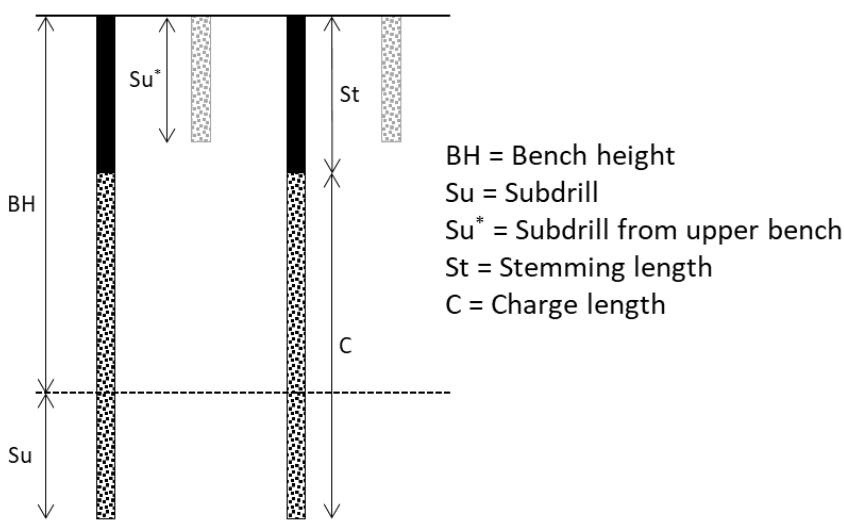

Figure 4: Preconditioning blast design using subdrill equal to one third of the bench height.

Operational challenges meant that the blast outcomes from the preconditioning technique were not consistent at first. The thick layer of fragmented material at the bench level enabled the large shovels to easily mine below the target floor elevation, therefore additional controls had to be implemented to ensure the bench level was mined accurately and the preconditioned material was maintained.

Drilling though the preconditioned material proved problematic, even though the 330 t (364 ton) haul trucks had compacted and stabilised the material while operating on the bench. Due to the percussion drilling action, the smaller drill fleet had some difficulty drilling through the preconditioned material than the large, rotary drills, and only experienced problems if the bench was water logged. Initially, it was difficult to retain the drill holes to full depth as the preconditioned material sometimes back filled or blocked the holes, resulting in the need to redrill some holes.

In order to stabilise the preconditioned material around the drill holes, a device was invented that supports the material at the collar of the hole. Called a Collar Keeper™, the device preserves the integrity of the top portion of the blast hole and prevents the drill cuttings from collapse. As shown in Figure 5, the device consists of a reusable, high-density polyethylene sheet between 1 and 3 mm (3/64 to 1/8 in) thick, depending on the size. The device is rolled into a cylinder before being inserted into the drill collar immediately after the hole has been drilled. The cylindrical form of the device offers additional vertical hole protection over traditional inverted cones and field trials have shown that the use of the device significantly reduces the loss of hole depths over time and protects the hole from damage from external forces such as drill tracks, vehicles tyres and equipment electrical cable movements. Keeping the device in the hole during loading minimises material entering the hole, which preserves the integrity of the detonator downline, reduces contamination of the explosives and enables more accurate loading of the explosive deck horizon, or stemming length.

Figure 5: Collar protection device, from left to right, before, during and after insertion.

Since overcoming the operational challenges, the preconditioning technique has delivered consistent ROM fragmentation, less oversize and crusher downtime due to bridging has dropped significantly. Split fragmentation systems were not installed until after the preconditioning technique was well established, however an example of the ROM fragmentation is shown in Figure 6 and in Figure 7, the layer of preconditioning visible behind a fired blast can be seen.

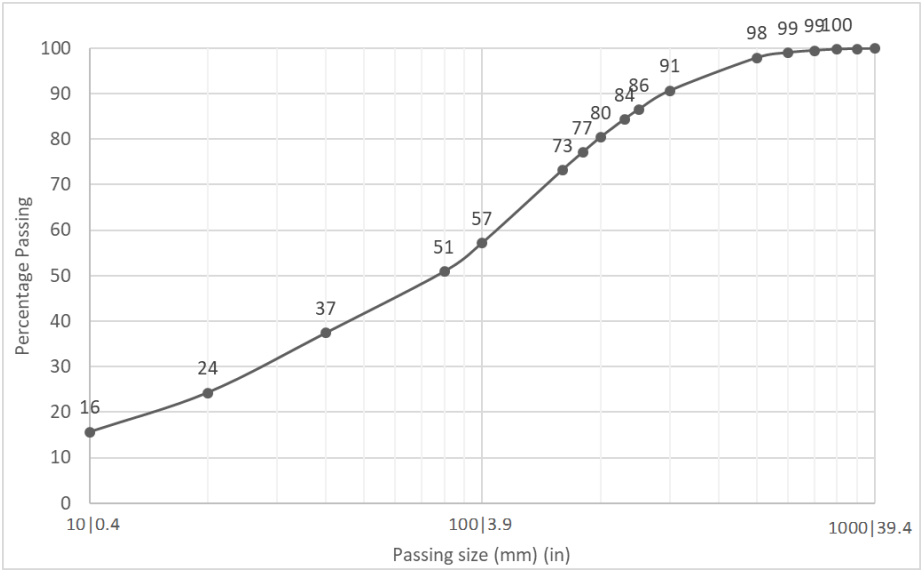

Figure 6: ROM ore fragmentation produced using the preconditioning technique.

Figure 7: Layer of preconditioned material, approximately 4 m (13.1 ft) thick, delivered by long subdrill, hole depth retention and accurate mining of the bench.

Quantifying Explosive Energy of the Preconditioning Technique

During the design and implementation of the preconditioning technique, existing methods of quantifying explosive energy in a bench were found to inadequate for quantifying the impact of subdrill in a bench, especially when design parameters changed from bench to bench.

Bench blasting is the method of drilling holes down from an upper surface, into the rock, loading them with explosives and blasting the rock to enable excavation down to a lower level. Blasting first takes place from a surface not previously impacted by drilling and blasting, called natural surface or terrain blasting, and any blasting below this initial bench is termed bench blasting. As shown in Figure 8, the key difference between the terrain and bench blasts, is that the bench blast is impacted by the subdrill from the previous bench (Su^*). The absence of subdrill in the natural surface blast will generally result in coarser fragmentation in the top portion, or stemming region of the first blast, compared to the bench blast.

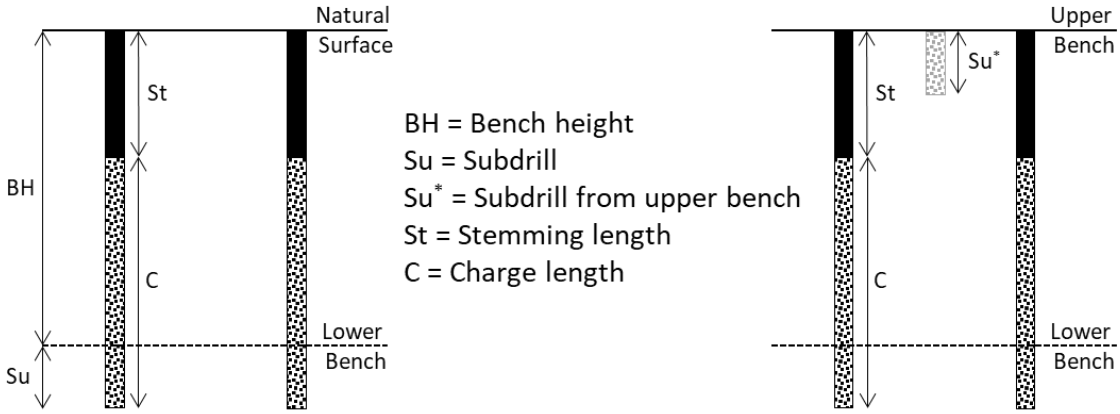

Figure 8: Natural surface blast on the left compared to a bench blast on the right.

The impact of the subdrill from the upper bench can be somewhat quantified using the formula shown in Equation 1, which is used to calculate the vertical distribution of the explosives in the bench. However, the formula does not include the hole diameter or explosive properties in the calculation, therefore it is not suitable for all cases.

$$\mathbf{VD} = (\boldsymbol{BH} - \boldsymbol{St} + \boldsymbol{Su}^{*})/\mathbf{BH} \qquad \textbf{Equation 1}$$

Where,

VD = vertical distribution of the explosives in the bench (%)
BH = bench height (m) (ft)
St = stemming length (m) (ft)
Su^* = subdrill length from upper bench (m) (ft), where present

Powder factor is the amount of explosive per unit of rock, which is either a volumetric or a mass measurement, as shown in Equation 2. When positive subdrill is used, the amount of explosives contained in the subdrill is included calculation but the unit of rock surrounding the subdrill is not.

$$\mathbf{PF} = \mathbf{M_C}/\mathbf{R} \qquad \textbf{Equation 2}$$

Where,

PF = powder factor (kg/m³ or kg/tonne) (lb/yd³ or lb/ton)
M_C = amount of explosives (kg) (lb)
R = unit of rock (m³ or tonne) (yd³ or ton)

If one considers the natural surface and bench blast examples in Figure 8, the powder factor calculated for each example using Equation 2 would be the same, which is not strictly correct. In fact, the bench blast has a higher powder factor than the natural surface blast due to the amount of explosives in the subdrill from the upper bench. If the blast design parameters used in the upper and lower bench's is the same, the powder factor of the lower bench can be calculated by including the explosives contained within the subdrill from the bench above. However, if the burden and spacing of the pattern in the upper bench is different to the lower bench the powder factor cannot be calculated using Equation 2. Moreover, if the diameter of the hole or the explosives are different between benches, standard formula are not suitable.

New Energy Distribution Method

The new method enables the powder factor, and energy factor, to be calculated for the full vertical horizon of a blast. The method involves calculating the powder factor or energy factor per unit of bench height, i.e. metre or foot, through the bench height, and takes into consideration all factors of the design and the explosive properties; the method has been termed linear powder or energy factor.

An example of the calculation comparing two different scenarios is shown in Figure 9. The powder factor is shown through the full bench height for two scenarios, on the left the two patterns are identical in the upper and lower benches, whereas on the right, the upper pattern has a smaller hole diameter, burden and spacing, so the powder factor is 3% lower.

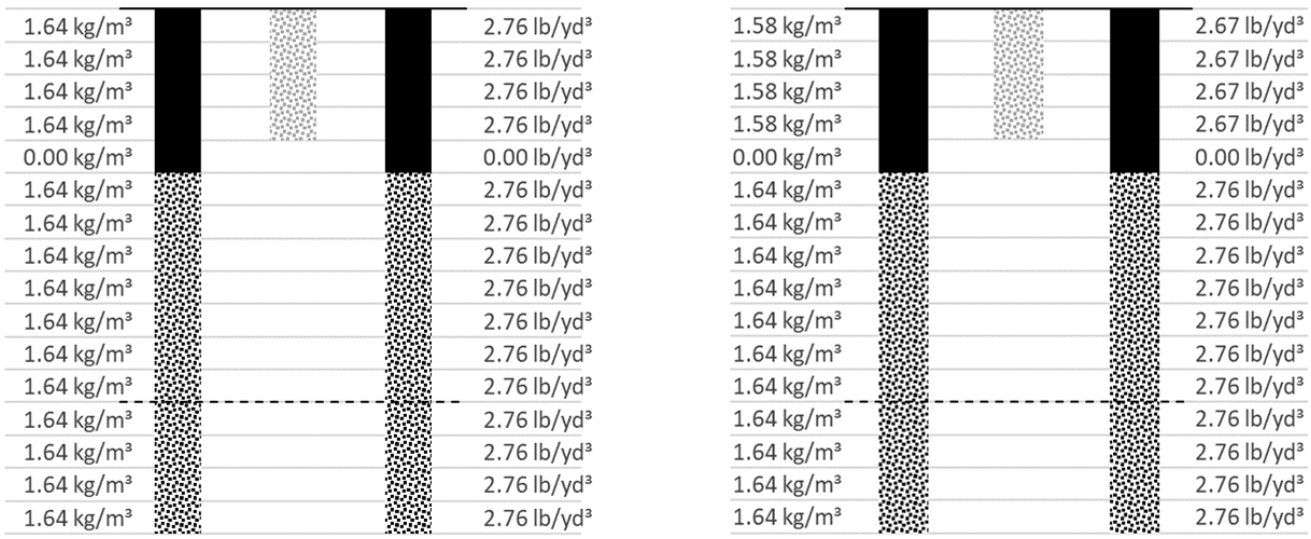

Figure 9: Linear powder factor calculation for two scenarios, on the left the pattern parameters in the upper and lower benches are identical and on the right, the upper pattern is a small diameter and smaller pattern.

The new method has been used at three open cut operations to optimise the length of subdrill to deliver more consistent energy and blast outcomes in the stemming region of the bench below. At one particular operation, which uses 115 mm (4½”) and 165 mm (6½”) diameter holes, blasting transitions from 5 m (16.4 ft) bench heights using 115 mm (4½”) diameter holes to 10 m (32.8 ft) bench heights using 165 mm (6½”) diameter holes. Fragmentation in the top of the first 10 m (32.8 ft) bench was always coarser than when blasting a 165 mm (6½”) hole diameter design above another 165 mm (6½”) hole diameter design. Calculation of the linear powder factor for each scenario shows that the shorter subdrill length of the 115 mm (4½”) hole diameter design was resulting in a 47% lower powder factor in the stemming region of the bench below, compared to when a 165 mm (6½”) hole diameter blast was fired above another 165 mm (6½”) hole diameter blast (Figure 10).

By increasing the subdrill in the 115 mm (4½”) hole diameter design, the powder factor in the stemming region of the bench below was matched (Figure 11), and the blasting outcomes became more consistent.

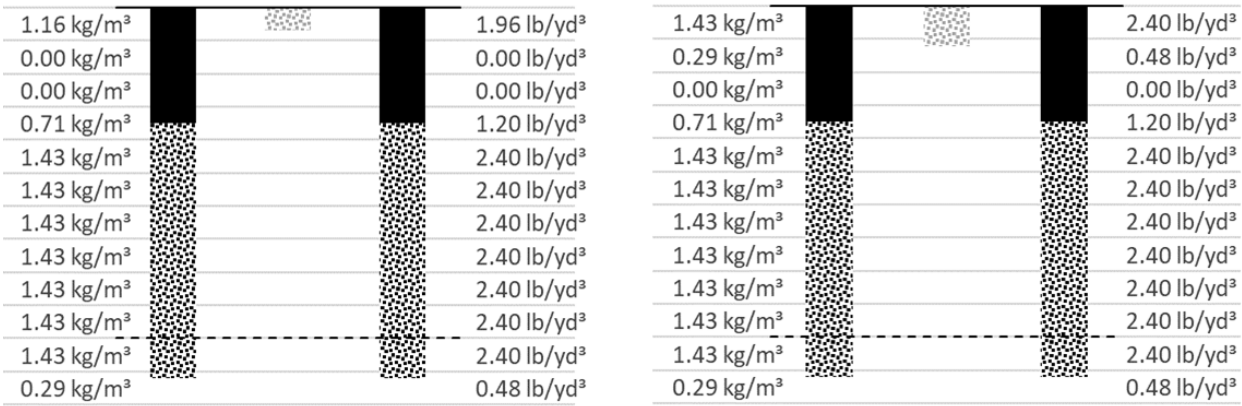

Figure 10: Linear powder factor calculations, on the left a 115 mm (4½") hole diameter blast is above a 165 mm (6½") hole diameter blast, and on the right, a 165 mm (6½") hole diameter blast is above another 165 mm (6½") hole diameter blast.

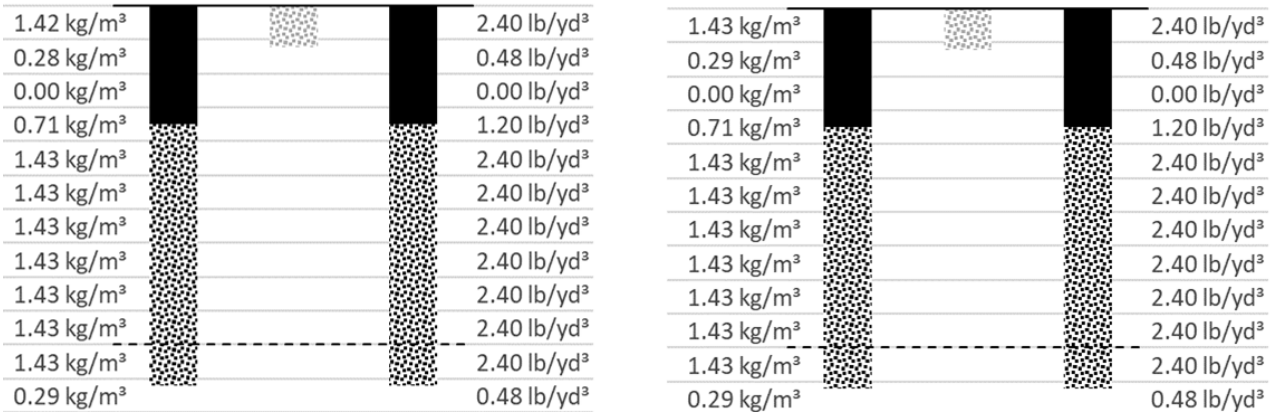

Figure 11: Optimised 115 mm (4½") hole diameter design above a 165 mm (6½") hole diameter blast, on the left, compared to the original 165 mm (6½") hole diameter blast is above another 165 mm (6½") diameter blast, on the right; note the comparable linear powder factor in the stemming region of both designs.

Conclusions

Long subdrill has proven to be an effective solution to deliver finer fragmentation in the stemming region of large diameter blast designs by creating a zone of preconditioned rock in the bench. So long as the operational challenges associated with the technique are managed, such as drilling through wet, broken ground and controlling the accuracy of pit floors, the technique is easier to apply in the field than more complicated designs.

The invention of a device to support the rock in the preconditioned zone and minimise the backfilling of drill cuttings into the hole has been instrumental in realising the full and consistent benefits of the technique.

By including the energy contributed by the subdrill from the upper bench blast in a linear powder factor calculation, the energy in a bench can be more accurately quantified and blast designs can be optimised to achieve more consistent ROM fragmentation. Further development of the linear powder factor calculation will include the impact of hydrostatic pressure on explosive density and fragmentation prediction.

Acknowledgements

The author would like to thank the technical and operational teams at the Sentinel Mine in Zambia, both past and present, for their continued hard work and dedication. Special thanks to the team from TBS Mining Solutions for always helping to find practical solutions for challenging problems.

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